SECTION 13

DRYING

BY

T. K. Sherwood, Sc.D., Professor of Chemical Engineering, Massachusetts Institute of Technology; Member, American Chemical Society, American Institute of Chemical Engineers.

CONTENTS

I. THE DRYING OF SOLIDS

II. THE DRYING OF GASES

DRYING

REFERENCES: Walker, Lewis, McAdams, and Gilliland, "Principles of Chemical Engineering," 3d ed., McGraw-Hill, 1937. Hirsch, "Die Trockentechnik," Springer, Berlin, 1927. Tiemann, "The Kiln Drying of Lumber," Lippincott, Philadelphia, 1917. Sherwood, *Ind. Eng. Chem.*, **21**, 976 (1929); **25**, 311 (1933); **26**, 1096 (1934). Sherwood, *Trans. Am. Inst. Chem. Engrs.*, **32**, 150 (1936). Ceaglske and Hougen, *Trans. Am. Inst. Chem. Engrs.*, **33**, 283 (1937).

I. THE DRYING OF SOLIDS

Drying commonly refers to the removal of a liquid from a solid, the liquid most frequently being water. The removal of water from a solid in solution is termed **evaporation**, but there is no sharp distinction between evaporation and drying in crystallization processes. Ordinarily, drying refers to the removal of water from a solid when the water is presented in relatively small amount.

Drying may be accomplished by various means, such as pressing, centrifuging, absorption, or vaporization of the water content. The paper machine combines pressing, absorption, and vaporization by the use of the press rolls, the felt, and the cylinders, or can dryers. In most cases vaporization of the water is the most expensive method of drying, and therefore as much water as possible is removed by pressing, centrifuging, or other mechanical methods before drying by vaporization is employed. For example, in the utilization of waste wood bark as fuel, the material is first pressed as dry as possible before entering the dryer proper.

Because of the relatively great importance in practice of air drying, *i.e.*, vaporization of the water into air or other carrier gas, the discussion which follows will be limited to this method of the drying of solids.

THE MECHANISM OF DRYING OF SOLIDS[1]

In the initial stages of the drying of a very wet solid under constant drying conditions, the surface is completely wet with water and the drying process is similar to the evaporation of water from a free liquid surface. As long as the surface is wholly wet the rate of evaporation is not a function of the water content of the solid, and, under constant drying conditions, the rate of drying continues constant. This stage is termed the **constant rate period.** However, at some definite water content the rate of drying begins to decrease, and the range from there to dryness is called the **falling rate period.** The water content of the solid at the end of the constant rate period and the beginning of the falling rate period is termed the **critical water content.** When dried for a very long time, the water content of the solid approaches an ultimate value which depends primarily on the relative humidity of the air and is termed the **equilibrium water content.**

The Constant Rate Period

During the constant rate period, the evaporation takes place at the surface of the wet solid, the rate of drying being limited by the rate of diffusion of water vapor through the surface air film out into the main body of the air. The solid assumes a constant equilibrium temperature, just as a free liquid surface assumes the wet-bulb temperature of the air. When the heat neces-

---

[1] Lewis, *Ind. Eng. Chem.*, **13**, 427 (1921). Sherwood, *Ind. Eng. Chem.*, **21**, 12 (1929); **21**, 976 (1929); **22**, 132 (1930); **24**, 307, (1932).

sary for vaporization is supplied only by conduction through the same surface air film through which the vapor diffuses, the surface assumes the wet-bulb temperature of the air. Where heat is supplied in other ways, such as by radiation, or by conduction from adjoining dry surfaces of the solid, the surface temperature is higher than the wet-bulb temperature, and the rate of drying is increased. An initial adjustment period is sometimes of importance, during which the wet material is warmed or cooled to the equilibrium temperature which is to prevail in the constant rate period.

Heat Conduction through Dry Surfaces. Heat flow from air or from a hot solid to adjoining dry surfaces and thence by conduction through the solid to the wet surfaces may frequently be an unsuspected but very important factor affecting the rate of drying, particularly in the constant rate period. In the case of a block or slab of material, for example, one or more faces of which are covered or otherwise waterproofed, the heat inflow through these edges increases the temperature of evaporation of the water and so increases the rate of drying.

Thus in the case of a small slab of whiting paste having covered edges and a ratio of dry edge to wetted face area of 1.6, the rate of drying in the constant rate period was found to be 70 per cent greater than that for a thin slab of the same material where the dry edge area was negligible [Sherwood, *Ind. Eng. Chem.* **21**, 976 (1929)].

Radiation from Surroundings. Radiant heat from surroundings warmer than the wet-bulb temperature of the air has a similar effect in increasing the solid temperature and consequently increasing the rate of drying. Radiation is important in practical drying problems because, in a heated dryer, material subject to radiation from steam coils or from other heating mediums is found to dry appreciably faster than where surrounded by other wet material. Furthermore, radiant heat may be used to increase the rate of drying and consequently to improve the capacity of a given dryer.

The rate of drying as pounds per hour, from a wet surface of A sq. ft., may be expressed approximately by the equation

$$\frac{dw}{d\theta} = KA(H_s - H_a) \tag{1}$$

where H_a is the absolute humidity of the air, H_s is the saturated humidity at the temperature of the surface of the solid (both expressed as pounds of water per pound of bone-dry air), and K is a constant. By utilizing the relation $h_c/K = s$ [Lewis, *Trans. A.S.M.E.*, **44**, 325 (1922); although this relation is now known to have no theoretical basis,[1] it is found experimentally to hold reasonably well for water vapor in air] and making a heat balance, one obtains

$$\frac{r_s}{s}(H_s - H_a) = (T_a - T_s) + \frac{pc}{h_c}(T_r^4 - T_s^4) \tag{2}$$

where r_s = latent heat of vaporization, B.t.u. per lb., at the surface temperature.

s = the humid heat of the wet air, B.t.u./(°F.) (lb. dry air) = $C_a + C_w H_a$.

where C_a = specific heat of dry air.

C_w = specific heat of water vapor.

T_a = temperature of air, °F. abs.

T_s = temperature solid surface, °F. abs.

---

[1] *Mech. Eng.*, **55**, 567 (1933).

T_r = temperature of surroundings, °F. abs.
p = black-body coefficient of the solid surface (see pp. 1002 to 1029).
c = radiation constant = 1.72×10^{-9}.
h_c = surface coefficient of heat flow, B.t.u./(hr.)(sq. ft.)(°F.).

Since the rate of drying is proportional to the product $h_c(H_s - H_a)$, the above equation, combined with the relation between H_s and T_s as given by the humidity chart (see below), may be used to predict the relative drying rates when the material is subject to various radiation effects.

Example 1. Consider the effect of radiation on the rate of drying of a thin sheet of wet material in air at 104°F., having a humidity H_a of 0.017. In the absence of radiation, the sheet will assume the wet-bulb temperature of 80.6°F. Now assume that the wet material is surrounded by a radiant heater the surface of which is at 374°F. (834°F. abs.) and let $p = 0.9$ and $h_c = 4.4$. B.t.u./(hr.) (sq. ft.) (°F.). The latent heat r_s will be in the vicinity of 1060 B.t.u. per lb. The above equation may then be written:

$$\frac{1060}{0.246}(H_s - 0.017) = (564 - T_s) + \frac{0.9 \times 1.72 \times 10^{-9}}{4.4}(834^4 - T^4)$$

which may be solved by referring to the saturation curve on the humidity chart for the relation between T_s and H_s. The solution is $H_s = 0.048$, and $T_s = 564$; *i.e.*, the sheet temperature has been increased to that of the air. The rate of drying has been increased to $(0.048 - 0.017)/(0.022 - 0.017) = 6.2$ times the former rate when the sheet received no heat by radiation.

Air Velocity. Since the rate of drying during the constant rate period is controlled by the rate at which vapor can diffuse through the surface air film, factors which affect the thickness of that film influence the rate of drying in this period. Perhaps the most important of such factors is the velocity of the air past the surface; the air velocity has a similar influence on the rate of drying as it has on the surface coefficient of heat flow from solid to gas in air heaters and in similar equipment.

Data pertaining to the rate of vaporization of water from flat surfaces are given by Hinchley and Himus [*Trans. Inst. Chem. Engrs.* (*London*), **2,** 57 (1924); *Chemistry and Industry*, **43,** 840 (1924)], Carrier [*Ind. Eng. Chem.*, **13,** 432 (1921)], Lurie and Michailoff [*Ind. Eng. Chem.*, **28,** 345 (1936)], and Shepherd, Hadlock, and Brewer, [*Ind. Eng. Chem.*, **30,** 388 (1938)]. These results may be represented approximately by the equation

$$G = 0.021(u\rho)^{0.8} \tag{3}$$

for flow of air parallel to the water surface. Here G represents the rate of vaporization as pounds per hour per square foot per millimeter Hg difference between the vapor pressure of water at the surface temperature and the partial pressure of water vapor in the air; ρ is the air density as pounds per cubic foot; and u is the air velocity past the wet surface as feet per second. This relation holds approximately for air rates between 3 and 20 ft. per sec. flowing parallel to the wet surface. When the surface is at the wet-bulb temperature, the driving force in millimeters Hg is almost exactly one-half the wet-bulb depression in degrees centigrade.

Data on vaporization with air flow *normal* to the surface are given by Carrier [*Ind. Eng. Chem.*, **13,** 432 (1921)] and by Molstad, Farevaag, and Farrell [*Ind. Eng. Chem.*, **30,** 1131 (1938)].

For data on vaporization of water from wet surfaces of various shapes, see the recent data of Powell, Inst. Chem. Engrs. (London), March 8, 1940.

Effect of Humidity. For practical purposes, the rate of drying may be assumed to be proportional to either the humidity difference $H_s - H_a$, as defined above, or the partial-pressure difference $p_s - p_a$, although these two quantities are not exactly proportional (p_s is the vapor pressure of the water at the surface temperature, and p_a is the partial pressure of water vapor in the air, *i.e.*, the vapor pressure of water at the dewpoint). Thus the rate of drying in the constant rate period may be accurately controlled, when desired, by controlling the air humidity. It should be noted that in the so-called constant rate period the rate of drying remains constant only as long as the drying conditions are constant; in the ordinary dryer the humidity of the air changes as the air passes over the material being dried, and the rate of drying changes progressively, in a countercurrent dryer increasing as the material becomes drier.

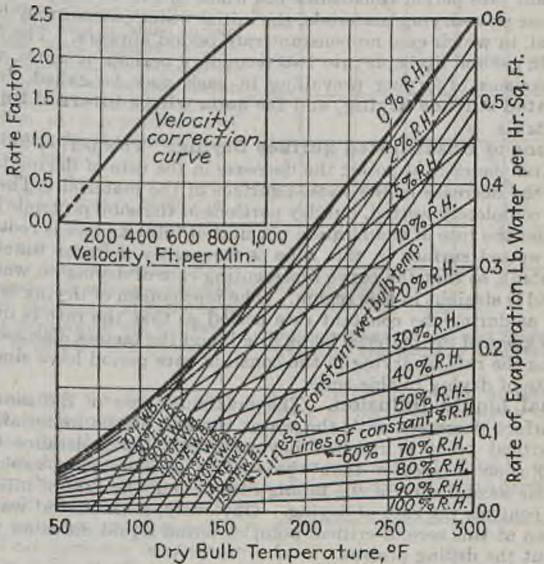


FIG. 1.—Rate of drying in the constant rate period. [*Shepherd, Hadlock, and Brewer, Ind. Eng. Chem.*, **30**, 388 (1938).]

Estimation of Drying Rate in Constant Rate Period. In the constant rate period the surface of the solid is completely wet, and the rate of vaporization is essentially the same as from a free water surface at the same temperature. Where it is possible to neglect heat inflow from adjoining dry edges and by radiation, the surface may be assumed to be at the wet-bulb temperature of the air, and the rate of drying is calculated by Eq. (3). Where radiation and conduction are appreciable, the rate will be greater than that calculated in this way. The methods of calculation used to obtain the probable surface temperature in such cases are outlined by Shepherd, Hadlock, and Brewer [*Ind. Eng. Chem.*, **30**, 388 (1938)]. The assumption that the surface will be at the wet-bulb temperature introduces a factor of safety in that the predicted rate will be low.

Data on vaporization from wet solids collected by Shepherd, Hadlock, and Brewer [*Ind. Eng. Chem.*, **30**, 388 (1938)] have been used by them as a basis for Fig. 1, which may be used to estimate drying rates in the constant rate period. Like Eq. (3), it applies only for vaporization at the wet-bulb temperature.

The Falling Rate Period

As pointed out above, the constant rate period ends at the critical water content, and the range from there to dryness is called the **falling rate period.** In case the critical water content is less than the desired final water content, the constant rate period constitutes the whole of the drying process. In the case of some slow-drying materials, the initial water content may be less than the critical, in which case no constant rate period appears. The falling rate period is in general divisible into two secondary periods or zones, which from the mechanisms of drying prevailing in each may be called the **zone of unsaturated surface drying,** and the **zone where internal liquid diffusion controls.**

The Zone of Unsaturated Surface Drying. This period follows immediately after the critical point; the decrease in the rate of drying in this zone is due to the decrease in the wetted surface of the material. The surface is no longer completely wetted, but dry portions of the solid protrude into the air film, so that the rate of evaporation per unit of total surface is reduced. The effective wetted surface in this zone is frequently a linear function of the water content, so that the curve representing rate of drying *vs.* water content of the solid is straight in this region. The mechanism of drying is essentially the same as during the constant rate period so that the rate is independent of the thickness of the material being dried, and the factors discussed above as influencing the rate of drying in the constant rate period have similar effects on the rate of drying in this zone.

Internal Liquid Diffusion. The maximum rate of diffusion of water to the surface decreases with the water content of the material, so that a second critical point is reached beyond which the resistance to internal liquid diffusion is greater than the surface resistance to vapor removal. During this second zone of the falling rate period, the rate of internal liquid diffusion controls the rate of drying. Obviously, if the initial water content is less than at this second critical point, internal liquid diffusion will control throughout the drying process.

The diffusion of liquids through many solids obeys the same fundamental diffusion laws as hold in the case of the diffusion of heat, and the Fourier equations of heat conduction hold for the drying of solids when internal liquid diffusion controls. For the case of a **slab** of thickness $2R$, the relation between water content and time has been derived and expressed in an infinite series for E in terms of τ, where E is the ratio of free (total minus equilibrium) moisture at time θ to the initial free moisture, and τ represents the group $K\theta/R^2$, K being the diffusion constant of water through the solid, and R is one-half the slab thickness. The values of E and τ as given by this theoretical relation, are as follows:

τ	0.02	0.05	0.10	0.15	0.20	0.30	0.50	1.00
E	0.84	0.75	0.642	0.563	0.496	0.387	0.236	0.069

This relation is of value in the analysis and extrapolation (preferably done graphically) of drying data over the range where internal liquid diffusion controls. The corresponding relations for various **other shapes** are given by Newman [*Trans. Am. Inst. Chem. Engrs.*, **27**, 310 (1931)].

Example 2. In order to compare experimental data with the theoretical curve, a special plotting paper may be constructed, using a uniform τ scale, but so changing the E scale as to force the theoretical relation to be a straight line. With this special paper, since the τ scale is uniform, data following the theoretical relation will fall on a straight line when plotted as E *vs.* τ, as E *vs.* θ/R^2, or as E *vs.* θ By comparison of the location of such a straight line with that of the theoretical line, the value of K may be calculated directly.

Figure 2 shows data obtained on two slabs of poplar wood respectively 1.27 cm. and 1.90 cm. thick, plotted as E *vs.* θ/R^2, using the special plotting paper described. Arbitrarily taking the point for comparison at $E = 0.60$, where $\theta/R^2 = 3.72$, and the theoretical value of τ is 0.126, we have

$$\tau = 0.126 = \frac{K\theta}{R^2} = 3.72 \times 3600 \times K$$

whence $K = 9.4 \times 10^{-6}$ in c.g.s. units (grams of water diffusing per second per square centimeter per unit concentration gradient, per centimeter thickness, where the concentrations are expressed as grams of water per cubic centimeter of wet wood).

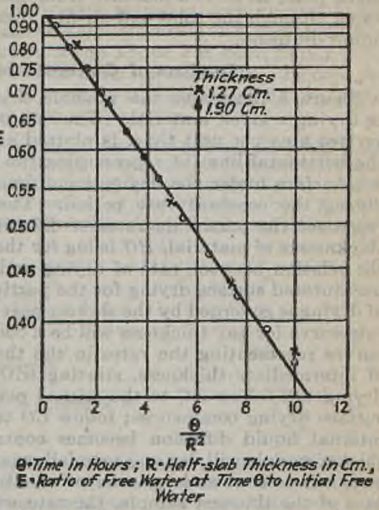


FIG. 2.—Use of special ordinate scale in plotting drying data illustrated by data on drying poplar wood slabs.

It is important to notice that under these conditions the time of drying varies directly as the square of the slab thickness, whereas, when surface evaporation controls, the time of drying varies directly as the first power of the thickness. Furthermore, *when internal liquid diffusion controls, air velocity has no influence on the rate of drying, and air humidity is of importance only so far as it affects the equilibrium water content.* Radiation and conduction of heat from the surroundings have little effect, except in raising the temperature of the solid and so improving the diffusion constant of liquid through the material.

Internal Evaporation. Since during the second zone of the falling rate period the rate of arrival of water to the surface is less than the rate at which evaporation could take place at the surface, the water in the solid near the surface tends to be depleted, and the plane or locus of evaporation tends to retreat from the surface.

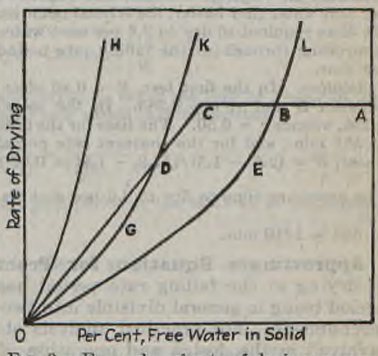


FIG. 3.—Examples of rate-of-drying curves.

This probably takes place only in porous or fibrous solids, such as pulp; and not in colloidal materials, such as clay. When evaporation occurs within the solid, the vapor formed must diffuse, not only through the surface air film, but through the relatively dry surface layer of solid. The actual evaporation takes place at such a distance from the surface that the rate of diffusion of vapor through the solid and air-film resistances is equal to the rate of internal liquid diffusion.

Outline of General Mechanism of Drying

Figure 3 illustrates the possible drying curves which may be followed in drying a given material. The rate of drying, as weight of water per unit surface area per unit time, is plotted against the water content of the solid, the horizontal line *CA* representing the rate of evaporation from the wholly wet surface under the drying conditions prevailing, *i.e.*, the rate of drying during the constant rate period. The three curves *HO*, *KGO*, and *LEO* represent the maximum rates of diffusion of liquid to the surface for three thicknesses of material, *HO* being for the thinnest. The line *ODC* represents the relation between rate of drying and water content for the conditions of unsaturated surface drying for the particular material. Then, since the rate of drying is governed by the slowest part of the diffusion process, the resulting rate curve for any thickness will be a composite curve made up of the **lowest** curves representing the rates in the three different zones. For the sample of intermediate thickness, starting with a water content at *A*, the rate of drying will follow *AC* to the critical point *C*, where the zone of unsaturated surface drying commences; follow *CO* to the second critical point *D*, where internal liquid diffusion becomes controlling, and then along *DGO*. The thicker sample will dry at a rate following *AC* to *B*, after which internal diffusion will control and no zone of unsaturated drying will appear. For the case of the thinnest sample, the rate will follow *AC* to the critical point *C*, thence along *CO* to dryness, the material being so thin that at no point does the rate of internal diffusion control. In this case the falling rate period consists only of the zone of unsaturated surface drying.

Example 3. A clay slab, dried at a sufficiently high air velocity that internal liquid diffusion is controlling, loses 60 per cent of its free water content in 320 min. Using a moderate air velocity, the same slab requires 650 min. to dry from 27.6 per cent to 5.2 per cent water (dry basis), the critical point occurring at 17.2 per cent water. Calculate the time required to dry to 2.6 per cent water, assuming internal liquid diffusion to be controlling throughout the falling rate period. The equilibrium water content is 1.5 per cent.

Solution. In the first test, $E = 0.40$ after 320 min. From the theoretical relation between E and τ, $\tau = 0.285$. In the second case, $E = (5.2 - 1.5)/(17.2 - 1.5) = 0.236$, whence $\tau = 0.50$. The time for the falling rate period is therefore $(0.50/0.285)320 = 561$ min., and for the constant rate period $650 - 561 = 89$ min. At 2.6 per cent water, $E = (2.6 - 1.5)/(17.2 - 1.5) = 0.07$ and the corresponding value of τ is 1.00. The necessary time to dry to 2.6 per cent water concentration is therefore $89 + \frac{1.00}{0.285} \times 320 = 1210$ min.

Approximate Equation for Practical Problems. The mechanism of drying in the falling rate period has been seen to be complicated, this period being in general divisible into two zones involving two different drying mechanisms. For practical analysis of drying data, and for computations involved in the design and operation of commercial drying equipment, the empirical equation

$$\log \frac{(W_c - W_E)}{(W - W_E)} = k\theta \tag{4}$$

may be used. W is the water content (dry basis) at any time θ after the start of the falling rate period, W_E is the equilibrium water content, W_c the critical water content, and k is a constant. This equation is based on the assumption that the rate of drying in the falling rate period is proportional to the free water content $W - W_E$. Although this assumption is only an approximation to the facts, the resulting equation given above has been found to fit the data well for the drying of a number of different materials. In case the initial water content W_0 is less than the critical, W_c is replaced by W_0, and θ refers to the total time from the start of the drying.

Example 4. A certain material is dried to 10 per cent water in 5 hr., the initial moisture content being 20 per cent, and the equilibrium or hygroscopic moisture content is 5 per cent. All moisture contents given are expressed as per cent of the dry weight. Assuming the solid to exhibit no constant rate period (*i.e.*, the critical is greater than the initial moisture content), estimate the time required to dry to 8 per cent water under the same drying conditions.

Solution. Substituting in Eq. (4),

$$\log \frac{(20 - 5)}{(10 - 5)} = k \times 5$$

whence $k = 0.0955$.

For the new conditions, the equation becomes

$$\log \frac{(20 - 5)}{(8 - 5)} = 0.0955 \times \theta$$

whence the time required $= \theta = 7.31$ hr.

Example 5. A very wet solid is dried from 36 to 8 per cent water under constant drying conditions, the time required being 5 hr. How long would be required to dry to 5.5 per cent water under the same drying conditions? The critical moisture is 14 per cent and the equilibrium moisture 4 per cent (all water contents expressed on the dry basis).

Solution. The assumption that the rate of drying is directly proportional to the free water content may be expressed

$$\frac{dW}{d\theta} = -k(W - 4)$$

On integration this becomes

$$\log_e \frac{(W_c - W_E)}{(W - W_E)} = k\theta_F = 2.3 \log_{10} \frac{(14 - 4)}{(W - 4)}$$

which is assumed to apply to the falling rate period, θ_F being the time of drying in the falling rate period. The time in the constant rate period may be written

$$\theta_c = \frac{(W_0 - W_c)}{k(W_c - W_E)} = \frac{(36 - 14)}{k(14 - 4)}$$

since the rate of drying during the constant rate period is the same as at the critical point, given by the first equation as

$$\frac{dW}{d\theta}\text{(constant rate period)} = -k(W_c - 4) = -k(14 - 4)$$

By adding the time for the two periods,

$$\text{Total time} = 5 = \theta_F + \theta_c = \left[\left(\frac{2.3}{k}\right) \log \frac{(14 - 4)}{(8 - 4)}\right] + \frac{(36 - 14)}{k(14 - 4)}$$

whence $k = 0.624$, $\theta_F = 1.47$, and $\theta_c = 3.53$.

To dry to 5.5 per cent water, $\theta_c = 3.53$.

$$\theta_F = \left(\frac{2.3}{0.624}\right) \log \frac{(14 - 4)}{(5.5 - 4)} = 3.03 \text{ hr.}$$

Total time = 3.53 + 3.03 = 6.56 hr.

Surface Checking and Case-hardening

Surface cracks and checking are caused by unequal shrinkage of the surface and interior of the material being dried. Although at the start of the drying, the moisture content may be uniform throughout the solid, appreciable moisture gradients are soon set up from center to surface, the difference between center and surface moisture concentrations probably being the greatest when internal liquid diffusion first becomes controlling. In the case of a material which shrinks as it dries, the surface tends to shrink before the interior, and unless the material is sufficiently elastic to take up the resultant strains either permanent distortion or surface cracking results. When these effects must be avoided, controlled humidity drying is employed, using air of sufficiently high humidity to decrease the rate of drying so that the moisture gradients set up in the material will not be great enough to cause sufficient differential shrinkage to cause cracking. Many materials shrink during drying through only a limited range of moisture contents, and controlled-humidity drying needs to be employed in only one part of the dryer.

In some cases, as the material becomes drier, shrinkage or other physical change takes place which causes an appreciable decrease in the diffusion constant of liquid through solid, causing the so-called "case-hardening" effect. As the surface layer dries, it becomes relatively impervious to the diffusion of the remaining water, which is trapped in the interior of the solid. This effect is noticeable in the drying of soap. Since it is caused by excessive difference between surface and interior moisture concentrations, it may sometimes be obviated, as suggested above, by controlled-humidity drying. For example, should the case-hardening develop only when the surface concentration becomes very low, air of relatively high humidity may be used while the first 50 to 75 per cent of the water is being removed, so that the case-hardening effect develops only in the latter portion of the drying process. In this way, the total drying time may frequently be reduced.

Laboratory Drying Tests

Since the use of forced air currents, and in general the fixing of the drying conditions throughout the dryer are governed by the nature of the material and its drying characteristics, it is important to have reliable test data on the manner in which the material to be dealt with dries. Such data may be obtained in simple laboratory drying cabinets, even without humidity control, and it is recommended that the results be studied by plotting the rate of drying *vs.* the water content, as in Fig. 3. A plot of free water content *vs.* time for the falling rate period, using a semilogarithmic coordinate scale, as suggested by Eq. (4), should also be constructed.

Differences in air conditions between plant operation and laboratory test can usually be allowed for, but special care should be taken to use a test sample having the same ratio of dry to wetted surface, and subjected to similar radiation effects as in the plant dryer. A common mistake is to use a small test sample and to accentuate the effect of heat conduction from dry surfaces; furthermore, a single test sample may receive radiation from steam coils or other source of heat in the drying cabinet which appreciably increases its rate of drying, whereas in the large-scale dryer most of the material may be surrounded by other similar material and not subject to radiant heat. These precautions are particularly important because errors made are in the unsafe direction: the tests tend to indicate a greater rate of drying than is actually attained under plant conditions.

Much can be learned regarding the general mechanism of the drying of sheet materials by two simple tests. These are (*a*) the drying of one and two thicknesses of the material in still air under constant drying conditions; and (*b*) the drying of a single thickness of the material in a blast of air having a sufficient velocity to evaporate the water from the surface at a greater rate than it can diffuse through the sheet to the surface. In both tests drying from the edges of the samples should be eliminated by tinfoil or other waterproof covering.

In test *a*, the samples should be suspended in a laboratory cabinet maintained at constant temperature and constant humidity. They are then weighed in place at frequent intervals, using a balance resting on the top of the cabinet, with a wire attached to one scale pan passing down into the cabinet and on which the sample to be weighed may be hung. The weighings must be made quite accurately, since the rate of drying is calculated from the difference between two successive weights. The samples are finally dried thoroughly, the tare determined, and the rate of drying (as weight of water per unit time per unit face area) plotted *vs.* percentage moisture (dry basis). The double-thickness sample may be made by sewing together two single-thickness sheets. This is tested similarly and the data obtained are plotted as rate of drying *vs.* moisture content on the same plot as the data for the single thickness. If surface evaporation is controlling throughout the drying process, the curves should coincide except for a slight effect of heat flow in through the dry edges. If, however, internal liquid diffusion is controlling during the falling rate period, the rate for the single sheet will be twice that for the double sheet during this period. If the samples exhibit a constant rate-of-drying period, the critical water content will be apparent from this plot. The data may also be plotted, as free water *vs.* time, on semilogarithmic coordinate paper, using only the data on the falling rate period.

Test *b* serves to place the curve representing the rate of diffusion of water to the surface. The sample is held in a stream of air flowing at 30 ft. per sec. (or more) and weighed frequently, at short intervals of time. For example, the small sample may be held in the exhaust from a small centrifugal blower, operating at room temperature. The rate of drying is again calculated and plotted *vs.* percentage moisture, in this case giving a curve concave to the rate axis. By comparison with the theoretical-diffusion equations, the diffusion constant of water through the sheet may be found. A comparison of this curve with the rate curves obtained from test *a* indicates the drying range where there is danger of internal diffusion becoming controlling and also indicates how much it may be possible to increase the rate of drying by employing high air velocity and low humidity.

CLASSIFICATION OF DRYERS

Dryers may be classified as **continuous or batch, adiabatic or constant temperature,** and also as to the manner in which the heat necessary for vaporization of the water is transferred to the material.

In the common air dryers the latent heat of vaporization is supplied to the stock by convection from the heated air, and such dryers may be termed "hot-air dryers." In the drying of thin sheets of veneered wood, the heat is supplied by contact with heated metal plates between which the wood is pressed. Similarly, in the can or cylinder dryers for textiles, pulp, and other sheet material, heat is received by conduction from a heated metal drum, the interior of which is supplied with steam. In such dryers, instead of the stock picking up heat from the air, the air may be colder than the sheet, and heat flow takes place in the reverse direction, *i.e.*, from stock to air. The wet material being dried may also receive heat by radiation, although this method of heat transfer is usually found to occur only in high-temperature dryers and incinerators. As pointed out above, radiation from the surroundings raises the temperature of the wet solid, which increases the rate of drying. Since the vapor pressure of the water in the solid increases rapidly with temperature, a high rate of radiant heat flow causes only a moderate increase in the temperature of the stock, and it would appear that this means of increasing the rate of drying, especially in the constant rate period, has not received the attention it merits.

Hot-air dryers may be classified as adiabatic and internally heated, the latter usually operated at constant temperature. In the adiabatic dryer, all the heat supplied comes in as sensible heat in the entering air. The air is preheated and in passing through the dryer cools as it gives up its sensible-heat content which is utilized in vaporizing water from the stock. Thus the temperature is lowest where the air leaves and highest where the air enters, so that where the flow of air and material is countercurrent, the driest stock comes in contact with the hottest air. In cases where the material being dried is sensitive to heat when dry, this is an important disadvantage of the adiabatic dryer. However, the wet-bulb temperature of the air remains approximately constant, so that such dryers are particularly well adapted to the drying of materials sensitive to heat when wet, *e.g.*, gelatine. As the air passes through an adiabatic dryer it cools, and its capacity for carrying water vapor is greatly diminished. The air carrying the water vapor away from the dryer has therefore a relatively low water-vapor-carrying capacity, and consequently large amounts of air are required, resulting in low heat efficiency. When steam coils or other sources of heat are within the dryer, enough heat is usually supplied to accomplish the vaporization of water from the material being dried, and the temperature of the air remains approximately constant, although it may be allowed to decrease or even caused to increase as it passes through the dryer. Frequently air is circulated through sections of the dryer and over steam coils either in the bottom or just outside the dryer. Part of the air so circulated may be passed from one section to the next, each section operating as a small cross-current adiabatic dryer. The dryer as a whole, however, usually operates at a constant temperature, although by removing air from each section various combinations of drying conditions may be maintained.

Intermittent dryers are charged with the material to be dried, which remains within the dryer until dry, when it is removed and the dryer recharged. Such dryers are usually of the **loft, compartment,** or **chamber** type and are widely used for small-scale and experimental work. They are cheaply and simply built and have the important advantage that when properly controlled they are able to operate on relatively complicated humidity and temperature schedules for the air in the dryer. However, close control and inspection must be maintained or the product from successive charges will not be uniformly dried. Continuous dryers have the important advantages inherent in continuous operation at any stage in a manufacturing plant. The continuous operation fits in more satisfactorily with the other steps in the manufacturing process, labor costs are reduced, less storage space for charge and product is required, and the dried product is apt to be more uniform. Perhaps the most important advantage of the continuous dryer over the intermittent dryer is the greater heat efficiency of the former, the saving in cost of heat being usually more than enough to offset the greater fixed charges on the continuous dryer.

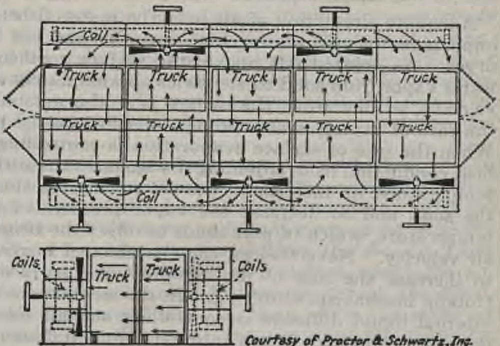
AIR CIRCULATION AND RECIRCULATION

Adequate circulation of air over the material being dried is of the utmost importance in maintaining high capacity and low heat consumption of the dryer. As pointed out above, the surface coefficient of vapor diffusion of water vapor from solid to air varies approximately as the 0.8 power of the air velocity, so that when the surface is at the wet-bulb temperature of the air the rate of drying is also proportional to the 0.8 power of the air velocity. When the rate of surface evaporation is controlling, but due to radiation or heat conduction from adjoining dry surfaces the surface is above the wet-bulb temperature, an increase in the air velocity at the same time tends to cool the solid and so decrease the vapor pressure of the water at the surface temperature, which in part tends to offset the beneficial effects of the higher air velocity. Nevertheless, the net effect of increased air velocity is always to increase the rate of drying, providing surface evaporation is the controlling mechanism of drying. In the second zone of the falling rate period, internal liquid diffusion is controlling *and air velocity can have no effect on the rate of drying*. When internal liquid diffusion is controlling, which in some cases may be throughout the whole of the drying process, it is consequently useless to attempt to cut down the time of drying by using a high air velocity past the surface of the material. Good air circulation within the dryer not only increases the surface coefficient of vapor diffusion but serves to eliminate stagnant-air pockets where the air humidity is high and very little drying takes place.

There are three general ways of attaining the advantages of high air velocity *over the surface of the material*. These are: (*a*) the use of large quantities of heated fresh air, (*b*) the use of a mixed feed of heated fresh air and hot waste air from the dryer, and (*c*) the repeated circulation of the air across or through the solid material as it passes through the dryer. The use of large quantities of heated fresh air accomplishes the desired purpose but involves an excessively large heat loss in the waste hot air. Circulation within the dryer or recirculation of waste hot air is therefore the method used, the choice depending on the arrangement and construction of the dryer. The repeated circulation of the air over and across the material as it passes through the dryer is usually to be preferred, as each pass of the air over the material involves a shorter length of air travel and lower friction and power consumption by the fans than when waste air is returned by ducts and recirculated through the whole of the dryer.

Figure 4 shows a plan view of a two-track tunnel dryer illustrating the principle of cross circulation within the dryer. The dryer, shown, houses ten trucks on which are piled trays holding the material being dried. Fans placed at the sides in alternate sections circulate the air over the trays on the trucks and over the steam coils set behind the fans. Because of the large cross section and short length of air travel, the back pressure on the fans is very low, and long-blade slow-speed fans are used, capable of circulating large amounts of air. Fresh air displaces sufficient hot humid air to carry away the water evaporated, but as it is possible to remove a minimum amount of nearly saturated air, the heat efficiency of the dryer may be maintained very high. At the same time the capacity will be high because of rapid drying caused by the high velocity of the air over the material.

In the case of the cross-circulation dryer just described, fresh air is supplied at one end of the tunnel and the hot humid waste air removed at the other end, the air usually passing countercurrent to the material. Under these conditions the humidity of the air in the dryer varies from that of the fresh air at the air-feed end to that of the humid waste air at the other end of the dryer. However, when the system of recirculating part of the waste air and mixing it with fresh air is employed, the humidity gradient through the dryer is from that of the mixed feed at one end to that of the waste at the other end; when using similar amounts of fresh air, the average humidity in the dryer will therefore be greater than when recirculation within the dryer is employed. This may be of no great importance when drying at 150° to 200°F., but at low temperatures the humidity driving force is ordinarily so small that the increased average humidity of the air has an appreciable effect on the rate of drying.



Courtesy of Proctor & Schwartz, Inc.

FIG. 4.—Two-track tunnel dryer showing cross circulation within the dryer.

In all cases where the air is to pass vertically over wet sheets of the material suspended within the dryer, the direction of air movement should be **downward,** since the air density increases as the air cools.

Condensation on the Walls of the Dryer. When recirculation of air through the whole of the dryer, or repeated circulation within the dryer, is used, the heat consumption may be maintained very low by withdrawing only the minimum amount of nearly saturated air. However, if the humidity of the air becomes greater than the humidity of air saturated at the temperature of the inner surface of the housing wall, *i.e.*, if the inner surface of the wall of the dryer is below the dewpoint of the humid air, condensation of moisture will take place. Provision may be made for carrying away water condensed in the waste-air flues, but condensation within the dryer itself must usually be avoided, because of corrosion of the dryer or spoilage of the material by drip from the roof. The degree of saturation of the waste humid air is therefore frequently limited by the liability of condensation of water within the dryer.

The inner-surface temperature of the housing may be estimated by the method of calculation described in Sec. 7 on the Flow of Heat. For example, suppose the dryer wall to be of wood $\frac{5}{8}$ in. thick, having a thermal conductivity of 0.05 B.t.u./(hr.)(sq. ft.)(°F. per ft.), and that the inner- and outer-air temperatures are 170° and 70°F., respectively. Assume that the surface coefficient of convection plus radiation (Sec. 7) for both inner and outer surfaces be 2.0 B.t.u./(hr.)(sq. ft.)(°F.), then the total resistance to heat flow through the wall is

$$R = \frac{1}{2.0} + \frac{0.625}{(0.05 \times 12)} + \frac{1}{2.0} = 2.04$$

Since the temperature drop is proportional to resistance, the drop through the inner-surface air film is $\frac{0.5(170 - 70)}{2.04} = 24.5°$, and the inner-surface temperature is $170° - 24.5° = 145.5°$F. The maximum safe dewpoint of the waste air is consequently 145.5°F., and the corresponding maximum absolute humidity of the waste air is 0.181 lb. water per pound of bone-dry air.

HEAT CONSUMPTION OF DRYERS

The cost of heat is an item of major importance in nearly every dryer, and in many cases constitutes well over half the total cost of operation. The

Table 1. Heat Losses through Dryer Housing Walls*

Values given are over-all coefficients of heat transmission expressed as B.t.u./(hr.)(sq. ft.) (°F. difference in temperature between inside surface and outside air)

4-in. Brick wall	0.77
4-in. Concrete wall	0.94
⅞-in. T. & G. sheathing	0.73
⅞-in. T. & G. sheathing and sheet steel	0.72
Sheet steel, ¾-in. air cell, sheet steel	0.63
Sheet steel, 1-in. improved Asbestocel sheets, sheet steel	0.44
Sheet steel, 1½-in. improved Asbestocel sheets, sheet steel	0.32
Sheet steel, 2-in. improved Asbestocel sheets, sheet steel	0.25
¼-in. Transite 2-in. improved Asbestocel sheets, sheet steel	0.25
⅞-in. T. & G. sheathing, builder's paper, 4-in. air space, builder's paper, ¼-in. Transite on 2 × 4-in. studs	0.38
Type *A*, Special Johns-Manville Special Built-Up	0.11
Type *B*, Special Johns-Manville Special Built-Up	0.22
Type *C*, Special Johns-Manville Special 2-in. panel construction	0.24
Transite encased insulating board, ½ in.	0.55
Transite encased insulating board, 1 in.	0.32
Johns-Manville Standard lumber dry-kiln housing	0.069
Oven Walls:	
Sheet steel, 6-in. 85% magnesia, ¼-in. Transite (temp. diff. 450°F.)	0.084
Sheet steel, 6-in. 85% magnesia, ¼-in. Transite (temp. diff. 350°F.)	0.082
Sheet steel, 6-in. 85% magnesia, ¼-in. Transite (temp. diff. 250°F.)	0.080
Sheet steel, 4-in. 85% magnesia, ¼-in. Transite (temp. diff. 450°F.)	0.124
Sheet steel, 4-in. 85% magnesia, ¼-in. Transite (temp. diff. 350°F.)	0.121
Sheet steel, 4-in. 85% magnesia, ¼-in. Transite (temp. diff. 250°F.)	0.117
Sheet steel, 2-in. 85% magnesia, ¼-in. Transite (temp. diff. 450°F.)	0.234
Sheet steel, 2-in. 85% magnesia, ¼-in. Transite (temp. diff. 350°F.)	0.228
Sheet steel, 2-in. 85% magnesia, ¼-in. Transite (temp. diff. 250°F.)	0.221

* Courtesy Johns-Manville Corporation.

total heat consumption may be divided as follows: (*a*) heat required to evaporate water, (*b*) the heat to heat the dry solid from the feed temperature to the discharge temperature, (*c*) the heat required to heat trucks, trays, or parts of the dryer itself if the operation is intermittent, (*d*) the heat in the waste humid air as sensible heat above the fresh-air temperature, and (*e*) the heat lost to the surroundings by conduction through the dryer housing.

When the initial and final moisture contents of the material are low, the heat lost as sensible heat in the dried product may be quite appreciable. Although it is true that this quantity could be reduced by operating the dryer at a lower temperature, the drying temperature is frequently fixed by the nature of the material and is not subject to variation. Furthermore, operation at a lower temperature usually results in an increased heat loss as sensible heat in the waste air, and no net saving is accomplished. The heat requirements under heading *a* are therefore not subject to control by the operator and may be termed the "theoretical" heat requirements of the dryer. Since this quantity is commonly between 1000 and 1100 B.t.u. per lb. water evaporated, the "theoretical" heat requirement, using low-temperature steam, is in the vicinity of 1.1 lb. steam per pound of water evaporated.

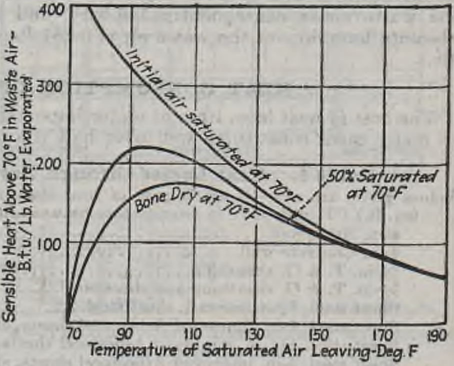


Fig. 5.—Sensible-heat loss in waste air as a function of the waste-air temperature.

The dryer housing may be constructed of wood, brick, or insulating boards of various types, the nature of the material and the thickness of the walls depending on the temperature maintained within the dryer. In estimating the heat loss from a proposed dryer housing, it is necessary to have a knowledge of the over-all coefficient of heat transfer through the dryer walls.

Table 1 gives values of this coefficient H, as B.t.u./(hr.)(sq. ft.)(°F.), for a number of typical constructions. The coefficient H is the reciprocal of the sum of the several resistances through which the heat must flow in series, *viz.*, the inner and outer air-film resistances, and the individual sections of the wall. These resistances may be calculated and the over-all coefficient H may be then estimated by the methods described in Sec. 7. For example, the over-all resistance of a $\frac{5}{8}$-in. wood partition was found on p. 1493 to be 2.04. The value of H is therefore 0.49 B.t.u./(hr.) (sq. ft.) (°F.), from which the rate of heat loss from the dryer may be calculated, knowing the inner and outer air temperatures.

The housings should be tight to minimize air leakage, either in or out. "Through metal," *i.e.*, bolts or screws extending through the wall, increases the heat loss to a surprising extent and should be eliminated where possible.

The third important heat quantity is represented by the loss as sensible heat in the waste humid air. A common misconception in this connection is that the lower the temperature of the air leaving, the less will be the sensible-heat loss. For a given percentage saturation, the absolute humidity increases faster with temperature than does the temperature rise above the fresh-air temperature. Thus, the sensible-heat loss, which is directly proportional to the temperature rise and inversely proportional to the increase in humidity, decreases as the final air temperature is increased. It is true, however, that considerable drying may be accomplished without appreciable change in temperature of the air passing through the dryer; under such conditions the sensible-heat loss is zero. Consequently for a definite percentage saturation of the waste air, the sensible-heat loss may be seen to go through a maximum. This point is illustrated in Fig. 5, which shows the sensible-heat loss per pound of water evaporated plotted *vs.* the temperature of the waste air, for various conditions of the fresh air entering at 70°F. Even in the case of the bone-dry fresh air, the maximum is seen to occur at a lower temperature than would ordinarily be employed, so that in general it is true that for a given percentage saturation of the waste air, the higher its temperature, the lower will be the sensible-heat loss.

The degree of saturation of the waste air is frequently taken as a criterion of the amount of heat lost as sensible heat in the waste air. This is justifiable, for the curve of sensible-heat loss *vs.* temperature at a constant percentage saturation of the waste air is seen to be fairly flat, so that the heat loss is mainly dependent on the degree of saturation. At a given temperature, the sensible-heat loss varies approximately inversely as the percentage saturation, approaching a minimum as the humidity approaches saturation.

It is important that the sensible-heat loss be calculated above the outdoor air temperature, even in cases where the air supply for the dryer is obtained immediately from a heated room. It is evident that the dryer should be charged with heating the fresh air from the outside temperature, even though some of the heating be done by the ventilating system, or by radiators in the room.

ECONOMIZERS

As pointed out above, the sensible-heat loss in the waste humid air represents a considerable fraction of the total heat requirements of the dryer, and the transfer of this heat to the incoming cold fresh air offers a possible means of reducing the heat consumption of the drying operation. Furthermore, when the dewpoint of the waste humid air is high, and the temperature of the fresh air is low, considerable condensation of water vapor may be effected in a suitable heat exchanger and add greatly to the heat so recovered. However, the over-all coefficients of heat transfer from gas to gas are normally so low that the required heat-exchange surface of a suitable type of economizer is excessively large, and the use of such equipment is found practical only where the temperature of the gases leaving is relatively high, or where the dewpoint is so high that considerable recovery of heat by condensation of water vapor is possible.

Adams and Cooper [*Ind. Eng. Chem.*, **22**, 127 (1930)] report six tests on a 10,500-sq. ft. copper-surface cross-flow Briner-type economizer operating in connection with a 60-ton sulfite pulp dryer. The fresh-air temperature varied from 13° to 74°F., the waste air from the dryer from 99° to 128°F., and the amount of waste air handled from 203,000 to 236,000 lb. (water-free basis) per hr. The sensible heat removed from the waste air varied from 400,000 to 1,030,000 B.t.u. per hr., and the heat recovered by condensation of water vapor varied from 2,470,000 B.t.u. when the fresh air was at 13°F. to 80,000 B.t.u. per hr. when the fresh air was at 74°F. The corresponding temperatures of the entering air after passing through the economizer were 71°F. when the fresh air was at 13°, and 101° when the fresh air entered at 74°F. It may be seen that the amount of condensation varied in such a way as to maintain the air feed to the dryer between 71° and 101°F., although the fresh-air temperature varied over a 61° range, *i.e.*, from 13° to 74°F. In these tests, the total heat in the waste air averaged about three times that necessary to heat the fresh air to the waste-air temperature; and the heat actually picked up by the fresh air was about half that necessary to heat it to the waste-air temperature. The actual heat recovered was consequently from 10 to 20 per cent of the total heat consumption of the dryer.

By the use of an absorber to remove water vapor from the waste air it is possible to operate with total recirculation. This accomplishes somewhat the same result as an economizer and also provides for humidity control when the outside fresh air is very humid. Direct-contact absorption in a strong aqueous solution of lithium chloride reduces the humidity to a low value without cooling the air, and a double or single effect evaporator is provided to reconcentrate the diluted solution. Excellent heat economy is possible with such a system, although many of the installations have been to facilitate low-temperature drying of leather and similar products during summer periods of high humidity [see Weisselberg, *Chem. & Met. Eng.*, **45** 418–421 (1938); Bichowsky, *ibid.*, **47**, 302 (1940)].

TYPES OF DRYERS

Commercial drying equipment is necessarily extremely varied in design because of the fact that materials dried in industry vary widely in physical form and in both physical and chemical properties. A brief description of only the commoner and more generally applicable types is given in the following section. Operating and performance data are included where they are available.

Spray Dryers for Liquids. [Stewart, *Chem. & Met. Eng.*, **35**, 471 (1928). Reavel, *J. Soc. Chem. Ind.*, **46**, 925, 951 (1927). Power, *Chem. Eng. Mining Rev.*, **20**, 201 (1928). Bowen, *Ind. Eng. Chem.*, **30**, 1001 (1938). Fogler and Kleinschmidt, *Ind. Eng. Chem.*, **30**, 1372 (1938).] Liquids may be dried by spraying into heated air, the large surface of contact between the drops and the air making possible a very high rate of vaporization. When the dry material is not sensitive to heat, the solution may be superheated and sprayed into a rotary dryer or heated enclosure and the dried product collected in the form of a powder. When a finely ground product is desired, it is thus possible to accomplish both drying and pulverizing in one operation. The process may also be used to produce a homogeneous mixture from a solution or suspension of two or more substances.

Efficient countercurrent action is not ordinarily possible in the spray process, so that the sensible-heat loss in the waste air is usually high, and the use of the method is confined to those cases where the short time of heating is particularly desirable. The spray process is of advantage in the drying of the following materials: easily oxidized substances, such as leuco-base dyes; easily dissociated substances; easily damaged ferments; substances with delicate flavors; substances physically or chemically sensitive to heat; organic or inorganic colloidal solutions; suspensions; and suspensions in solutions. It is used in drying blood products, eggs, gelatine, soap, sugar, or beverage powders, blackstrap molasses, and its use is said to be the only possible way of obtaining dried glucose as a powder. Because of its relatively poor heat economy, the process is not usually recommended for the drying of cheap materials but is of particular importance in the food and beverage industries and for the drying of fruit juices to produce soluble powders.

The important steps in the spray-drying process are the atomization of the liquid, the drying process, and the recovery of the product. Several types of sprays may be used, (1) a centrifugal bowl or disk atomizer, (2) a pressure-type nozzle, or (3) a two-fluid or atomizing nozzle. In general the first two form hollow spherical particles and the third forms irregular dense particles, but any type may produce a product of varying form and density depending on atomizer design and operation and the physical properties and moisture content of the feed. The centrifugal type of atomizer is the most common since it yields a product of more nearly uniform particle size and has the advantages of high capacity, low maintenance, accessibility for cleaning, low liquid pressure requirements, and little tendency to clog. The disk is driven by an electric motor or steam turbine at speeds normally between 3000 and 12,000 r.p.m. The liquid is fed to the disk and thrown out horizontally by centrifugal force, breaking up into a spray as it leaves the edge of the disk. The pressure-type nozzle, usually of the spin-chamber type, operates under pressures of 100 to 2,000 lb. per sq. in. It does not produce as uniform a particle size as the centrifugal type, has low capacity, and tends to plug and to erode with use. In the two-fluid atomizing-nozzle liquid under low velocity is delivered to the nozzle and there atomized by an impinging stream of air at pressures up to 100 lb. per sq. in. It has most of the advantages of the centrifugal disk except for non-uniform particle size. The fineness and uniformity of the spray are of utmost importance because the capacity of the dryer is governed by the time required to dry the largest of the drops formed. The largest drops tend to dry incompletely and to form wet lumps in the product, making it necessary to reduce the rate of liquid feed. If the largest drops are satisfactorily dried, the smaller drops will also be well dried; if the spray is uniform and the smaller drops are all nearly as large as the largest, the capacity of the dryer will obviously be greater than when the average size of the drops is only a fraction of that of the largest.

The spray is located at either the top or bottom of a large cylindrical drying chamber into which is fed heated air or hot products of combustion. Usually the chamber has a height approximately equal to the diameter, although with pressure sprays the height may exceed the diameter appreciably. The drying air or gases may range from 200° to 1200°F. and usually pass through the dryer in parallel flow to the product. In the Kestner spray dryer the hot gas enters tangentially at the top, thus being given a centrifugal motion which it retains in passing down through the drying chamber. The spray is located at the top and revolves in the same direction as the rotation of the gas in the chamber; the spray and gas are therefore thrown together with a parallel motion with the object of preventing the formation of uncertain eddy currents which might allow short-circuiting of spray or hot gas and so hinder the efficient operation of the dryer. A small amount of cold air is passed vertically down over the spray to prevent drying until the sheet of liquid formed by the revolving disk has completely broken up into a fine spray.

Various methods are employed for the recovery of the product. Usually a portion which collects on the floor of the drying chamber is swept by rakes into hoppers or conveyors or blown out by air. The portion which is carried out in the discharged gases is collected in a cyclone which may be supplemented by a bag filter or scrubbing tower. The feed liquor is sometimes used to scrub the gases, thus effecting at the same time a concentration of this liquid by the heat available in the gases.

Capacities of spray chambers vary widely with the properties of the liquid, the degree of atomization, the temperature of the gases, and the size of chamber. Generally the capacities vary from 0.1 to 3.0 lb. water evaporated per hour per cubic foot, the low value being for 20- to 50-mesh particles at low gas temperatures and the high value for 300-mesh particles at gas temperatures of 800° to 1200°F. Drying chambers vary from 10 (for particles below 350 mesh) to 30 ft. in diameter. They must be large enough so that none of the spray will hit the walls when wet. Gas temperatures vary from 200° to 1200°F., and gas velocities are usually 50 to 120 ft. per min. based on the average gas temperature.

Thermal efficiencies range from 70 per cent for high gas temperatures to 20 per cent for low air temperatures (300°F. or less). Fuel requirements vary from 2000 to 5000 B.t.u. per lb. water evaporated. Labor for operating a dryer, exclusive of materials handling, is 12 to 24 man-hr. per day. Maintenance is 5 to 10 per cent of the total installed cost. A 20-ft. diameter dryer requires about 50 hp. The installed cost of a spray dryer exclusive of buildings will be from $10 per cubic foot for large chambers to between $25 and $60 per cubic foot for small chambers.

Spray dryers are applicable to large production since they do not perform well in sizes below 5 ft. diameter. Space requirements and investment cost are high. However, this method of drying produces a product which needs no subsequent grinding or blending and generally has peculiar, valuable properties of low bulk density, uniform particle size, free-flowing and non-caking tendencies.

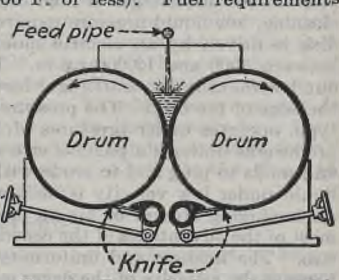


FIG. 6.—Twin-roll drum dryer with center feed. (*Courtesy Buffalo Foundry and Machine Co.*)

Drum Dryers for Liquids. Another common type of continuous dryer for liquids is the drum dryer, consisting of one or two steam-heated metal cylinders revolving around a horizontal axis. Liquid is applied in a thin film to the cylinder surface and dried by heat transferred from the condensing steam in the interior. The dried product is removed, after traveling as far around the drum as possible, by a straight "doctor" knife set close to the drum surface. The dried material drops into a chute or hopper and the water vapors are generally removed by natural or forced draft through a hood over the dryer. Drum dryers have been employed commercially in the drying of milk, soap and detergents, dyes, fruit and vegetable extracts, glue, cereals, salt solutions, etc.

There are three general types of drum dryers: single-, double-, and twin-drum dryers. The single consists of one cylinder; the double and twin, two cylinders set close together and parallel, mounted on the same frame, and driven by one motor. The latter two differ only in the rotation of the rolls. In the double the rolls rotate toward each other at the top, and in the twin they rotate away from each other at the top (see Fig. 6). The latter is particularly adaptable to drying salt solutions where crystals may form. All three types may be operated under vacuum on temperature-sensitive materials or on those from which valuable solutions are being evaporated.

A special type of drum dryer adapted to the predrying of pastes is a finned single-drum dryer. The drum surface is grooved around the circumference so that the heating surface is materially increased. Pastes are forced into the grooves by rolls and removed by finger scrapers. The final product, generally not completely dry, is dried further in an appropriate air dryer, such as the horizontal conveying screen dryer (Aeroform dryer).

Various methods of feeding drum dryers are employed. In the "pan" feed the rolls dip into the liquid; in the "spray" or "splash" feed liquid is thrown against the bottom surface of the roll; and in the "slurry" feed it is recirculated through pipes past the roll surface. These methods are chiefly used with single-drum dryers. With double- and twin-drum dryers the liquid is usually fed between the rolls to the upper trough, which is fitted with endboards. This type of feed is sometimes applied to single-drum dryers by the use of a close-fitting frame.

For the successful operation of a drum dryer it is important that the liquid feed be evenly distributed over the surface of the roll so as to give a film of uniform thickness and concentration. For this reason doctor rolls are often used with twin- and single-drum dryers and constant level controls where trough or pan feeds are employed. It is also important to remove as much of the dried product as possible so that a thin film will not remain on the roll. The doctor knife must be close fitting and sharp. Frequent honings are desirable, and, where the drum surface is likely to wear irregularly and form grooves, a flexible knife is preferred.

The capacity of drum dryers is proportional to the dryer width and diameter and to the speed of rotation. The speed of rotation determines the time available to dry the film of liquid picked up. The time required depends not only on the amount of water in the film but on its thickness. In other words, for a given feed consistency a thick film of liquid not only contains more water than does a thin film but its rate of drying is reduced because of the greater resistance to flow of heat and vapor through it. However, the speed of rotation also determines the thickness of the film, the slower the speed generally the thicker the film. Usually, a greater capacity is obtainable when using a given feed by drying thin films at a fairly high roll speed, especially if a very dry product is desired. With very thin films and high roll speeds the time of contact of the material with the hot roll is greatly reduced, which is an important advantage in drying liquids sensitive to heat. Usual speeds of rotation are 4 to 12 r.p.m.

The over-all heat-transfer coefficient depends on the individual resistances of the steam film, the metal wall, and the material film and has been found to vary in practice from 10 to 50 B.t.u./(hr.)(sq. ft.)(°F. temperature difference). The material film is generally controlling, although inefficient removal of non-condensable gases and condensate inside the roll may increase the resistance of the steam film appreciably. The usual capacity varies from 2 to 8 lb. of water evaporated per hour per square foot of roll surface. This capacity is largely dependent on the steam pressure and hence on the over-all temperature difference.

Drum dryers are supplied in a variety of standard sizes. Single-drum dryers range from 2 ft. diameter by 1 ft. 8 in. long to 6 ft. diameter by 12 ft. long. Double- and twin-drum dryers range from 2 ft. diameter by 2 ft. long to 5 ft. diameter by 12 ft. long.

The total installed cost of a drum dryer is about twice the purchase cost. Steam consumption varies from 1.3 to 2.0 lb. steam per pound water evaporated. Maintenance costs are about 5 to 10 per cent of the installed dryer cost.

Drum dryers produce a flaked or powdery product which needs no further grinding or blending. They are difficult to operate, however, where a hard, thin coating builds up and adheres on the roll surface so that it cannot be removed easily by the doctor knives.

For a further discussion of drum dryers, see Van Marle [*Ind. Eng. Chem.* **30**, 1006 (1938)] and Harcourt [*Chem. & Met. Eng.* **45**, 179 (1938)].

Pan Dryers. Pan or agitator dryers are particularly adapted to the intermittent drying of small batches of solids, *e.g.*, grains, flour, coal, ores, dyes, salts, and flotation concentrates. They are operated either under atmospheric pressure or under vacuum. The use of agitation generally shortens the drying time over that obtained either in atmospheric or in vacuum tray dryers. These dryers are not suitable for materials which cake on hot metal surfaces or whose quality is degraded by breakage of the particles.

Agitator dryers are usually considered to comprise the following types: atmospheric or vacuum pan, vacuum rotary, screw conveyor, or trough. In all types the material is agitated in contact with a heated metal wall. The pan dryer (3 to 6 ft. diameter by 1 to 2 ft. deep) is in the form of a shallow circular pan with flat or bowl-shaped bottom and vertical sides. A plow or scraper agitator rotates around a vertical shaft and stirs the material during drying. Discharge of material is accomplished through an opening in the floor or side wall. The vacuum rotary dryer (1.5 to 6.5 ft. diameter by 3.5 to 36 ft. long) comprises a stationary, cylindrical steam-jacketed shell within which a set of agitator blades revolve around a horizontal axis. Vacuum is applied (as also for the vacuum pan dryer) by steam jet or vacuum pump through a suitable condenser, the line including a dust bag or appropriate dust collector. The dryer is not easily cleaned. The screw conveyor dryer (6 to 24 in. diameter by almost any length) is also a steam-jacketed cylindrical shell through which material is moved by means of a screw conveyor. Such a dryer may consist of a number of such units mounted in parallel one above the other to conserve floor space, the material discharging from the exit of one into the feed end of the one below. Only enough air is passed through the dryer to remove vapors and not dust. The trough dryer (4 ft. by 24 ft. to 12 ft. by 60 ft.) is a modification of the screw conveyor dryer in which material is moved by oscillating rabble arms through a steam-jacketed or gas-heated trough open to the atmosphere.

The drying of materials in agitator dryers is dependent on the heat transfer from the condensing steam through the dryer walls into the drying material. The main resistances are those of the material and material film along the wall. While the material contains surface moisture, its temperature will approximate the boiling point of the solvent under the absolute pressure prevailing in the dryer. As the material dries beyond this point, its temperature rises approaching the temperature of the dryer wall.

The capacity of these dryers varies with the material, being 1 to 3 lb. water evaporated per hour per square foot of heating surface for materials with high moisture content, not carried to extreme dryness, and 0.1 to 0.5 lb. water per hour per square foot for low-moisture materials carried nearly to their equilibrium moisture content. Pan dryers are generally loaded to 1 to 2 ft. depth, and vacuum rotary dryers, 50 to 60 per cent full. Agitators revolve at 2 to 8 r.p.m. Steam consumptions for pan and vacuum rotary dryers range normally from 1.3 to 1.8 lb. steam per pound water evaporated for high-moisture materials; and for screw conveyor dryers, 1.5 to 2.5 lb. steam per pound water evaporated. For low-moisture materials (near the equilibrium moisture content) thermal efficiencies will be much lower.

The total installed cost of one of these dryers will usually be between two and three times the purchase cost. Operating labor costs are appreciable since the operation is batchwise. Maintenance costs average between 5 and 10 per cent of total installed cost of dryer including vacuum equipment.

The following data are reported by Atwater and Borkland [*Chem. & Met. Eng.*, **26**, 226 (1923)] for a 6-ft. pan dryer jacketed on the bottom and sides. The dryer was operated at atmospheric pressure with a small blower exhaust to remove the vapors. In these tests the steam consumption was stated to average 1.7 lb. per lb. water evaporated; and the power used for the agitator, 0.31 hp. per 100 lb. of wet charge.

Material	Charge, lb.	% water, wet basis		Lb. water evaporated	Minutes
		Initial	Final		
Coal	325	23.6	1.0	76	127
Sawdust	378	52.2	0.3	197	310
Gypsum	606	16.2	0	98	62
Fireclay	500	39.7	2.3	191	170
Sand	585	16.0	0.2	93	76

Rotary Dryers. Rotary dryers are applicable to the continuous drying of granular, crystalline, or lumpy materials which are relatively free-flowing throughout the drying operation and which do not contain too large a proportion of dust. Materials dried in this way include inorganic crystalline salts, ores, coal, sand, sugar, oxalic acid, etc.

A rotary dryer consists essentially of a rotating cylinder, mounted on rollers and inclined to the horizontal. [Smith, *Ind. Eng. Chem.*, **30**, 993 (1938).] Material is fed in at the upper end and discharged at the lower. There are three common types distinguished by the manner in which heat is supplied for drying: (1) the direct rotary in which flue gases or hot air are passed through the dryer in direct contact with the material and either countercurrent or parallel to its flow, (2) the indirect-direct rotary, in which heat is transferred from the gases both indirectly through the metal walls and by direct contact with the material, and (3) the totally indirect where heat is transferred solely through metal walls.

In the direct rotary the rotating cylinder is equipped with flights on the inner surface which serve to lift the material and shower it down through the hot gases. For performance data on single shell dryers using hot flue gases, see Alliott [*J. Soc. Chem. Ind.*, **38**, 173*T* (1919)] and Horgan [*Trans. Inst. Chem. Engrs. (London)*, **6**, 131 (1928)]. Various designs are offered to divide the cross-sectional area into segments to reduce dusting and grinding action in large diameter dryers. This type of rotary is operated with either countercurrent or parallel flows of gases and material. Countercurrent flow gives greater thermal efficiency and is generally used but parallel flow is advantageous with materials which must not be heated to too high a temperature when dry and which must be dried to a certain final moisture content and not to complete dryness. Either steam-heated air or flue gases may be employed depending on the temperature-sensitivity of the material.

The indirect-direct rotary is presented in a number of designs. In one the dryer, located in a brick-lined chamber, is heated externally by flue gases which subsequently pass through the dryer cylinder (see Fig. 7). In another design the dryer is made up of two concentric cylinders. Hot gases pass through the center cylinder and then reverse direction to pass through the annular space between cylinders countercurrent to, and in contact with, the drying material. In another design the heated gases pass through axially arranged ducts, attached to the inner surface of the dryer cylinder, before returning through the dryer in contact with the material. The application of this type of rotary dryer is the same as that of the direct rotary, with the possible exception of abrasive materials. It is thermally about 35 per cent more efficient and requires less space than a direct rotary dryer.

The totally indirect rotary dryer is typified by the steam-tube dryer [Bill, *Ind. Eng. Chem.*, **30**, 997 (1938)] in which the drying cylinder is fitted with steam tubes running the length of the dryer along its inner surface. Other variations are possible where hot gases supply heat through the outer shell or through an inner concentric cylinder or both. This dryer type is useful where the materials will stand a high temperature but no contamination from products of combustion, *e.g.*, clay or whiting. The steam-tube rotary is useful where materials require a well-regulated temperature during drying or where the material is too dusty for use in a direct rotary.

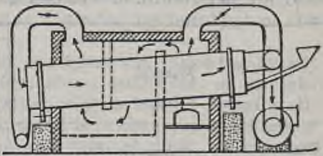


FIG. 7.—Enclosed single-shell rotary dryer.

In addition, there is the Roto-Louvre dryer, of comparatively recent development, in which hot air is blown through louvers in the wall and up through the bed of material in the rotating dryer cylinder. By a special design the air is introduced only through the louvers covered by the material. This dryer takes less space, is more expensive, but exerts less degrading action on crystal materials than a direct rotary dryer [see Erisman, *Ind. Eng. Chem.* **30**, 996 (1938)].

The rotary kiln is of similar design and operation to a direct rotary dryer, except for the absence of lifting flights and the brick lining of the inner walls. It is usually operated at a high temperature. In the kiln the drying operation is usually of secondary importance to the roasting or calcining action.

The size of a rotary dryer depends chiefly on three factors: (1) total amount of heat to be transferred per unit time, (2) permissible gas velocity through the dryer, and (3) thermal efficiency desired. A rough comparison of the relative capacities of the various types of rotary dryers is given in the following table based on equal dryer volume and similar operating temperatures:

	Relative Capacity
Direct rotary	1.0
Indirect-direct (double-shell)	1.35
Indirect (hot gases)	0.7
Indirect (steam tube)	3.0
Roto-Louvre	1.5

Air velocities through a dryer should be chosen conservatively since the quantity of material carried out varies as the second to fifth power of the velocity, depending on particle-size distribution. A conservative choice will allow for future increases in dryer capacity or changes in particle size of the feed. Air velocities vary from 50 to 100 ft. per min. for very dusty materials to as high as 900 ft. per min. for coarse materials. Dryer capacity varies approximately as the 0.67 power of the mass velocity of the air (expressed as pounds per square foot per hour). A Roto-Louvre dryer usually employs velocities of 120 to 300 ft. per min. through the material bed.

In general the highest air temperatures that the material and dryer construction will stand should be employed so as to get highest thermal efficiency. Representative efficiencies expressed as percentage of the total heat used to evaporate moisture and heat material to exit temperature are given below.

Dryer	High temp. (direct-firing), %	Low temp. (steam-heated air), %
Direct rotary	55–75	30–55
Indirect-direct (double shell)	75–85	
Totally indirect (hot gases)	50–60	
Totally indirect (steam tube)	.....	75–85
Roto-Louvre	60–80	

The operating costs of a rotary dryer are generally lower than of other types of continuous dryers. Labor required for actual dryer operation, exclusive of material handling, is about 25 per cent of one man's time. Steam consumption for a direct steam-heated air dryer is normally about 3 to 4 lb. steam per pound water evaporated. Power is required for driving the fan, rotating the dryer, and handling the feeding and discharging of the material. Empirically the horsepower for a direct rotary (for dryer drive and fan) can be expressed as about $0.5D^2$ (where D = dryer diameter, ft.). Of the total power, 20 to 60 per cent will be required for the fan, depending on air velocity and pressure drop of the system. Yearly maintenance charges will amount to 5 to 10 per cent of the total installed cost of the drying equipment.

Manufacturer's data on performance and operating costs are given in Tables 2 and 3.

In the operation of a rotary dryer the following points are worth considering: A uniform feed rate is important in securing dryer capacity and product uniformity. Air leakage should be avoided. Dryer seals should be inspected frequently and adjusted so as to keep a tight seal both from the standpoint of air leakage in and dust loss out. Sometimes two fans, one on the inlet and one on the outlet, are serviceable in regulating the air pressure within the dryer at atmospheric pressure. Control of dryer performance is best accomplished by the temperature of the discharge material. Knockers are useful in preventing the sticking of materials to the walls in the wet end of the dryer. The handling of sticky materials can sometimes be aided by the partial recirculation of dried product. Uniform air distribution entering the dryer, both as to temperature and velocity, is very important. Often oversized air heaters, installed to accommodate maximum capacity, cause unequal temperature distribution when operated at low steam demands.

"Can" or Cylinder Dryers for Sheet Materials. Materials in the form of continuous sheets are commonly dried on "can" or cylinder dryers, in which the sheet is heated by contact with revolving steam-heated cylinders, the water vapor being removed in a stream of air. The cylinders, which may be from 2 to 6 ft. in diameter and up to 20 ft. wide, are placed in two or three horizontal rows, one above the other, and the sheet passed over cylinders alternately in each row. The axes of the rotating drums are staggered, and the surfaces of successive dryers in the same row are close together, so that the

Table 2. Manufacturer's Data on Performance of Rotary Dryers*

Data	Double-shell type Direct contact with gases Countercurrent flow					Double-shell type Indirect contact with gases		Single-shell type Direct contact with gases Countercurrent flow	Single-shell type Direct contact with gases Parallel flow
	Clay	Coal	Coal	Stone	Ore	Clay	Clay	Stone	Flotation concentrates
Diameter, ft.	5	5.8	7.5	7.5	8.7	5	7.5	7.5	5
Length, ft.	30	35	55	55	65	30	55	70	35
Moisture in material fed, %	24.5	9.0	11.8	8.9	14.2	27.3	19.8	5.2	12.5
Moisture in material discharged, %	2.7	0.6	1.1	0.7	4.7	0.7	1.3	0.3	3.2
Temp., outside air, °F	52	28	65	74	40	50	70	47	38
Temp., fan exhaust, °F	140	128	168	145	165	226	221	390	355
Temp., material fed, °F	50	40	63	70	48	58	64	50	41
Temp., material discharged, °F	305	240	245	280	220	237	248	360	238
Type of fuel:	Coal	Coal	Coal	Coal	Oil	Coal	Oil	Coal	Oil
Calorific value of fuel, B.t.u. per lb. or gal.	13,000	13,200	13,900	14,000	144,000	12,405	138,000	14,000	14,100
Fuel consumed per hr., lb., or gal.	265	305	715	940	143	280	65.8	1,260	24.7
Horsepower required for dryer drum	11	16	48	53	73	12	54	56	10.5
Horsepower required for dryer fan or fans	3	4	9	9.5	16	4.5	16	11	3.5
Capacity, dried material per hr., lb.	7,256	22,000	46,000	76,400	98,500	3,915	18,500	104,000	13,100
Water evaporated per hr., lb.	2,096	2,031	5,580	7,096	10,906	1,414	4,267	5,375	1,392
Water evaporated per lb. or gal. of fuel	7.9	6.66	7.8	7.55	76.2	5.05	64.8	4.27	56.3

* Courtesy of The Hardinge Company.

arc of contact of the sheet with the drum is considerably more than a semicircle, the total length of contact being usually from 50 to 70 per cent of the total length of the sheet in the machine. In pulp and paper dryers, the sheet is frequently held in contact with the rolls by an endless felt, which at the same time absorbs moisture from the sheet and is itself dried. If, in drying paper, the drums are all the same size and driven at the same speed, the tendency of the paper to shrink as it dries stretches the sheet and may cause injury to the product or breaking of the sheet. Elaborate electric drives have been developed which adjust the speed of the rolls to allow for the decreased lineal speed of the sheet as it shrinks and so to overcome this difficulty. The framework supporting the drive and drum bearings is usually covered with a hood from which the humid waste air is exhausted to a stack or monitors on the roof. Condensation on the cool ceiling and consequent drip on to the sheet is thus prevented.

Table 3. Operating Costs for Rotary Dryers*

Data	Single-shell type, air heated	Single-shell type, direct heat	Double-shell type, indirect heat
	Ammonium sulfate	Stone	Kaolin
Original moisture, %	1.63	4.8	24
Final moisture, %	0.16	0.3	2
Source of heat	Steam coils	Oil	Coal
Tons produced per hr	3.75	21	5.5
Shell dimensions	4 × 30 ft.	5 × 30 ft.	80 in. × 50 ft.
Power, hp	12	17	42
Manufacturer's price	$4700	$3200	$16,500
Erection cost (including foundations, reducers, motors, etc.)	$1210	$1490	$2280
Cost steam or fuel, per hr	$0.354	$2.70	$0.85
Cost of power, per hr	$0.18	$0.357	$1.03
Labor charged to dryer, per hr	$0.25	$0.30	$0.55
Estimated repair cost, per hr	$0.045	$0.021	$0.044
Total operating cost, exclusive of depreciation and interest, per hr	$0.829	$3.378	$2.474
Total operating cost, exclusive of depreciation and interest, per ton of product	$0.221	$0.161	$0.45

* Manufacturer's data, courtesy The Hardinge Company.

The value of the coefficient of heat flow from steam to sheet is determined by the conditions prevailing on the inside and on the surface of the dryers. Four important causes of low coefficients are: (1) poor removal of air from the steam in the dryers, (2) poor removal of condensate, (3) the accumulation of oil or rust on the interior of the dryers, and (4) the accumulation of a fiber lint on the outer surface of the drums. One of the best ways of removing air from the steam is to fit each drum with a small pet cock which is slightly opened all the time, although this method has the disadvantage that when the machine is shut down the operators may neglect to close the pet cocks, in which case air is sucked in and fills the drum, so that on starting up again time is lost in expelling the air. Short vertical pipes dipping nearly to the bottom of the inside of the drums are usually provided to siphon off the condensate, but these pipes sometimes rust and break off, so that the drum runs half full of water. Instead of the siphon pipe small buckets attached to the end of the drum on the inside may be used, which lift the condensate as the drum revolves, depositing it in an axial outlet. One reported test (Lewis, McAdams, and Adams, *Pulp Paper Mag. Can.*, February, 1927, p. 122) on a sulfite pulp dryer, in which the actual sheet temperatures were measured, gave a value of about 33 B.t.u./(hr.) (sq. ft.) (°F.) for the over-all coefficient of heat flow from steam to sheet.

The capacity of a drum dryer may be increased appreciably by the use of air jets directed at the face of the sheet across its width. High air velocity tends to reduce the air-film thickness on the surface of the sheet and so to increase the surface coefficient of vapor diffusion. With thin paper or textile sheets, surface evaporation is controlling throughout the drying, so that the use of high air velocity is of value not only during the constant rate period but also in the falling rate period. Such air jets should be directed only at the wet sheet and not across the bare dryer surface, as this would increase the heating of the air and decrease the heat economy. The over-all average rate of drying obtained in a number of tests on paper dryers has been calculated as pounds per hour per square foot of effective dryer surface, as tabulated in Table 4. The results of a questionnaire regarding paper drying have been published by a committee of the Technical Association of the Pulp and Paper Industry, and include performance data from a number of mills on newsprint [*Tech. Assoc. Pulp Paper Ind.*, **17**, 210 (1934)], writing papers [*ibid.*, **19**, 199 (1936)], and miscellaneous papers [*Paper Trade J.*, **103**, No. 3, p. 26 (1936)]. Data on the performance of board and kraft paper dryers have been assembled by Stamm [*Tech. Assoc. Pulp Paper Ind.*, **15**, 208 (1932)] and by Montgomery [paper presented at the annual meeting of the Tech. Assoc. Pulp Paper Ind., February, 1933].

The data collected in Table 4 also indicate the steam economy which may be expected with drum dryers drying newsprint and bond paper. The heat required to heat the air from room temperature to the exhaust temperatures varied from 134 to 520 and averaged 276 B.t.u. per lb. water evaporated. However, the dryer should be charged with the heat required to heat this air, not only from room temperature, but from the outside temperature to the exhaust temperature. The outside temperatures were not reported, but assuming them to be 50°F. in each case (the tests were run between November and March), the calculated heat required to heat the air used from 50°F. to room temperature averaged 437 B.t.u. per lb. water evaporated. The total heat requirements in these tests therefore averaged about 1100 + 276 + 437 or about 1800 B.t.u. per lb. water evaporated. The steam requirement indicated is about 1.8 lb. steam per pound of water evaporated, which may be compared with 1.67 which is the average of the measured values. The heat loss increases with, but is by no means proportional to, the amount of air used because when large amounts of air are used the exhaust temperature is much lower. This is made evident by a comparison of the values of pounds of air per pound of water evaporated, with the rise in air temperature above the room temperature.

With dryers of this type most of the heat lost to the surroundings is recovered, because the air used is drawn in from the room. Practically all the heat supplied which is not actually required to vaporize water from the sheet is represented by the sensible-heat content of the waste humid air above the fresh-air temperature. The steam requirements consequently vary with the season of the year, and the data relating to steam requirements given in Table 4 tend to be high because the tests reported were made in the winter.

Table 4. Test Data on Newsprint and Bond-paper Dryers*

Data	Test								
	A	*B*	*C*	*D*	*E*	*F*	*G*	*H*	*I*
1. Paper	News	News	News	News	News	Bond	Bond	Bond	Bond
2. Dryer diameter, ft.	6	4	4	4	3	3	3	....	3
3. Number of dryers	37	34	28	30	19	20	27	....	20
4. Effective surface, sq. ft. (approx.)	5,760	3,700	2,410	3,210	1,100(?)	625	1,080	....	1,050(?)
5. Sheet speed, ft. per min.	1,000	690	623	505	286	155	100	60	
6. Absolute humidity, room air	0.0074	0.0147	0.0118	0.0108	0.0163	0.0100	0.0114	0.013	0.005
7. Absolute humidity, exhaust air	0.114	0.0536	0.080	0.0429	0.0401	0.0304	0.035	0.022	0.012
8. Room-air temp., °F.	77.1	84.1	84	75	93	82	89	87	75
9. Exhaust-air temp., °F.	132	109.3	126	105	113	113	113	100	89
10. Lb. water per lb. dry paper, feed	2.375	2.52	2.645	2.66	1,895	1.940	1.652	2.00	2.027
11. Lb. water per lb. dry paper, product	0.095	0.081	0.092	0.093	0.086	0.063	0.037	0.025	0.031
12. Lb. per hr. dry paper	6,810	5,010	3,457	3,270	1,037	606	1,017	926	1,028
13. Lb. water evaporated per hr.	15,520	12,200	8,830	8,400	1,875	1,137	1,650	1,830	2,050
14. Lb. dry air per hr. (calculated)	145,600	316,000	129,300	259,000	79,000	56,000	70,000	204,000	292,000
15. Lb. steam per hr.	29,030	25,214	11,850	12,140	2,793	1,422	2,630	2,783	5,055
16. Lb. steam per lb. water evaporated	1.87	2.07	1.34	1.45	1.49	1.27	1.59	1.52	2.46
17. Steam temp., °F.	223	250	341	230	341	233	240.5	187	252
18. Lb. dry air per lb. water evaporated (calc.)	9.4	25.7	14.7	30.9	42	49.2	42.4	111	143
19. Sensible heat of exhaust air above room temp., B.t.u. per lb. water evaporated	134	168	160	240	218	394	264	375	520
20. Sensible heat of room air above 50°F. B.t.u. per lb. water evap.	66	228	130	201	469	410	429	1,070	930
21. Average rate of drying, lb. per hr. per sq. ft. effective surface	4.83	3.3	3.66	2.61	1.7(?)	1.75	1.53	....	1.95(?)

* *Tech. Assoc. Pulp Paper Ind.*, **5**, 32 (1922); **6**, 112 (1923); **7** 182 (1924).

Slater and Warner [*Mech. Eng.*, **44**, 587 (1922)] report a test run on a newsprint dryer made when the outside-air temperature was 83°F., showing a steam consumption of 1.40 lb. steam per lb. of water evaporated, noticeably less than the average of 1.67 for the tests reported in the above table. The sheet speed was 650 ft. per min., and the average rate of drying, as pounds per hour per square foot effective surface was 2.73, which agrees with the above results.

The same authors report the following data on the power requirements for such machines, the figures in each case being per 100 ft. per min. sheet speed:

1. Modern high-speed, motor-driven machines, having four presses, 40 72-in. dryers, and operating at 700 to 1000 ft. per min., 25 to 30 hp.
2. Turbine-driven machines, rope-drive transmission, three presses, 32 60-in. dryers, running 550 to 700 ft. per min., 30 to 34 hp.
3. Engine-driven machines, Marshall drive, three presses, 32 48-in. dryers, running 350 to 600 ft. per min., 33 to 38 hp.

A type of cylinder dryer operated under vacuum is the Minton paper dryer [*Tech. Assoc. Pulp Paper Ind.*, **7**, 1 (1924)] in which the paper sheet enters an enclosure surrounding the drum dryer of the usual type through specially designed seals which permit the passage of the continuous sheet but not the passage of air.

Cabinet, Compartment, and Tunnel Dryers. Many types of dryers are fundamentally nothing but heated enclosures supplied with fresh air, sufficient humid air being exhausted to remove the water vaporized. These may be intermittent in operation, in which case the material is placed within the dryer and left until dry, or continuous, in which case the material passes continuously through the dryer on trucks or conveyors of various types. They may be self-contained or consist merely of a heated room or "loft." Various combinations are possible as to the manner in which the heat is supplied, the moisture removed, and the material loaded and unloaded. Between the obviously intermittent loft dryer and the tunnel dryer in which the material is passed continuously in a direction countercurrent to the flow of air, there are several intermediate combinations. For example, the tunnel dryer in which different sections are operated at different air temperatures and humidities may be considered as several continuous compartment dryers, operated in series.

Loft dryers were formerly widely used and are still common in many industries. They are usually poorly ventilated, and, as natural-convection currents are relied upon to provide air circulation, the material in different parts of the room is not dried at the same rate. Furthermore, the material in sight of the steam coils tends to dry much more rapidly than that in the center of the mass. For these reasons they are being replaced by self-contained dryers provided with forced-air circulation. Fine writing paper, however, is still dried hung over poles and placed in heated lofts, as the strength of the product is better than when dried on the "can" or drum dryer.

The term "compartment dryer" refers to a self-contained heated enclosure, which may or may not be provided with forced-air circulation. In small laboratory dryers the material is placed on trays which slide into the drying cabinet, where they remain until removed. In larger dryers the material on racks or cars is moved into the dryer through a large door, which is then closed. When dry, it is removed either through the same door, or through a similar door at the other side of the dryer. Granular or lumpy material is placed on flat trays, preferably with screen bottoms, which are stacked on trucks, allowing sufficient space between the trays for the air to pass freely over the material. The tray loading is from 1 to 4 lb. of wet material per square foot but varies considerably with the difficulty of removing the moisture from the interior of the solid mass. The trucks are fitted with either flanged wheels and run on tracks, or with flat swivel wheels so that they may be pushed anywhere on a factory floor. In the latter case, guide rails of channel iron are bolted to the floor in the dryer to guide the truck through. When several trucks must be moved every time one truck is placed in the dryer, they may be moved mechanically by a chain drive.

Compartment and tunnel dryers are usually heated by steam, although electricity or hot products of combustion may be used. The dryer may operate adiabatically, the fresh air being preheated and no heat being supplied within the dryer itself, or steam coils within the dryer may be used to reheat the air after passage over the material. The combination of a fresh-air preheater and steam coils within the dryer is common. The air is caused to pass successively over the coils and the material to be dried by natural convection, by mechanical circulation, by circulation induced by water sprays or air jets, or by a combination of these methods.

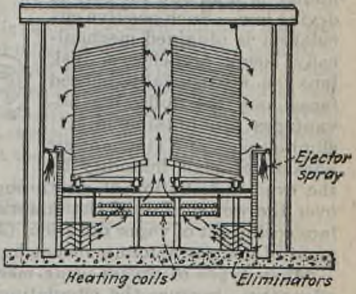


FIG. 8.—Tiemann lumber dryer.

In dryers relying on natural-convection currents, the air distribution may be improved by the use of condenser coils which serve to cool and dehumidify the air before its passage over the heating coils. One such dryer [Carrier and Stacey, *Ind. Eng. Chem.*, **13**, 438 (1921)] has condenser coils below a false bottom and steam coils attached to the walls behind a partition at the sides. The air rises over the coils through the chimney-like space between the partition and the wall, being heated as it rises. It then passes down over the material stacked in the middle of the dryer, picking up moisture and being cooled, and so down through the false bottom where it again comes in contact with the condenser coils. Moisture is there precipitated without greatly cooling the main body of the air, which then passes up the sides over the heating coils again.

The Tiemann lumber dryer (Tiemann, "The Kiln Drying of Lumber," Lippincott, 1917), shown in Fig. 8, utilizes both natural convection and induced circulation. The lumber is carried on cars on parallel tracks, being piled on the cars in such a way as to allow free air passage through the pile. Horizontal steam coils are placed beneath the floor, the air rising over these up through the space between the two trucks and thence through the pile of lumber. It is then drawn down through vertical passages at the sides by ejector water sprays, passing through entrainment eliminators before coming again to the heating coils. The humidity of the air in the dryer and consequently the rate of drying may be controlled by varying the temperature of the water used for the sprays. Canvas flaps attached to the ceiling are provided to prevent passage of air around the top, and so to ensure positive circulation of the air through the pile of lumber.

As pointed out above, air velocity and humidity have little influence on the rate of drying when internal liquid diffusion is controlling, so that with such slow-drying materials as lumber the relatively slow natural or induced circulation is permissible. However, with a very large number of materials, surface evaporation is controlling over a large portion of the moisture-concentration range, and positive circulation at relatively high velocities is necessary, not only to improve the surface coefficient of vapor diffusion but to eliminate stagnant-air pockets where the humidity is high and the rate of drying low. Such positive circulation is obtained mechanically, using propeller or disk fans or centrifugally housed fans, and the following advantages are obtainable: (1) elimination of stagnant-air pockets of high humidity by the even distribution of air throughout the dryer, (2) high velocity of air over the wet surface of the material, with consequent high value of the surface coefficient of vapor diffusion, (3) positive removal of moisture by displacement of definite amounts of hot humid air by fresh air.

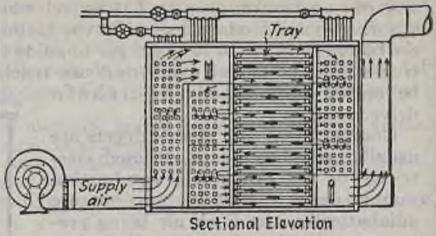
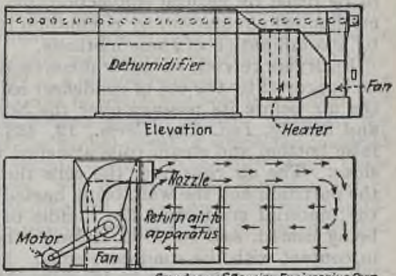


Fig. 9.—Gordon tray dryer.

Many types of dryers using mechanical circulation obtain only the third advantage, because the circulating system is used only to supply fresh air and to remove waste air from the dryer. In order to obtain the first two advantages, baffles must be provided which force the air over the material and prevent short-circuiting from inlet to outlet. A dryer particularly well designed for this purpose is the Gordon tray dryer (Carrier and Stacey, *op. cit.*), illustrated in Fig. 9. Baffles are so placed as to force the air over the trays in several passes and over steam coils which reheat the air after each pass over the material.



Courtesy of Carrier Engineering Corp.

Fig. 10.—Carrier ejector dryer.

Figure 10 shows a diagrammatic end elevation of the Carrier ejector dryer in which circulation over and through the material on trucks is obtained by the use of air jets placed near the top at one side of the dryer. Air for the ejectors is supplied by an external fan to which some of the waste air may or may not be returned. The air caused to circulate over and through the material by the ejectors is said to be from three to five times the amount of air actually handled by the fan. The diagram shows the end elevation of such a dryer, which may obviously be of any desired length, the ejectors being spaced at short intervals in the direction of the dryer length. Table 5 gives representative cost data for typical dryers of this type.

Positive air circulation in the type of dryer illustrated in Fig. 10 may also be obtained by the use of propeller or disk-wheel fans instead of air ejectors. The air is drawn through the trucks to the fan and then forced over steam coils and back over the tops of the trucks as illustrated for the ejector dryer. The fan handles all of the air circulating instead of only a fraction of it, but the pressure drop of the air in circulating within the dryer is ordinarily much less than through the recirculating ducts and the ejectors themselves in the former system. Instead of returning over the tops of the trucks and circulating continuously in the same direction, the air may be returned through the trucks at a point a little farther along the dryer, as illustrated in Fig. 4. This figure shows a modern tunnel-truck dryer holding 10 trucks, being divided into five sections by the ends of the trucks, which act as baffles, and by the location of the fans. One fan is placed in each section on alternate sides of the dryer, as shown, and the air caused to circulate back and forth over the material on the trucks and over steam coils placed between the fans and the sides of the dryer. This system not only eliminates the false ceiling compartment for the return of the air, but, by continually changing the direction of the air, provides for more even drying at the two sides of the dryer. The fan shafts protrude outside the dryer and pulleys are attached, which are driven by belts connected to shafting driven by an electric motor. Fresh air is drawn in at one end of the dryer and exhausted at the other end by means of a small blower or a stack.

Table 5. Truck-dryer Costs*

Data	*A*	*B*
Type	Ejector	Ejector
Number of trucks	2	28
Product	Pigment	Clay products
Wet weight of feed, lb. per week	13,440	1,120,560
Initial moisture, wet basis	63.8%	10%
Approx. first cost of dryer	$1,640	$13,300
Estimated weekly fixed costs, including repairs	$ 3.77	$ 30.30
Estimated weekly power costs, at 2 cts. per kw.-hr	$ 3.80	$ 64.00
Estimated weekly steam costs, at $1.25 per 1000 lb	$ 31.70	$ 315.00
Weekly attendance	5.00	10.00
Total cost per week	$ 44.27	$ 419.30
Average cost per lb. water evaporated, cts	0.52	0.38
Lb. steam per lb. water evaporated	2.95	2.25

* Manufacturer's data, courtesy the Carrier Engineering Corporation.

The **vertical turbo** dryer (Büttner system) is a continuous tray dryer, with rotating shelves, from which the material is scraped to the tray next below, and across which heated air is circulated [see Weisselberg, *Ind. Eng. Chem.*, **30**, 999 (1938)].

Continuous Screen Dryers. Continuous Screen dryers include various designs, in all of which the hot air or gas is passed vertically through a bed of material carried on a vibrating or moving screen. The horizontal screen dryer is the most common type. Material carried on a screen passes through a series of sections in which air is recirculated up or down through the material. Each section is equipped with fan and motor and heating coils in an arrangement similar to a tunnel dryer. One commercial design (D-L-O, Oliver United Filters, Inc.) employs hot gases [Irwin, *Ind. Eng. Chem.*, **30**, 1002 (1938)]. Air is drawn in at one end of the dryer and exhausted at the other, thus approximating either parallel or countercurrent flow. In the Aeroform dryer the paste is partially dried on a grooved-drum dryer and the preformed damp sticks then dried in a continuous screen-conveyor dryer. Hot air is blown through the bed of material in stick form as it moves through the dryer on the conveyor [Hurxthal, *Ind. Eng. Chem.*, **30**, 1004 (1938)].

The vibrating dryer consists of a horizontal or slightly inclined screen over which material is conveyed by the vibratory motion of the screen. Hot air or flue gas is passed vertically through the bed. In other designs the screen extends around the outer surface of a drum as with a top-feed filter, the material being fed on to the screen at the top of its revolution and hot air being drawn into the center through the bed of material. A positive rotary exhauster or high-speed centrifugal fan is employed to handle the air due to the high pressure drop through the filter bed.

Conveying screen dryers are limited to relatively free-flowing, granular, crystalline, or preformed materials. A bed of the material must be sufficiently porous to permit passage of air at reasonable velocities and pressure drops. Representative materials are cotton linters, cellulose acetate, viscose rayon, and tobacco for the horizontal conveying screen dryer; lithopone, white lead, and lead and calcium arsenates for the Aeroform dryer; coal and sugar for the vibrating screen dryer; and crystalline and granular materials which fall within narrow size limits and do not require such long drying times as for the crystal filter dryer.

The horizontal screen dryer is generally fabricated in a multiple of standard units. The approximate dimensions are as follows: 8 to 9 ft. high by 11 to 16 ft. wide by 4 to 7 ft. long (depending on the manufacturer); and the units house a screen 8 to 9 ft. wide by 4 to 6 ft. long each. Thus any desired length of screen can be obtained by arranging the units in series. The crystal filter dryer varies in size from 3 ft. diameter by 1 ft. wide to 6 ft. diam. by 4 ft. wide.

The capacity of these dryers varies with the moisture content of feed and discharge, the air temperature, air velocity, and particle size of material. The following table gives the range of capacities to be expected as well as of certain operating variables when drying granular materials:

	Capacity, lb. H_2O/(hr.) (sq. ft.) steam-heated air	Air velocity, ft./min.	Particle size range mesh	Depth of loading, in.	Pressure drop through material, in. H_2O
Horizontal conveyor screen	0.2–3.0	150–250	1–20	1–4	0.5–1
Vibrating	1–10	250–750	1–20	1–6	0.5–5
Crystal filter	5–10	100–250	20–80	0.5–3	5–30

Thermal efficiencies vary considerably within this group. The horizontal conveying screen with recirculating air will show efficiencies ranging from 50 to 70 per cent for materials with considerable moisture and as low as 15 per cent when drying to nearly complete dryness. The vibrating dryer and crystal filter dryer also show thermal efficiencies of 15 to 50 per cent.

Operating costs vary widely also. Depending on moisture content, steam consumption varies from 1.8 to 8 lb. steam per pound water evaporated. Power requirements approximate 0.1 hp. per sq. ft. for the horizontal conveying screen, 0.5 hp. per sq. ft. for the vibrating dryer, and 0.5 to 2.5 hp per sq. ft. for the crystal filter dryer. Yearly maintenance averages about 10 per cent of installed costs. Labor requirements vary depending on the time required for adjusting feed and handling the discharge material.

This group of dryers have the following advantages: material is dried more rapidly than in tray dryers; it is not subjected to mechanical agitation or tumbling; little dusting occurs; the dryers are flexible as to capacity and continuous operation; and insoluble impurities are distributed uniformly over all particle surfaces and are not concentrated over a limited surface as when air is passed over a bed of material. The dryers are, however, limited in application to definite particle size and drying characteristics; uniformity of drying is difficult to obtain unless material is dried to equilibrium dryness; and the extent of drying is more difficult to determine by any continuous measurement such as by air temperature.

Vacuum Tray Dryers. Vacuum tray dryers are employed for the batch drying of temperature-sensitive or easily oxidizable materials which would otherwise be handled in atmospheric tray dryers. Such a dryer consists of a vacuum-tight chamber constructed generally of cast iron or steel plate and equipped with hollow shelves on which the trays of material rest. Steam, hot water, or other medium is used to heat the shelves. Auxiliary equipment includes a vacuum system (steam jet or pump) and suitable condenser.

Vacuum tray dryers vary in size from 1 to 20 heating shelves with total tray surface of 2 to 2000 sq. ft. The larger dryer chambers have over-all dimensions of 9 ft. wide by 18 ft. long by 12 ft. high.

The evaporative capacity of a vacuum tray dryer depends on several factors, the most important probably being the moisture contents of the feed and product and allowable temperature of the heating medium. Usually the drying cycle consists (1) of a constant rate period where the temperature of the material is at the boiling point of water or solvent at the absolute pressure prevailing in the dryer and (2) a period in which the rate of evaporation falls off and the temperature of the material rises toward the temperature of the heating medium. It is estimated roughly that an over-all heat transfer coefficient of 1 B.t.u. per hr. per sq. ft. tray surface per degree Fahrenheit temperature difference is usually obtained, equivalent to an over-all rate of evaporation of 0.03 to 0.2 lb. water evaporated per hour per square foot tray surface. High rates apply to high-moisture content materials and low rates to low-moisture content materials carried nearly to complete dryness [see Ernst et al., *Ind. Eng. Chem.*, **30**, 1119, 1122 (1938)].

The thermal efficiency of vacuum tray dryers is usually 60 to 80 per cent. Fuel requirements vary from 1.2 to 1.7 lb. steam per pound water evaporated for high-moisture-content materials. Power is required only for the vacuum system. Annual maintenance costs are 5 to 10 per cent of the total installed cost. Labor is high but depends on the drying time, facilities for loading and unloading trays, etc.

Vacuum tray dryers have the same advantages and disadvantages as an atmospheric tray dryer with the following exceptions: there is little dusting during the drying; volatile solvents may be easily recovered by the use of a suitable condenser; and explosive mixtures of solvent and air are avoided. Power consumption is higher in a vacuum tray dryer and initial installation costs are higher.

Some of the points to be observed in the operation of a vacuum tray dryer are the following: The trays should be kept as flat as possible to obtain maximum area of contact with the heated shelves. Preferably the heating medium should not be applied to the shelves until after the vacuum has been produced in the dryer to prevent overheating or boiling over of the material at the start of drying. A free space above the material in the tray should be left to allow free removal of vapors. Air vents should be placed in the steam exhaust. Shelves should be kept free of scale and rust. A thermometer in the discharge vapor line (or observation of drip from a condenser) gives a good indication of the progress of the drying.

EQUILIBRIUM OR HYGROSCOPIC MOISTURE

The water retained by a solid in equilibrium with air at a definite humidity is termed the "equilibrium, or hygroscopic water content." It is also called "regain" because of the tendency of thoroughly dried solids to absorb water from air until the equilibrium value is reached. The hygroscopic moisture is usually expressed as percentage water, dry basis, as a function of the percentage relative humidity of the air, a plot of these two variables giving a typical S-shaped curve for nearly all solids. Data for papers, textiles, and a variety of miscellaneous solids are given on pp. 1088 to 1092.

Besides being of value in determining the necessary drying conditions near the end of the drying process, data on equilibrium water contents are of value in estimating the water acquired by materials in storage. Mildew fungi will thrive in still atmosphere only at relative humidities above 75 per cent, so that drying below 8 per cent water is sufficient to prevent such deterioration in most organic materials.

DRYER CALCULATIONS

The general process of air-drying a solid may be divided into the separate processes of first getting the water to the surface of the solid and evaporated, and, second, removal of the vapor by a stream of air. Similarly the calculations in connection with the design and operation of a dryer may be divided into two groups: first, those having to do with the estimation of the time of drying and the effect of the variables which influence the rate of drying; and second, the calculation of the heat requirements and the air quantities required to carry away the water vapor formed. The first group of calculations require a knowledge of the mechanism of drying of the given solid and have been discussed in the first part of this section. The second group involve problems in ventilation, requiring a knowledge of the psychrometric relations for air and water vapor, which are best expressed graphically in the form of a standard humidity chart.

Use of the Humidity Chart (see also pp. 1084 to 1085). The use of the proper type of humidity chart is most helpful in calculations of the air and heat quantities, and drying conditions in a proposed dryer. A form of the Grosvenor chart, in which humidities are expressed as weight of water per unit weight of dry air, is reproduced below, and the method of using it explained. The several terms employed in connection with the humidity chart are first defined.

Humidity (H) is the number of pounds of water carried by 1 lb. dry air. The **saturated humidity** is the humidity when the air is saturated with water vapor, *i.e.*, when the partial pressure of water vapor in the air is equal to the vapor pressure of water at the same temperature. The **percentage absolute humidity** is the humidity as a percentage of the saturated humidity at the same temperature, which is somewhat lower than the **percentage relative humidity**, the latter being the partial pressure of water vapor in the air as a percentage of the vapor pressure of water at the same temperature.

The **dewpoint** is the temperature at which a mixture of air and water vapor is saturated with water vapor, *i.e.*, the temperature at which the saturated humidity is equal to the humidity of the given air-water vapor mixture.

The **humid heat** is the specific heat of humid air on a basis of a unit weight of the dry air, *i.e.*, the heat required to raise the temperature of 1 lb. dry air plus the water vapor which it contains, 1°F.

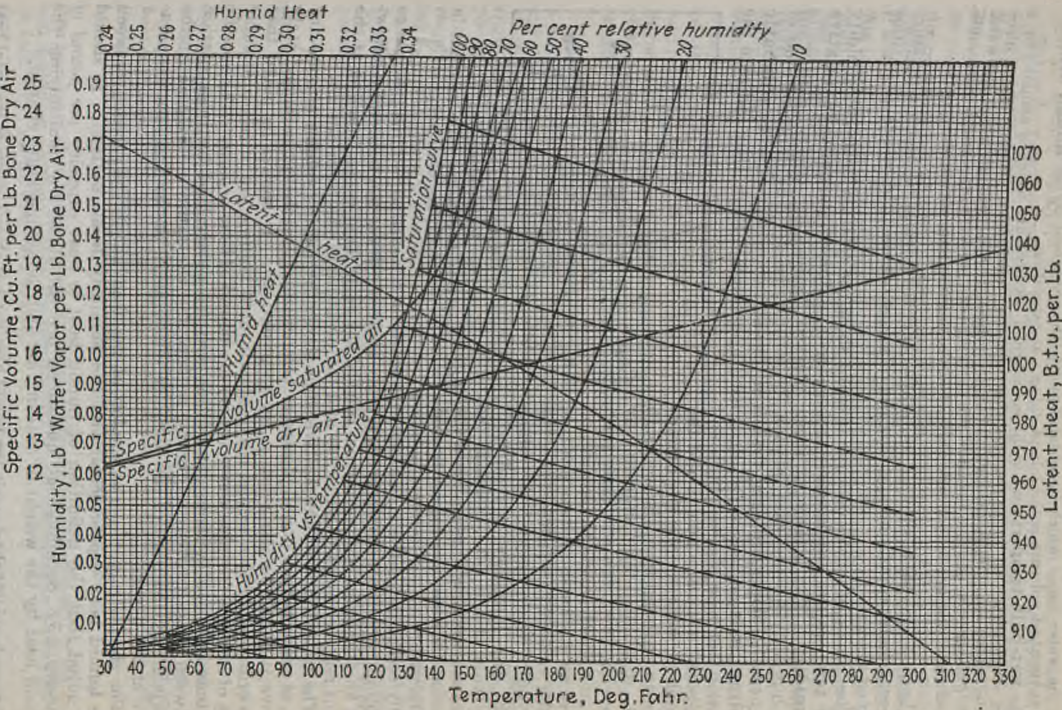


FIG. 11.—Humidity chart.

The **humid volume** or **specific volume** is the volume in cubic feet of 1 lb. dry air plus the water vapor which it contains and is the reciprocal of the **humid density**. The **saturated volume** is the humid volume at saturation.

The **wet-bulb temperature** is the equilibrium temperature which is assumed by a water surface when all of the heat necessary for the vaporization which is taking place is received by conduction through the same surface-air film through which the water vapor diffuses back out into the air. For saturated air, the air temperature, or dry-bulb temperature, the wet-bulb temperature, and the dewpoint are identical.

Figure 11, p. 1515, shows a humidity chart for normal barometric pressure on rectangular coordinates, giving the relations between the several quantities defined above.

The family of curves concave upward show the relation between humidity and temperature, each curve being for a different **percentage relative humidity**. The highest of these, marked "saturation curve," represents the humidity-temperature relation for saturated air. The group of lines of flat negative slopes ending at the left at the saturation curve are the **adiabatic cooling lines** and serve two very useful purposes. First, they indicate the relation between the wet-bulb and dry-bulb temperatures and the humidity; for example, the humidity may be found from the wet-bulb and dry-bulb temperatures by following down the adiabatic cooling line which meets the saturation curve at the wet-bulb temperature, to the vertical line representing the dry-bulb temperature, at which point the humidity is read from the scale at the left. Second, the adiabatic cooling lines indicate the relation between the humidity and temperature of air being adiabatically cooled or heated in contact with water or a wet solid. Thus the change of humidity with temperature in a truly adiabatic dryer can be seen by following an adiabatic cooling line up toward the saturation curve, starting at the point representing the humidity and temperature of the fresh hot air.

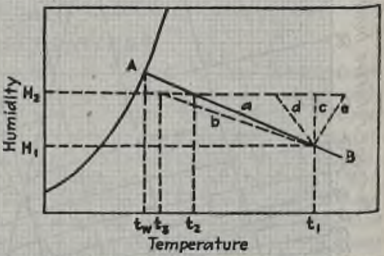


Fig. 12.—Humidity-temperature relations in various types of dryers.

The straight line of flat positive slope shows the specific volume of bone-dry air, as cubic feet per pound plotted *vs.* temperature, and the curve immediately above it is the similar relation for saturated air. It should be noted that the ordinates of this second curve of saturated volume are in terms of cubic feet of mixture (air plus water vapor) per pound of *bone-dry air*. The specific volume of air of various humidities may be determined by interpolation between these two curves, or by multiplying the saturated volume at the dewpoint by the ratio of the absolute temperature to the absolute temperature at the dewpoint.

The steep straight line at the left of the chart shows the relation between the humid heat (upper scale) and the humidity (scale at left). It should be noted that the definition of humid heat is heat capacity *per pound of bone-dry air*. Sensible-heat quantities are calculated by multiplying the humid heat by the weight of *bone-dry air* and the temperature change.*

---

* Some humidity charts omit the humid-heat line but show a curve of the total heat of saturated air above an arbitrary datum temperature, which is of general application because the total heat of air at any temperature and humidity is the same as the total

For convenience, a line for the latent heat of vaporization is also shown plotted against temperature, the scale for the latent heat being at the right.

Humidity-temperature Relations within the Dryer. As pointed out above, the adiabatic cooling lines on the humidity chart indicate the relation between the temperature and the humidity of air passing through a truly adiabatic dryer, *i.e.*, one in which all of the sensible heat given up by the air in cooling goes to evaporate water from the wet stock. Referring to the diagrammatic humidity chart shown in Fig. 12, where AB is one adiabatic cooling line, it follows that air entering an adiabatic dryer at a temperature t_1 and a humidity H_1 will cool following this cooling line toward the point A. Air leaving with a humidity H_2 will consequently have cooled to t_2, the wet-bulb temperature of the air throughout the dryer being t_w. Because of heat loss to the surroundings the operation is seldom truly adiabatic, and the final temperature is somewhat lower than t_2, so that the true humidity-temperature relation is represented by the line b, having a smaller slope than the adiabatic cooling line. The ratio $(t_1 - t_2)/(t_1 - t_3)$ then gives a measure of the heat efficiency of the dryer. For the case of dryers containing steam coils and which are maintained at a constant temperature, the humidity-temperature relation is obviously represented by the vertical line c, assuming the initial and final humidities to be H_1 and H_2 as before. The heat supplied within the dryer itself is usually less but may be greater than the total heat requirements of the dryer. If less, the cooling is indicated by some such line as d; and if greater, by a line such as e having a positive slope.

Example 6. A dryer is to be designed to produce 1000 lb. per hr. of product containing 4 per cent water from a wet feed containing 42 per cent water (wet basis). Fresh air at 70° with a relative humidity of 40 per cent will be preheated to 200°F. before entering the dryer, and will leave the dryer with a relative humidity of 60 per cent. Assuming the dryer to operate adiabatically, calculate: (*a*) the cubic feet per minute fresh air at 70°F.; (*b*) the heat supplied the preheater as B.t.u. per hour.

Solution. The humidity of air at 70°F. with a relative humidity of 40 per cent is found from the humidity chart to be 0.006. This air is preheated to 200° without changing its humidity and in passing through the dryer cools, following parallel to the nearest adiabatic cooling line on the chart to the 60 per cent relative-humidity curve, at which point its humidity and temperature are seen to be 0.028 and 103°F., respectively. Each pound of dry air consequently picks up 0.028 − 0.006 = 0.022 lb. water vapor in passing through the dryer. The evaporation is

$$1000 \times 0.96[(42/58) - (4/96)] = 655 \text{ lb. water per hour}$$

and the air required to carry away this water vapor is

$$\frac{655}{(0.022 \times 60)} = 496 \text{ lb. per min.}$$

By interpolation between the specific-volume curves for dry and for saturated air at 70°F., the fresh air is found to have a specific volume of 13.4 cu. ft. per lb. dry air. The volume of fresh air required is therefore 13.4 × 496 = 6660 cu. ft. per min. The humid heat of the fresh air is found from the humid-heat line to be 0.243 and the heat required in the preheater is consequently

$$0.243 \times 496 \times 60 \times (200 - 70) = 941{,}000 \text{ B.t.u. per hr.}$$

The heat requirements of such a dryer neglecting heating of the solid and of the trucks and neglecting radiation loss to the surroundings would consequently be 941,000/655 or 1440 B.t.u. per lb. water evaporated.

---

heat of saturated air at its wet-bulb temperature. The humid-heat line is more generally applicable, however, and calculations using it are believed to be more accurate than values read from a total-heat line on a small chart.

Example 7. Repeat the calculations of Example 6 for a similar dryer, but assuming the air to enter the dryer at 150°F., and sufficient heat to be supplied within the dryer itself to maintain the air temperature at 150°F., at which temperature it will leave with a relative humidity of 60 per cent.

Solution. From the chart, the humidity of the air leaving at 150°F. with a relative humidity of 60 per cent is seen to be 0.111. The air required is therefore

$$\frac{655}{(0.111 - 0.006)60} = 104 \text{ lb. per min.}$$

or 104 × 13.4 = 1392 cu. ft. per min. fresh air.

The heat required in the preheater is

$$0.243 \times 104 \times 60 \times (150 - 70) = 121{,}000 \text{ B.t.u. per hr.}$$

Assuming the stock to enter at 70°F., the heat required for evaporation is equivalent to that required to heat the water to 150°F., plus that required to evaporate it at 150°F., **no matter what the actual temperature of evaporation may be.** Thus, the heat supplied within the dryer will be

$$655 \times (80 + 1007) = 712{,}000 \text{ B.t.u. per hr.}$$

Neglecting heat to heat the trucks and dry solid, and radiation losses, the total heat requirements of this dryer will be

$$\frac{(712{,}000 + 121{,}000)}{655} = 1272 \text{ B.t.u. per lb. water evaporated}$$

Example 8. A dryer is to produce 800 lb. per hr. of a product containing 10 per cent water from a feed containing 60 per cent water (wet basis). In order to produce the desired drying conditions the air will be supplied to the dryer at 212°F. with a dewpoint of 111°F., and leave at 154°F. with relative humidity of 50 per cent. Part of the waste humid air will be recirculated and mixed with fresh air entering at 70°F. with a relative humidity of 52 per cent before being reheated. Again neglecting heat loss by radiation and the heat required to heat the trucks and the dry solid, calculate the air and heat requirements of the dryer.

Solution. The rate of vaporization of water is $800 \times 0.90 \times [(^{60}/_{40}) - (^{10}/_{90})] =$ 1000 lb. per hr. From the chart, air having a dewpoint of 111°F. is seen to have a humidity of 0.060, and the air leaving at 154°F. with a relative humidity of 50 per cent has a humidity of 0.100. The air required is therefore

$$\frac{1000}{(0.100 - 0.060)60} = 416 \text{ lb. per min.}$$

The fresh air enters with a humidity of 0.008, so if x represents the fraction of the waste air recirculated (on a bone-dry basis), then from a humidity balance

$$0.100x + 0.008(1 - x) = 0.060$$

whence $x = 0.565$, *i.e.*, 56.5 per cent of the waste air is recirculated and mixed with fresh air. The fresh air will be (1 − 0.565) × 416 = 181 lb. per min., and, since the specific volume of the fresh air is 13.4 cu. ft. per lb., the volume of fresh air required will be 181 × 13.4 = 2425 cu. ft. per min. at 70°F. The total heat requirements of such a dryer are most easily calculated as the sum of the heats required to heat the water and form vapor at 154°F., plus the heat to heat the fresh air from 70° to 154°F., *i.e.*,

$$1000(84 + 1005) + 0.243 \times 181 \times 60 \times (154 - 70) = 1{,}311{,}000 \text{ B.t.u. per hr.}$$

or 1311 B.t.u. per lb. water evaporated.

Example 9. 6000 lb. per hr. wet filter cake containing 26 per cent water (dry basis) is to be dried to 3 per cent water (dry basis) in a countercurrent tunnel-truck dryer, using cross circulation of air over the trays. The dryer will utilize thermostatic control instruments to maintain a constant air temperature of 145°F. The fresh air will enter the preheater 50 per cent saturated at 70°F., and leave the dryer at an absolute humidity of 70 per cent. The cake will be placed on trucks on screen-bottom trays, 4 lb. dry cake per square foot of tray area and 240 sq. ft. of tray surface per foot of length of the dryer. Calculate the required dryer length.

NOTE. The critical moisture content may be taken as 10 per cent (dry basis) and the equilibrium moisture may be assumed negligible under the conditions at the air entrance. The air velocity over the trays is sufficient to obtain a surface coefficient of heat flow from air to stock of 2.5 B.t.u./(hr.) (sq. ft.) (°F.) for both upper and lower surfaces of the cake.

Solution. Since the weight of dry cake is 2 lb. per sq. ft. exposed wet surface, the evaporation per square foot is $2 \times (0.26 - 0.10) = 0.32$ lb. in the constant rate period and $2 \times (0.10 - 0.03) = 0.14$ lb. in the falling rate period. The humidities of the air entering and leaving the dryer are found from the chart to be 0.008 and 0.125, respectively. Since all of the water leaving the stock is picked up by the air we may write

$$G(H_2 - H_1) = R(W_2 - W_1) \tag{5}$$

where G and R are the pounds per hour of bone-dry air and of bone-dry stock, respectively, H is the air humidity, W the pounds of water per pound dry stock, and the **subscripts, 1 and 2** refer to conditions at the air entrance and exit respectively. This indicated linear relationship between H and W is illustrated in Fig. 13, which shows the air temperature and humidity plotted against the water content of the material. The humidity at the critical point (10 per cent water) is found from this plot to be 0.044, and the corresponding wet-bulb temperature at 145°F. is found from the humidity chart to be 107.7°F. The wet-bulb temperature at any other point in the dryer may be determined in a similar manner and plotted *vs.* water content of the solid as shown in Fig. 13. Neglecting heat flow to the cake by radiation or conduction from adjoining dry surfaces, the wet surface in the constant rate period may be assumed to be at the wet-bulb temperature. (The rate is not constant in this period because of changing air humidity.) The temperature difference between air and solid at the critical point is consequently $145° - 107.7° = 37.3°$F. and at the air exit $145° - 134.7° = 10.3°$F.; the logarithmic mean temperature difference is 21.0°F. Assuming a latent heat of 1020 B.t.u. per lb., the time required in the constant rate period is

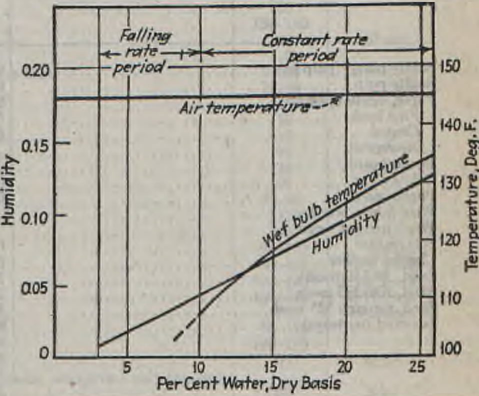


FIG. 13.—Conditions in dryer of illustrative design problem.

$$\frac{(0.32 \times 1020)}{(2.5 \times 21)} = 6.2 \text{ hr.}$$

The rate of drying at the critical point is

$$\frac{(2.5 \times 37.3 \times 100)}{(1030 \times 2)} = 100\left(\frac{dW}{d\theta}\right) = 4.53 \text{ per cent per hour}$$

Assuming the rate of drying in the falling rate period to be proportional to the free water content,

$$100\left(\frac{dW}{d\theta}\right) = -100C(W - W_E) = -C(10 - 0) = 4.53$$

Consequently $C = -0.453$, and Eq. (4) may be written

$$\log_e \frac{(10 - 0)}{(3 - 0)} = 0.453\theta$$

from which θ, the time for the falling rate period, is 2.7 hr. The total time of drying is therefore, $6.2 + 2.7 = 8.9$ hr., and

$$\text{The dryer length} = \frac{(8.9 \times 6000)}{(1.26 \times 4 \times 240)} = 45 \text{ ft.}$$

Table 6. Approximate Critical Moisture Contents Obtained on the Air-drying of Various Materials, Expressed as Percentage Water on the Dry Basis

Material	Thickness, in.	Critical moisture, % water, dry basis
Sulfite pump (pulp lap)	0.039	110
Sulfite pulp	0.25–0.75	60–80
Paper, white eggshell	0.0075	41
Fine book	0.005	33
Coated	0.004	34
Newsprint		60-70
Beaverboard	0.17	Above 120
Celotex	0.44	160
Poplar wood	0.165	120
Wool fabric, worsted		31
Wool, undyed serge		8
Sole leather	0.25	Above 90
Chrome leather	0.04	125
Sand, 50–150 mesh	2.0	5
Sand, 200–325 mesh	2.0	10
Sand, through 325 mesh	2.0	21
Sea sand (on trays)	0.25	3
	0.5	4.7
	0.75	5.5
	1.0	5.9
	2.0	6.0
Brick clay	0.62	14
Plastic clay brick mix	2.0	19
Flint clay refractory brick mix	2.0	13
Silica brick mix	2.0	8
English china clay	1	16
Kaolin		14
Subsoil, clay fraction 55.4%		21
Subsoil, much higher clay content		35
Barium nitrate crystals, on trays	1.0	7
Carbon pigment	1	40
Copper carbonate, (on trays)	1 –1.5	60
Iron blue pigment, (on trays)	0.25–0.75	110
Lithol red	1	50
Lithopone press cake (in trays)	0.25	6.4
	0.50	8.0
	0.75	12.0
	1.0	16.0
Niter cake fines, on trays		Above 16
Prussian blue		40
Pulp lead, initially 140% water		Below 15
Rock salt (in trays)	1.0	7
Stannic tetrachloride sludge	1	180
White lead		11
Whiting	0.25–1.5	6–9
Gelatine, initially 400% water	0.1 –0.2 (wet)	300

Table 7. Approximate Temperature and Time of Drying of Various Materials

Material	Temp., °F.	Time
Apples, sliced	175	6.5 hr.
Beans	140	18 hr.
Blankets	120	40 min.
Brick, common	170	30 hr.
Cabbage, raw	150	4.5 hr.
Candied peel	165	2 hr.
Carrots, raw	140	5 hr.
Casein, on trays in tunnel dryer	140	8 hr.
Cement sacks	120	1 hr.
Coconut	145–155	4–6 hr.
Cocoa-fiber mats	170–210	10 hr.
Coffee	160–180	24 hr.
Cores, oil sand, for molding ½–1 in. thick	300	30 min.
3 in. thick	480	2.5 hr.
16 in. thick	700	10 hr.
Cotton linters	180	
Feathers	150–180	
Films, motion picture, on drums	80	20 min.
Furs	110	
Glue, bone, thin sheets on wire trays	70–90	6–9 days
Glue, skin	70–90	2 days
Glue size on furniture	130	4 hr.
Gut	150	
Gypsum wallboard { start wet	350	1 hr.
finish dry	190	
Hanks, on poles	120	2 hr.
Hides, thin	90	2–4 hr.
Hides, heavy	70–90	4–6 days
Hops	130	12 hr.
Leather, thick sole	90	2–6 days
Lumber, 1½-in. oak	90–125	8 days
2–3-in. oak	90–125	3–4 weeks
green hardwood	100–180	3–180 days
green softwood	160–220	2–14 days
softwood, 1 in. thick	175	3–5 days
Macaroni	90–110	
Matches	140–180	
Millboard sheets	95	10 hr.
Molds, green sand, cast-iron flasks, one surface only exposed,		
8 in. thick	600	6 hr.
13 in.	700	13 hr.
Nuts	75–140	24 hr.
Peaches	135	25 hr.
Pears	140	24 hr.
Peas	150	6 hr.
Potatoes, sliced	85	4 hr.
Potatoes, steamed	170	6.5 hr.
Pottery	120	
Rags	180	
Ramie fiber	140	10 hr.
Rubber	80–90	6–12 hr.
Sand, loose, 1 in. deep	300	10–15 min.
Shade cloth	240	1–2 hr.
Shirts	120	20 min.
Soap	100–125	12–72 hr.
Stains and varnish on wood	110	7 hr.
Starch	180–200	1–12 hr.
Stock feed, mixed	180–220	20–30 min.
Sugar	150–200	20–30 min.
Tannin and other chemicals, spray dried	250–300	Instantaneous
Terra cotta	150–220	12–96 hr.
Tobacco, leaf	85–130	12 hr.
Tobacco, stem	180–200	12 hr.
Wallboard	200–250	12–24 hr.
	340	5 hr.
Wool	105	

NOTE. In the solution on page 1519 the neglect of heat flow to the material by radiation and by conduction from the trays introduces an unknown factor of safety, because the actual surface temperature will be somewhat above the wet bulb. Although this decreases the temperature difference, the rate of drying is actually greater, because of the increased vapor pressure of water at the surface. A second factor of safety is introduced into the design by determining the constant C from the rate of drying at the critical point. If unsaturated surface drying is of importance in the falling rate period, humidity will have an effect, and, as the average humidity of the air in this period is less than at the critical point, the time for the falling rate period will be somewhat less than calculated.

DATA ON CRITICAL MOISTURE CONTENTS AND DRYING CONDITIONS FOR VARIOUS MATERIALS

In spite of the importance of the critical moisture content of solids in determining the time of drying and the most suitable drying conditions, very few data are available as to the value of the critical moisture for various materials. Table 6 shows such data collected from various sources for a number of different common materials, but it should be emphasized that where possible the drying characteristics of the sample to be dried should be determined by suitable laboratory tests. The values tabulated are approximate at best, since the critical moisture content varies not only with the material but with the rate of drying and with the initial moisture content.

It appears that the constant rate period ends when the moisture content *at the surface* reaches some specific value. If the rate of drying is great, the moisture gradients within the solid will be steep and the average moisture content considerably greater than that at the surface. It is for this reason that the critical moisture content (average through the material) increases with increase in rate of drying.

Table 7 indicates the approximate time of drying for a large number of miscellaneous materials when dried at the temperatures indicated. Obviously the drying times in such a tabulation are only approximate, and will vary with the thickness or loading of the material on trays, the air velocity over the material, the air humidity, and other variables as discussed above.

II. THE DRYING OF GASES*

It is frequently necessary to remove practically all of the water vapor from air or other gas which is to be liquefied or used in a chemical process, where the presence of more than slight traces of water vapor must be avoided. Air to be liquefied must be dried because of the stoppage of the tubes of the heat exchangers by ice when the air which is being cooled contains water vapor. The drying of air usually refers to the much more complete removal of water vapor than is accomplished by dehumidification. Four important methods [Holden, *Chem. Met. Eng.*, **28**, 801 (1923). Quarendon, *Ind. Chemist*, **15**, 279 (1939)] of drying gases, namely, by **reagents**, by **compression**, by **refrigeration**, and by **adsorption**, are discussed below.

Reagents. The common laboratory method of drying gases by contact with solid or liquid reagents having marked affinity for water may also be used on a plant scale. Although equilibrium between gas and reagent may not be obtained practically, the equilibrium relationship is a criterion of the degree of moisture removal possible with a given reagent. Table 8 (see also p. 1118) gives the moisture concentration of air in equilibrium with a number of different materials, in milligrams of water vapor per liter of gas dried at 25°C. ("International Critical Tables," vol. 3, p. 385, McGraw-Hill, 1928).

---

* See Sects. 9. 10, and 11.

Solid reagents should be packed in vertical pipes or columns, and the gas passed evenly through the mass of the lumps or granules. With packed horizontal tubes, channeling along the top of the passage reduces the capacity and efficiency of the apparatus (for data on pressure drop in packed tubes, see Sec. 6, p. 771, and Sec. 10, p. 1119). Lump absorbents, such as fused KOH, $CaCl_2$, anhydrous calcium sulfate ("Drierite"), or magnesium perchlorate ("Anhydrone"), are best supported in perforated bucket-like containers which fit inside a tower of small diameter and rest one upon the other. Unless supported in some such manner, the material packs down as it absorbs water and melts, forming a dense mass through which it is very difficult to force the gas.*

Table 8. Milligrams Water Vapor per Liter of Gas Dried at 25°C.
Approximately equal to the equilibrium vapor pressure of water in millimeters mercury.

A filter at −194°C	1.6×10^{-23}	CaO	0.2
P_2O_5	$<2 \times 10^{-5}$	Granulated $CaCl_2$	0.14 to 0.25
Fused KOH	0.002	95.1% H_2SO_4	0.3
Al_2O_3	0.003	Fused $CaCl_2$	0.36
H_2SO_4	0.003	$ZnCl_2$	0.8
MgO	0.008	$CuSO_4$	1.4
Fused NaOH	0.16		

When liquid absorbents are used, the gas should either be passed through a spray of the reagent or passed up through a tower packed with a suitable packing of large surface over which the liquid is distributed (see p. 1197). Sulfuric acid may be used in a packed column, but, besides the usual difficulties of handling an acid, cooling coils must be provided to remove not only the latent heat of condensation of the water vapor, but the very appreciable heat of solution of the water in the acid. A strong aqueous solution of lithium chloride is quite effective and less corrosive. It may be regenerated in steam-heated evaporators. In using any liquid reagent entrainment must be avoided because of possible contamination of the material being dried.

A process for drying city gas has been developed, using glycerin as a drying agent, the diluted glycerin being reconcentrated in a small vertical-tube evaporator and reused. Various types of gas washers may be employed and the glycerin loss may be made very small. The plant [Tupholme, *Gas Age-Record*, **63**, 311–313 (1929)] is said to reduce the dewpoint from 62.5° to 27°F., at an estimated cost of 0.3 ct. per 1000 cu. ft. of gas treated. In a plant handling 1,000,000 cu. ft. of gas per 24 hr., the costs are estimated at $1.30 per day for depreciation, and $1.95 per day for steam, cooling water, and maintenance. Ethylene glycol is quite hygroscopic and may be used in place of glycerin.

Compression. Air may be dried by compression, since the humidity of saturated air at a given temperature decreases rapidly as the total pressure is increased. This is illustrated by Table 9 which shows the saturated humidity of air at 68°F., as a function of the total pressure. If the air is initially unsaturated, it is first saturated by compression before any condensation takes place. In ordinary adiabatic compression the air, which is heated in the compressor, is cooled in an after-cooler, where condensation takes place.

Leaving the after-cooler the air tends to be supersaturated, containing water as a mist or fog and should be passed through a mist separator before being

---

* See also Sec. 9, p. 1079.

dried further by other means. This is especially important where chemical towers containing such reagents as lump KOH are used, as these towers act as separators, and the water thrown out dissolves the reagent which is then rapidly used up.

Table 9. Saturated Humidity at 68°F.

Pressure, Atm.	Saturated Humidity, Lb. Water per Lb. Dry Air
1	0.0147
2	.0072
5	.0029
10	.00144
50	.000287
200	.000072

When the desired percentage condensation of the water vapor in the air is low, compression cannot compete on a cost basis with dehumidification. For higher percentage condensation, the required pressures involve equipment and power costs which are ordinarily prohibitive; consequently drying by compression is not resorted to, unless the gas must be compressed for a later step in the process.

Refrigeration. By cooling air well below its dewpoint by means of brine or ammonia expansion coils, it is possible to condense most of the water content and greatly reduce its humidity. Thus, the saturated humidity of air at 11°F. is only about 10 per cent, and at −20°F. is less than 2 per cent of that at 68°F. Difficulty is encountered in drying to low humidities, however, because of the formation of ice on the cooling surfaces, and the consequent necessity of a frequent shutdown of part of the coils so that this ice formation may be thawed. The ice acts as a thermal insulator and the surface temperature at which condensation takes place is therefore considerably above the temperature of the cooling medium. A common error is to assume that the air leaving the cooling coils must be saturated at its dry-bulb temperature. Actually, it is considerably drier than saturated because its dewpoint tends to approach the surface temperature of the coils.

Large plants have been built for the purpose of drying air blast for blast furnaces by passage over refrigerated brine coils (Gayley dry blast). Refrigeration is also used in cooling water supplied to sprays in dehumidifiers.

Adsorption.* Certain porous solids show a great affinity for water vapor because of the intense capillary action exhibited by the large number of ultramicroscopic pores. For example, silica gel, which is an artificially prepared form of porous SiO_2, will take up 20 to 30 per cent of its own weight of water and still appear dry. The back pressure of water vapor over the material is very low, its efficiency as a desiccator falling somewhere between that of P_2O_5 and H_2SO_4. The efficiency of adsorption remains high up to the "break point," after which its capacity for further adsorption falls off rapidly. The gel may then be reactivated by heating, without change in the structure or efficiency. Silica gel will also absorb CO_2 if the pressure is below the critical, and the CO_2 can be removed by decompression.

The silica gel is placed in stationary containers through which the air is passed in parallel. Successive units are cut out for reactivation, several units remaining in operation while the gel in one is being reactivated. For reactivation it is unnecessary to remove the gel from the container, so that dusting is eliminated.

---

* See Adsorption, Sect. 11, pp. 1269–1336.

A large air-drying plant has been built in Scotland [Lewis, *J. Iron Steel Inst. (London)*, September, 1927; *Engineering* (London), p. 583, Dec. 30, 1927; *J. Soc. Chem. Ind.*, **46**, 902 (1927)] to handle 35,000 cu. ft. per min. of fresh air for a blast furnace, using silica gel as the drying medium. It consists of six units, five being in operation while the sixth is being reactivated. Each unit consists of a large steel box in which are trays containing the gel in granular form. The bottoms are perforated and the air passes through the trays in parallel.

Table 10. Operating Costs of Silica Gel Gas-drying Installations*

Capacity, cu. ft. per min., free measure	Pressure, lb. per sq. in. gage	Absolute humidity		Operating costs 1000 cu. ft. free measure, cts.
		Initial	Final	
3,000 air	0	0.0114	0.0009	3.9
7,000 air	0	0.0114	0.0009	3.3
16,000 air	0	0.0114	0.0009	3.1
100 air	100	0.0028	0.00007	1.05
300 air	300	0.001	0.00007	0.35
150 CO_2	1000	0.00026	0.00005	0.30

* Manufacturer's data, courtesy The Silica Gel Corporation.

The gel requirements are approximately 2 lb. per cu. ft. per min. fresh air, and the moisture content of the air is reduced from 3.1–4.9 gr. per cu. ft. to 1.1–1.6 gr. per cu. ft. Reactivation is accomplished by combustion of blast-furnace gas, the gas used representing the equivalent of 7 tons of coal per day. An air filter is found necessary to remove dirt from the fresh air and to prevent plugging of the gel. The air-drying installation is said to be largely responsible for a recorded increase of 12 to 17 per cent in the capacity of the blast furnace.

Activated alumina is also used successfully [Derr, *Ind. Eng. Chem.*, **30**, 384 (1938)], although capable of absorbing somewhat less water than silica gel. When used in thick beds, the air treated is dried to a dewpoint of as low as −70°C. Beds of 2 to 3 in. and air velocities of 75 to 150 ft. per min. serve to dry air partially, and this treatment may be employed as an alternative to dehumidification in air conditioning. Gases other than air may be dried successfully providing they are handled at pressures below the critical. Liquids can also be dehydrated [Derr and Willmore, *Ind. Eng. Chem.*, **31**, 866 (1939)].

SECTION 14

MIXING OF MATERIAL

INCLUDING GASES, LIQUIDS, PASTES, PLASTICS, AND SOLIDS

BY

Kenneth S. Valentine, A. B., Ch. E., New York Manager, The Patterson Foundry and Machine Co.; member, American Institute of Chemical Engineers.

Gordon MacLean, Ch. E., Sales Manager, The Turbo-Mixer Corporation; member, American Chemical Society, American Institute of Chemical Engineers.

CONTENTS

PAGE

FUNDAMENTALS OF MIXING

PAGE

TYPES OF MIXERS

FITTING THE MIXER TO THE OPERATION

Recommended Equipment

THE TRANSMISSION OF POWER TO MIXERS

FROM LABORATORY TO PLANT

HEAT TRANSFER IN MIXERS

MIXING OF MATERIAL

The subject of *mixing* has been one of the most difficult of the unit operations of chemical engineering to submit to scientific analysis. To date there has been developed no formula or equation that can be used to calculate the degree or speed of mixing under a given set of conditions. Steps have been taken to improve this situation, however, notably by Hixson and White and their respective coworkers and more recently by Büche and by Brothman and Kaplan.

It is sometimes said that the power input to a mixer alone gives a true measure of the thoroughness of mixing because a definite amount of work is required to mix the particles of material within the container. However, this can *never* be true in practice, because of the immeasurable interferences—cross currents, eddies, etc.—which are set up (even in the mixing of plastics and solids) within the container. Consequently, a tremendous amount of power might be consumed in producing a very vigorous local action with good mixing around the mixing element but with no action at all outside of this zone because the energy has been dissipated in producing local interferences.

Because of the fact that the mixing art is so empirical, and because of the almost infinite variety of substances to be mixed, the number of mixers which have been developed is enormous. Some are good and some are poor, but there are few standards. Each industry has developed mixers peculiarly for its own use. Such diversification is not only unnecessary, but it is the greatest obstacle to a sound coordination of knowledge of the subject. When mixing is viewed from the standpoint of the physical characteristics of the materials to be mixed, analysis will indicate that about 40 distinct types will very satisfactorily cover every mixing operation in every industry.

The main purpose of this section therefore, is to classify all mixing problems according to the materials to be mixed and to recommend a type or types of mixer to be used for each of these problems.

FUNDAMENTALS OF MIXING

General Objectives of Mixing

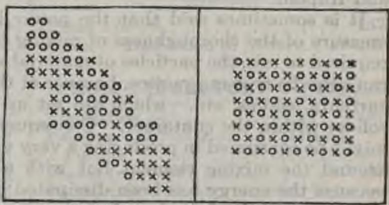
Whether we are dealing with liquids, solids, or gases, or any combination of these phases, the fundamental object to be accomplished by theoretically perfect mixing is always the same. It can be stated as follows: *In all cases, two or more materials existing either separately or in an unevenly mixed condition are, by mixing, to be put into such a condition that each particle of any one material lies as nearly adjacent as possible to a particle of each of the other materials* (see Fig. *A*). In practice these perfect results are never obtained, and it may be stated that in some few cases such theoretically perfect mixing would be undesirable. The result of mixing may be a blend, a dispersion (suspension, or emulsion), a solution, or a chemical reaction, and in industrial work the desired quality of the finished product in almost every case is the largest factor in determining the required thoroughness of mixing. For instance in preparing cod-liver oil emulsion (with water), the most intimate mixture is the most desirable because this product must never show separation after being sold, whereas in treating (temporarily emulsifying) certain gasolines with caustic soda solution a too intimate mixture is to be avoided because subsequent rapid separation is necessary.

Two general rules must be fulfilled in solving every mixing problem.

Rule 1. A degree of mixing sufficient to yield the desired results must be produced. This rule refers only to microscopic and ultramicroscopic characteristics of the mixture.

Rule 2. A satisfactory rate and direction of motion of the **entire** body of material (however remote from the mixing element some portions are) must be established and maintained, so that **all** of the material within the container may be mixed to the desired degree within the optimum time. This rule refers to characteristics of the mixture that may be observed with the eye.

Considering Rule 1, intimacy or degree of mixing depends on the differential rate of flow of the various constituents of the mixture. This differing rate is produced either by direct physical contact between the ingredients of the mixture and the mixer itself (of which the container must always be considered a part), or by the state of motion imparted to the materials by the mixing element, or by both of these. Shear, either directly or through the intermediate steps of momentum and impact, is the most important factor in translating the forces generated by the mixer into differential rate of flow. The greatest intimacy is generally produced in the vicinity of the mixing element, because the most vigorous motion occurs in that region.



UNMIXED
Two Materials Showing
Uneven Particle Distribution

MIXED
Two Materials Showing
Even Particle Distribution

FIG. A.—The effect of mixing.

A substantial part of the total power input is expended in bringing about this intimacy of mixing, while the remainder is utilized in maintaining the necessary general flow.

When Rule 2 is fulfilled, the entire contents of the container will be evenly distributed and in proper condition to be repeatedly subjected to the more intense mixing action required by Rule 1. Stratification and settling will render the performance of an otherwise excellent mixer quite worthless. Unless both horizontal and vertical flows are sufficient, and unless all of the material in the container is moved frequently into the zone of intensified action, whatever mixing, dispersion, solution, or reaction takes place there will be **completely** nullified because of insufficient mixing in remote parts of the container. This point cannot be too strongly stressed, for it is often neglected, with disastrous results to yields, time, and power consumed. In cases of liquid mixing, too much turbulence interferes with the proper operation of Rule 2, because so much of the momentum imparted by the mixing element to the liquid is lost in local eddies and interferences that the residual momentum is insufficient to carry with any vigor to the extremities of the container.

Under Types of Mixers, the fundamental principles of design as applied to the solution of typical problems will be discussed.

Practical Objectives of Mixing

Mixing may accomplish one of the following things:

1. Simple Physical Mixture. Mixing to a satisfactorily blended product:

a. Two or more miscible fluids such as molasses and water.

b. Two or more uniformly divided solids such as powdered dyes of different shades.

c. Mixtures of phases where no reaction or change of particle size takes place. Examples of this are concrete mixing and mixing of sand soap.

2. Physical Change. This may be:

a. Dissolution. The passing of gas, liquid, or solid into another phase by solution, such as the dissolving of chlorine, butanol, or salt in water; the deodorization of vegetable oil by blowing superheated steam through it; leaching; etc. The work of Hixson and Wilkins at Columbia University forms a valuable and practical addition to the subject of dissolving solids in liquids [Performance of Agitators in Liquid-solid Chemical Systems, *Ind. Eng. Chem.*, **25**, 1196 (1933)]. It constitutes an important verification of the "Cube Root Law" of Hixson and Crowell [*Ind. Eng. Chem.*, **23**, 923, 1002, 1160 (1931)].

b. Crystallization. The formation of crystals from a supersaturated solution.

c. Adsorption. The selective removal of minor constituents by surface phenomena, as in the bleaching of oils by fuller's earth, the decolorization of syrup, etc., by means of vegetable carbon.

d. Flocculation. The collection of a precipitate into flocs for the purpose of settling or filtration.

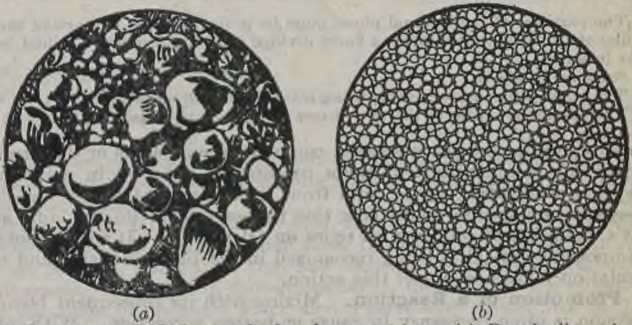

(*a*) (*b*)

FIG. *B*.—Ice-cream mix before and after homogenizing. (*a*) Rough dispersion; (*b*) Fine dispersion.

3. Dispersion. Mixing to a quasi-homogeneous product:

a. Two or more immiscible fluids.

b. One or more fluids with finely divided solids.

Common Examples. a. Mayonnaise (liquid in liquid); marshmallow whip (gas in liquid).

b. Paints (solid in liquid); ore flotation (solids, immiscible liquids, and gas in liquid).

The continuous phase is the external phase, while the discontinuous (or disperse) phase is the internal phase. If the internal phase is liquid, the dispersion is an emulsion; if it is solid, the dispersion is a suspension; and if it is gas, it is a foam.

For practical purposes the degree of dispersion of the internal phase desired in a finished product varies considerably.

Figure *B* shows a comparison of the ingredients of ice cream before and after homogenization. Both *a* and *b* appear about the same to the eye, but, under the microscope, *b* is seen to consist of much finer and more uniformly sized droplets than *a*. Therefore *b* will produce a more uniform finished product because the butterfat particles have been so completely dispersed that all danger of subsequent agglomeration has been removed.

The particle size of the dispersed phase is measured in microns. (1 micron $= 1\mu = 0.001$mm.) So-called suspensions or emulsions have a particle diameter greater than 0.1μ. Colloidal solutions have particle diameters between 0.1μ and 0.001μ.

Emulsions may be classed as permanent and temporary. In Fig. *B*, *b* would probably be permanent, while *a* could be temporary.

To form dispersions or emulsions which will not *segregate*, one or more of these conditions must be fulfilled:

a. The specific gravity of the phases must be made equal or gravity separation prevented by overcrowding in dispersed phase.

b. The particles must be in Brownian movement due to their small size.

c. If neither of the former conditions can be fulfilled the viscosity of the outer phase must be high enough to prevent settling or

d. The outer phase must have a slight set (curve *ABC* on Fig. *G*, p. 1536).

To prevent *separation* of dispersions by aggregation of the internal phase, one or more of the following conditions must be fulfilled:

e. The particles of the internal phase must be protected from contacting each other by a film of a third material, either a finely divided solid or a protective colloid, adsorbed by the interface.

f. Same as (*b*) above.

g. The outer phase must have a strong set (butter-holding brine).

h. The specific gravities of the two phases should not be widely different.

Taking examples from successful commercial emulsions or dispersions, we find various combinations of these conditions. The oil in mayonnaise is kept from segregating by (*a*) and from separating out by (*e*) and (*h*). A high-grade non-settling paint gets this property from (*b*), (*c*), (*d*), and (*e*). Scott's Emulsion of cod-liver oil relies on (*c*) and (*e*). It will be noted that the importance of (*e*) is being recognized in the paint industry and modern formulation takes account of this action.

4. Promotion of a Reaction. Mixing with its consequent blending or dispersion is often necessary to cause or hasten a reaction. With intensive mixing, it is possible in some cases to lower the temperature or pressure at which a given reaction will occur. Examples are: (*a*) neutralizing, (*b*) halogenating, (*c*) hydrogenating oils, (*d*) hydrolyzing starches to sugars, etc. Furthermore, by thorough mixing, undesirable side reactions are prevented by eliminating the danger of local excess of reagents or local overheating.

Physical Factors in Mixing

The physical factors which play a part in all mixing operations are:

1. The consistency, or apparent viscosity, of the mixture at mixing velocities. This is the most important physical factor.
2. The specific gravity of the continuous phase and the relative gravities of each phase.
3. Other physical properties of the materials before, or during, mixing.
4. Relative proportion of the materials and their order of addition to the mixture.

1. Consistency, or Apparent Viscosity, at Mixing Velocities. These important terms, for the purposes of this section, will be considered synonymous. Therefore they require discussion, and the curves in Figs. *C*. *H*, *I*, *J*, and K constitute a very necessary part of this discussion. The terms viscosity, plasticity, pseudoplasticity or thixotropy, and inverted plasticity must also be explained.

The curves in *H* were obtained on the Gardner mobilometer, illustrated in Fig. *F*. The other curves were obtained by hanging certain weights in grams on the driving cord of the modified Stormer viscometer illustrated in Fig. *E* and measuring the resulting revolutions per second of the blades, the blades being immersed in a pint can of the material under test, to a depth of 1.55 in.

Viscosity is the resistance of a material or a mixture to flow when force is exerted upon it. **Fluidity** expresses the opposite to viscosity. The c.g.s. unit for measuring viscosity is the poise, which is defined as follows: A **poise** is that force which, when exerted tangentially on 1 sq. cm. surface of either of two horizontal planes 1 cm. apart, will move one plane at the rate of 1 cm. per sec. in reference to the other plane, the space between the two planes being filled with viscous liquid.

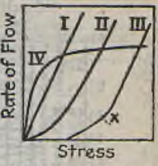


FIG. *C*.—Typical consistency curves. I. True viscosity. II. Pseudoplasticity. III. Plasticity. IV. Inverted plasticity.

$$\mu = \frac{Fs}{V}$$

where F = force applied, dynes.
V = relative velocity of planes, cm. per sec.
s = distance apart, cm.
μ = viscosity, poises.

The centipoise, $\frac{1}{100}$th poise, is in more common use than the poise.

Figure *D* gives a means of transposing some of the ordinarily used units to the centipoise scale.

Referring to Fig. *C*, curve I, it is seen that, for a truly viscous substance or mixture, the rate of flow, or rate of shear (expressed, for instance, in centimeters per second), starts from zero and increases at a constant rate as the stress increases. The rate of flow-stress curve, therefore, is a straight line, and the cotangent of the angle which any of these lines makes with the base, as *AOD*, Fig. *G*, indicates the viscosity. Many substances and mixtures have properties approaching such true viscosity; for example, water, glucose solutions, gasoline, or glycerol (see also Figs. *H* and *I*). These are mostly pure liquids, true solutions, or dilute suspensions. It must be assumed that the molecules, colloidal particles, or larger particles here are loosely related to each other. There is no interlocking or overcrowding.

Plasticity differs from viscosity only in the fact that when a constantly increasing force is imposed on a material or a mixture, a definite yield point must be reached before flow is established. **Mobility** expresses the opposite of plasticity.

The flow of a material which approaches a plastic substance or mixture, is represented by curve III (Fig. *C*). The origin is not zero, for a definite minimum stress, known as the "yield point," is required to start this material flowing, but from that point on it appears to flow as does a viscous material, with constantly increased rate of flow as the stress is increased up to the point where there is a rather sharp upward bend in the curve at x. A theoretically true plastic would not bend at x but would continue in a straight line. The 56 per cent asbestine (magnesium silicate)-in-linseed-oil curve on Fig. *G* is an example of true plasticity.

It may be said that there are few, if any, true plastics, though many materials, such as modeling clay, synthetic resins during molding, cheese, etc., approach this state. It must here be assumed that there is an interlocking arrangement of the molecules, colloidal particles, or larger particles which refuses to change until the yield point is reached, when the interlocking structure breaks down completely.

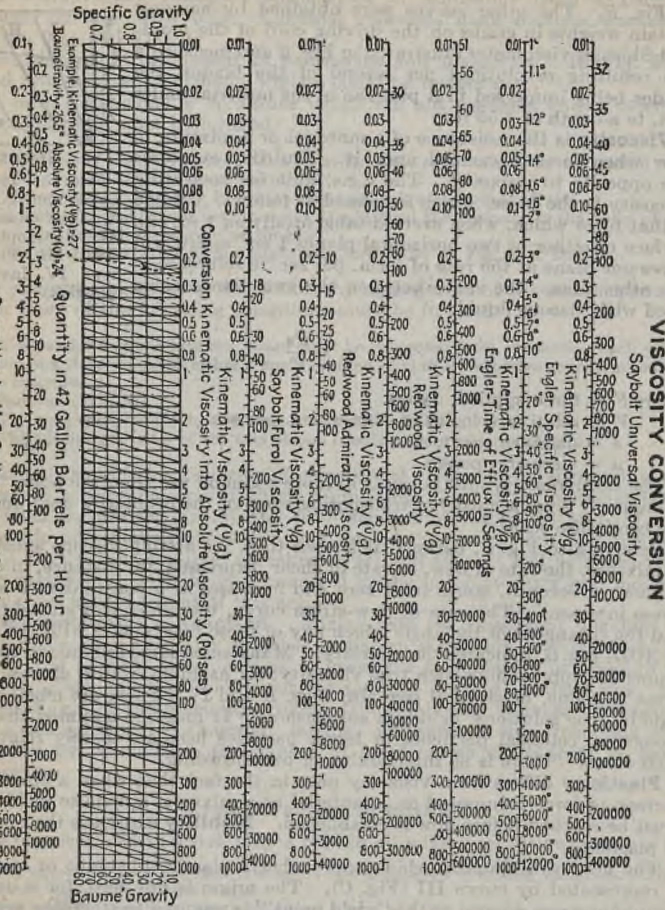


FIG. *D*.

Pseudoplasticity or Thixotropy. Many materials or mixtures have flow curves which are mixtures of the above two types. Curve II on Fig. *C* illustrates so-called pseudoplasticity [Williamson, *Ind. Eng. Chem.*, **21**, 1108 (1929); Williamson and Heckert, *loc. cit.*]. Though starting from the origin, the curve for a pseudoplastic shows a very slow increase in rate of flow (rate of shear) for moderately low shearing stresses. In other words, the lower portion of curve II stays down near the abscissa and in so doing indicates plastic characteristics, though no definite yield point is evident; but as the shearing stress is further increased we begin to get a very sharply climbing curve, which finally becomes a straight line when sufficiently high shearing stress is imposed. In this upper range, viscous flow is evident. In Fig. *G*, the curves for (*a*) 60 per cent lithopone in linseed oil, (*b*) Scott's Emulsion, (*c*) 40 per cent aluminum powder in linseed oil, and (*d*) Hellman's Mayonnaise illustrate this property in varying degrees. Figure *H* is the result of a study made on the same materials in a Gardner mobilometer, employing higher percentages of solids to give pastes of high consistencies. The types of curves in both Fig. *G* and Fig. *H* are much the same, indicating similar properties of mixtures of like materials, regardless of the proportion of each in the mixture, *i.e.*, regardless of whether the mixture is a liquid or a paste.

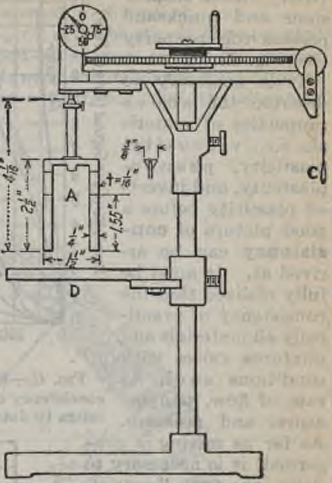


FIG. *E*.—Modified Stormer viscometer.

The relatively high "apparent viscosity" indicated by the lower portion of the curves is known as "false body." Total false body may be expressed as $\frac{(\eta_0 - \eta_\infty)}{\eta_\infty}$ where η_0 = apparent viscosity at zero rate of shear, and η_∞ = apparent viscosity at infinite rate of shear (Williamson, *loc. cit.*). This gradually accelerated rate of flow (shear) as the stress increases is probably due to a progressive orientation of particles to parallelism with the lines of liquid flow and the consequent release of inlocked liquid. Pseudoplasticity is particularly evident in some colloidal dispersions and in some mixtures of solids with liquids, such as certain pyroxylin lacquers, paints, paper-pulp suspensions, heavy gypsum slurries, etc.

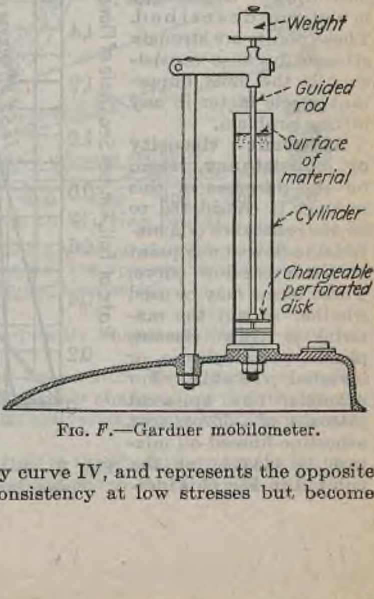


FIG. *F*.—Gardner mobilometer.

Inverted plasticity is illustrated by curve IV, and represents the opposite case. These materials are of thin consistency at low stresses but become increasingly thicker as the force is increased. Heavy starch suspensions and quicksand possess this property (see Fig. *J*).

It is necessary to describe the above properties of materials, *i.e.*, viscosity, plasticity, pseudoplasticity, and inverted plasticity before a good picture of **consistency** can be arrived at. It must be fully realized that the consistency of practically all materials and mixtures varies with conditions such as rate of flow, temperature, and pressure. As far as mixing is concerned, it is necessary to define the term "consistency" in some useful way, and to be practical, a standard means for measuring consistency must be described. These points are strongly stressed because consistency is the most important single factor in any mixing problem.

Apparent viscosity or consistency, then, for the purposes of this section, is considered to be the resistance of a material to flow at any point on its stress-flow curve. These terms may be used whether or not the material is truly viscous, plastic, pseudoplastic, or inverted plastic. For example, the apparent viscosity of a 56 per cent asbestine-linseed-oil mixture (a plastic) at the point *A* in Fig. *G* is indi-

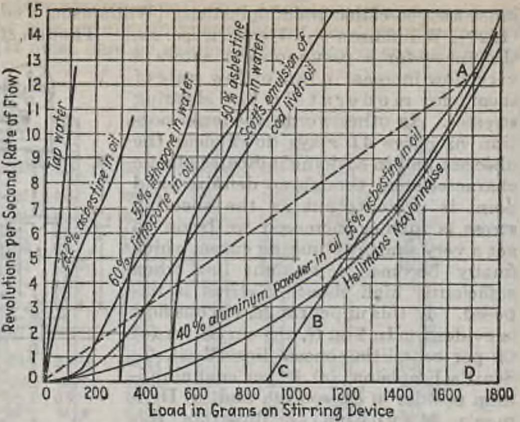


FIG. *G*.—Behavior of some specific materials of moderate consistency on the modified Stormer viscometer. Dotted line refers to data obtained with a 74° Brix glucose solution.

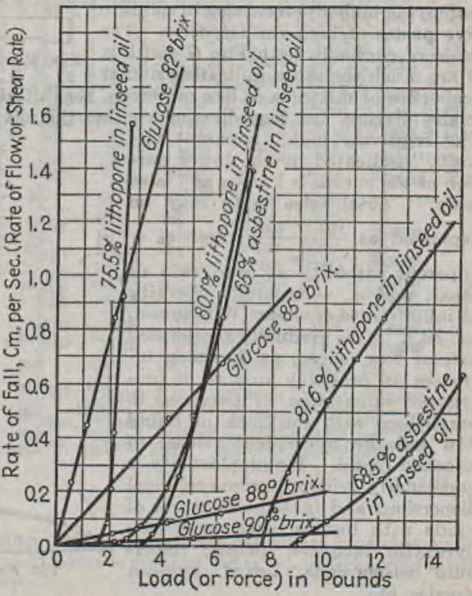


FIG. *H*.—Behavior of some of the same materials in a Fisher-Gardner mobilometer with a four-hole disk.

cated by cot *AOD*, and at point *B* by cot *BOD*. It should also be noted that at point *A* it will have the same apparent viscosity as a 40 per cent aluminum powder in linseed oil (a pseudoplastic), and as a 74° Brix glucose solution (a truly viscous material) *OA*, because the lines cross at this point.

However, at point *B*, which represents $2\frac{1}{2}$ r.p.s. of the stirrer, the apparent viscosity of the 56 per cent asbestine-linseed-oil mixture is now far greater than that of the 74° Brix glucose solution *OA*, when stirred at the same number of r.p.s., since cot *BOD* is far greater than cot *AOD*. Also, the apparent viscosity of the 40 per cent aluminum-powder linseed-oil mixture now lies between the other two.

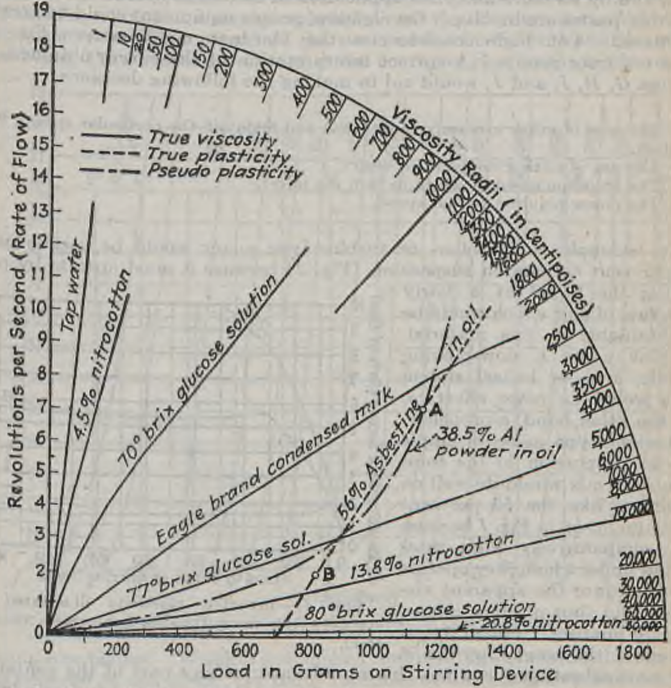


FIG. *I*.—Viscosity in centipoises of several truly viscous materials, one true plastic (55 per cent asbestine in oil) and one pseudoplastic (38.5 per cent aluminum powder in oil) at varying rates of stirring (modified Stormer viscometer).

Therefore it is seen that at any given temperature the apparent viscosity, or consistency, of a truly viscous material never changes, but that the apparent viscosity of all other types of materials varies widely with variation of stirring speed (rate of flow).

Figure *I* relates variable data of this type to the absolute system so that the apparent viscosity of any material under any given set of mixing conditions may be expressed in centipoises. For example, for all practical purposes, a 13.8 per cent solution of 21 second nitrocotton is found to exhibit true viscosity at all loads and all rates of flow, this viscosity being about 10,500 centipoises. As another example, the point *A* on Fig. *I* is seen to be about 2300 centipoises, and point *B* is 9400 centipoises. Therefore this shows a great increase of consistency of 56 per cent asbestine as the stirring speed drops from 7.1 r.p.s. to 2 r.p.s.

The modified Stormer viscometer has been very successfully used in the past, and by its more universal application in mixing problems where liquids and thin pastes are involved, the choice of proper equipment could be greatly facilitated. For high consistencies the Gardner mobilometer (Fig. *F*) gives accurate results. A correct interpretation of the curves obtained, as in Figs. *G*, *H*, *I*, and *J*, would aid in making the following decisions:

1. The type of mixer necessary to establish and maintain the particular type of flow required.
2. The size of mixing elements necessary.
3. The optimum speed at which to turn the mixer.
4. The power required at that speed.

For example, a propeller- or turbine-type mixer would be useless on a 42 per cent cornstarch suspension (Fig. *J*) because it must operate by propelling the liquid at a fairly high rate of flow which would be unobtainable in this material. In this case a slow-moving paddle, arm, or helical ribbon type would be more effective. On the other hand, a propeller- or turbine-type mixer of ample size in proportion to the total volume of mix would do well on materials like the 56 per cent asbestine-in-oil in Fig. *I* because the comparatively high rates of flow under which they operate would reduce the apparent viscosity and thus produce a freer flowing mixture. For practical purposes, the point *C* on the 56 per cent asbestine curve has this significance: at any part of the container where a force (per unit area) corresponding to this point, or any lesser force, is exerted there will be *no motion* of the material; hence mixing will cease.

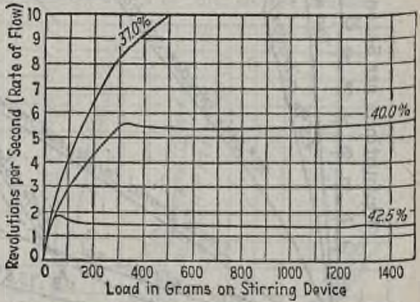


FIG. *J*.—Inverted plasticity illustrated by starch suspensions (modified Stormer viscometer).

Other comments on Figs. *G*, *H*, *I*, and *J*:

It is significant to note the strong plastic tendency of lithopone in water as compared with lithopone in oil.

It is also significant to note the much higher yield point of 56 per cent asbestine as compared to 60 per cent lithopone in the same grade of linseed oil.

All curves shown are drawn in accordance with the definition of viscosity, but for "eye" comparisons of this kind the abscissa should really be the "grams weight."

Using different coordinates, higher consistency materials can be pictured more accurately as shown in Fig. *H*. The grade of materials used in the preparation of these curves may be identified as follows:

Nitrocellulose solutions from 21-second cotton	Hercules Powder Co.
Cornstarch	Corn Products Refining Co.
Glucose	Corn Products Refining Co.
Lithopone	Krebs Pigment & Color Corp.
Asbestine	Commercial
Linseed oil	Spencer Kellogg Co.

Figure *K* illustrates a practical application of the modified Stormer viscometer in the testing of paint consistency by referring to standard curves prepared by testing known mixtures. The paint to be tested is preferably contained in a pint can and is thoroughly stirred to make sure that all pigment is in suspension. It is placed on the stand of the instrument and the stand raised until the surface of the paint in the container rises to the marks on the spatula-like blades. Weights are applied and readings taken until a reading falls between 24 and 36 sec. per 100 revolutions of the spatulas. This reading is then spotted on the chart and translated into consistency by dropping vertically to the horizontal scale. The consistency is merely a superimposed arbitrary scale in which 10 consistency represents a medium-bodied paint, 9K to 3K progressively thicker paints, and 9N to 5N progressively thinner paints. The usual types of ready-mixed paints seldom run thicker than about 6K or thinner than about 6N. The instrument does not function with pastes thicker than about 3K.

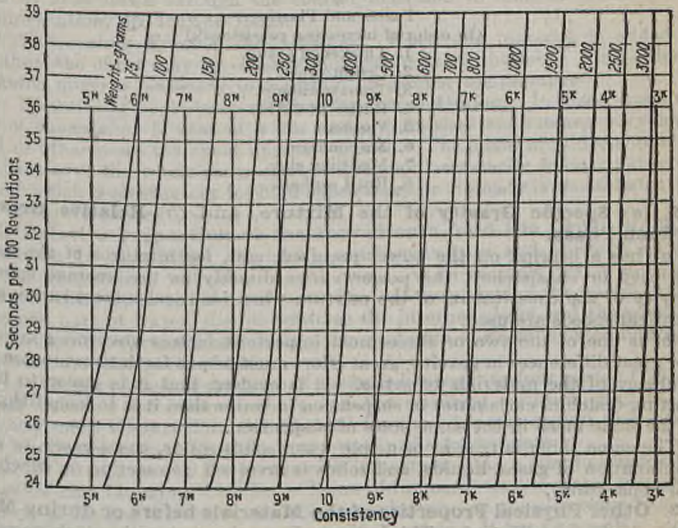


Fig. *K*.—Standard curves for use in classifying paints (modified Stormer viscometer).

The authors are greatly indebted to W. W. Heckert of E. I. duPont de Nemours & Co. for the data from which the curves in Figs. *G*, *I*, and *J* were prepared, to E. S. Steinbring of the same company for the data pertaining to Fig. *K*, and to O. F. Redd, Technical Director of The Patterson Foundry and Machine Co. for the Gardner mobilometer data shown in Fig. *H*.

Examples of Degrees of Consistency. The following tables express ranges of consistency in terms of materials with which everyone is familiar. (Consider them taken at low rates of shear in all cases.)

Liquids at 68°F.	Approximate Viscosity in Centipoises
1. Ether	0.1
2. Water	1.0
3. Kerosene	10
4. Medium motor oil, S.A.E. 10	100
5. Glycerin or castor oil	1,000
6. Blue Label Karo corn syrup	10,000
7. Molasses	100,000
8. Confectioners' glucose	1,000,000 (or more)

Pastes and Plastics
(In order of increasing consistency)

1. Thickened gravy.
2. Cream sauce.
3. Tomato catsup.
4. Butter at 72°F.
5. Vaseline.
6. Mayonnaise.
7. Modeling clay.
8. Road asphalt.

2. (*a*) Specific Gravity of the Mixture, and (*b*) Relative Gravity of Each Phase.

(*a*) has a bearing on the power required, and, for mixtures of the same viscosity or consistency, the power varies directly as the average specific gravity of the constituents of the mixture when identical mixers running at identical speeds are used.

(*b*) is one of the two or three most important factors governing mixing. For great differences in gravity, great effort must be put forth to overcome the tendency of the materials to settle. It is evident that it is easier to hold whiting (calcium carbonate) in suspension in water than it is to keep litharge of the same mesh in the same state of suspension.

The same thing is true in principle when solids-solids, gases-gases, or any combination of gases, liquids, and solids is involved (see section on Mechanical Separation).

3. Other Physical Properties of the Materials before or during Mixing. *a. Ease of Wetting.* This refers only to the ease with which powders are wetted when they are being mixed into a liquid, a pasty, or a plastic mass. It is much easier to mix clay into water than to mix zinc stearate into water. It has been shown that solids are more easily wetted by the liquids which are closest to them in chemical structure. For example, zinc stearate would be much more easily wetted by alcohol than by water, because the stearate and the alcohol are both organic compounds. In zinc stearate the long aliphatic chain far overbalances the inorganic zinc with respect to facility of wetting.

One of the most difficult substances to wet thoroughly with water or even with oil is carbon black. This material resists wetting because of the tremendous surface of its particles, which consequently occlude much air. This air must be forced out before the particles can be thoroughly wetted.

Adhesiveness, or stickiness of solid particles, also adds to the difficulty of wetting these particles. Often a very dry powder which is difficult to wet may be incorporated by lowering the surface tension of the liquid.

b. Surface Tension. This greatly influences the particle size and permanence of emulsions, as well as the bubble size of gas dispersions in liquids. One of the chief purposes of emulsifying agents is to alter the surface tension of one or more of the liquid phases (see under Dispersion, p. 1531).

c. Particle Size. This applies only to solids, and their mixture with each other or with liquids. Obviously a fine particle size will give a smoother finished product. Also, a mixer which will prevent particles from aggregating and will break up preformed aggregates will give a smoother finished product. In dissolving, fine particles will go into solution more rapidly. Then, again, it is more difficult to keep large particles in suspension in liquids because of their greater tendency to settle. On the other hand, exactly the opposite is true in mixing solids with solids, for in this case the finer particles always tend to filter down through the coarser ones and to seek the bottom (see Sedimentation, pp. 1619 to 1652).

d. Temperature Effect of the Addition. When one material is added to another, the effect may be either endothermic or exothermic. Very often a jacketed mixer is necessary to maintain the proper temperature.

e. Variation of Consistency or Viscosity during Mixing. In many cases the initial consistency is that of water and the finished consistency very high, and in other cases the exact opposite is true. The mixer provided should be efficient over the whole range, and it does not necessarily follow that every mixer which is satisfactory for high consistency or viscosity is also satisfactory for the lower range.

4. Relative Proportion of the Ingredients, and the Order of Their Addition to the Mix. Obviously exact quantities and methods cannot be outlined, as they will differ in each case. As an example, water and kerosene oil when separate have low viscosity. If four parts of the oil are beaten into one part of water, the oil becomes the internal phase and the resulting emulsion will be creamy and thick. The one part of water is extending itself to cover the four parts of oil.

On the other hand, if one part of water is beaten into four parts of oil, the water becomes the internal phase and the resulting emulsion may be but slightly more viscous than the oil itself for the oil is not spread over nearly the interfacial surface that the water was in the former case.

Order of addition to the mix is also important. As a simple illustration, consider clay blunging, *i.e.*, the addition of lumps of clay to water to make a slurry, or slip, in the cement or ceramic industry. If the clay were first put in the container, and the water then added, mixing, if not impossible, would at least require excessive power through the thick stages. Therefore, it obviously is best to start with the water and add the clay while stirring. In other cases the procedure is equally important but not so obvious.

TYPES OF MIXERS

In the proper design of mixers, not only the mixing element but also the shape of the container must be considered. A very fine mixing element in the wrong vessel may be utterly useless. Furthermore, the exact result to be attained should be kept in mind so that ample mixing may be provided to obtain that result with a large factor of safety. Usually the additional cost required by this extra provision is trifling compared with the cost of all the equipment involved in a process.

Since mixing occupies a place at the very heart of the process, it is important to do it well. A properly designed mixer may avert a bottleneck in the plant.

The variety of devices used for mixing is extremely large, and many of them have no claims to distinction. Before mixing technology can advance very far, it will be necessary to recognize certain fundamental forms around which our studies and our knowledge may be built. This, of course, does not preclude the future development of new and better forms, but it does give a basis for a certain amount of standardization which is now vitally needed.

Mixers may be grouped under five primary classifications: (*A*) flow mixers; (*B*) paddle or arm mixers; (*C*) propeller or helical mixers; (*D*) turbine or centrifugal-impeller mixers; (*E*) a few miscellaneous types. These may, in turn, be divided so that about 40 truly useful and practical types will cover the entire field of mixing.

A. Flow Mixers

The materials are practically always pumped through this type of mixer and the mixing effect is produced by interference with the flow. They are used only in continuous or circulating systems for the thorough mixing of miscible fluids. They are rarely used for the mixing of two phases where extreme intimacy is desired. The word "turbulence" does not necessarily imply satisfactory mixing.

1. Jet mixers, such as oxyhydrogen torches, rely on the impingment of a jet against another jet, usually with both jets fed under pressure. This mixer is sometimes used for liquid mixing but finds its greatest application in the mixing of combustible gases just before ignition.

FIG. 1.—Jet mixers.

2. Injectors consist essentially of a main pipe and an auxiliary pipe, jet, nozzle, tube, or orifice, through which a second ingredient is injected into the main stream. This simple and inexpensive type of mixer is widely used for the mixing in any proportions of gases with gases, gases with liquids, and liquids with liquids. Bunsen burners, oil burners, spray guns, cement guns, carburetors, atomizers, and nozzle mixers (see Fig. 2) for the mixing of immiscible liquids, are all examples of this type. Either gas or liquid may be the main ingredient. In some cases the velocity of flow in the main pipe induces a flow of material in the auxiliary pipe. In other cases, material is fed through the auxiliary pipe under sufficiently high pressure and velocity to cause the flow through the main pipe. This may be material recirculated from the tank itself by means of an outside pump. A requisite of rapid, thorough mixing in this type is that the mass velocity in the auxiliary stream be considerably higher than in the main stream. Chilton and Genereaux [*Trans. Am. Inst. Chem. Engrs.*, **25**, 102–122 (1930)] found that when mixing two gases with this type of mixer, good mixing can be obtained by making the mass velocity of the added stream two to three times that in the main stream.

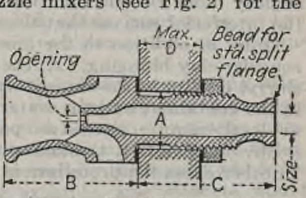


FIG. 2.—The injector mixer.

3. Orifice-columns or **turbulence mixers,** largely used for the continuous treating of petroleum distillates, may be like *A* in Fig. 3, which shows a simple orifice column, or like *B*, which shows a Duriron nozzle carefully designed to give maximum turbulence. They rely for their action on the translation of pressure into turbulent velocity, finding many applications where viscosity is low enough to allow reactions to be completed with the very short holding time available. Both types are very simple to install.

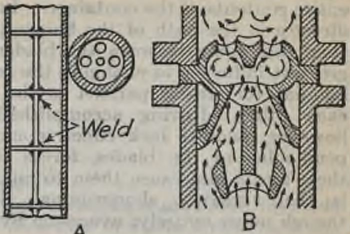


FIG. 3.—Baffle-plate and orifice-column mixers.

4. **Circulating mixing systems,** such as the air lift, "vomit" tubes, long draft tubes, and outside circulating pumps, are usually used to produce a slow turnover of the contents of large tanks by means of comparatively small mixing units. In practically all of these circulating types a very small proportion of the material is being agitated at one time, making them unsuitable where continued intimacy of mix is desired. They are never useful where rapid, thorough mixing is required. Other materials, such as gases, liquids, or slurries, may be introduced in the risèr or pump to ensure preliminary absorption or mixing before being discharged into the main tank.

FIG. 4.—Circulating mixer.

Circulating systems are also used for blending large amounts of solids, usually in excess of 1000 cu. ft. The simplest form consists of two or more bins feeding through automatically regulated feed valves to a belt or conveyor. The material is then elevated and redistributed equally to the bins at the top. The operation is continued until the required amount of blending has resulted.

5. **Centrifugal pumps** without recirculation are sometimes used to mix liquids which have been previously proportioned and they are often useful where blending alone is desired. The "holding" time (or detention time) is usually less than a second and this is only sufficient for instantaneous reactions between immiscible materials.

6. **Spray and packed towers,** while used most commonly for the absorption of a pure gas in a liquid or for the removal of some part of mixed gases, are also finding increased application in removing a constituent of a liquid mixture by means of an immiscible liquid of higher or lower specific gravity. Countercurrent operation is the rule for this type of equipment, which has much to do with its success in many applications.

Packed towers are not desirable where there is any tendency to form a precipitate, for the cleaning problem is usually serious.

FIG. 5.—Centrifugal pump.

B. Paddle or Arm Mixers*

This is probably the oldest type of mixer and consists essentially of one or more blades, horizontal, vertical, or diagonal, fastened to a horizontal, vertical, or diagonal shaft (axis) and rotated axially (though not always

---

* Some of the material in this section prepared by H. W. Bellas.

centrally) within the container. Thus the material to be mixed is actually pushed, or carried, around in a circular path. In thin liquids, paddles always impart a swirling motion to the entire contents of the container. In all cases, that material directly in the path of the blades is always pushed faster than that lying between the blades. This factor has the greatest influence in changing the relationship of successive laminae (or strata) parallel to the blades with respect to each other. Having accomplished this important step, however, paddles lack effective means of producing, perpendicular to the blades, forces that would cut through these strata and cause them to mix with each other. This is their greatest shortcoming. Stratification is largely, though never entirely, overcome by tilting the blades (Fig. 9*b*).

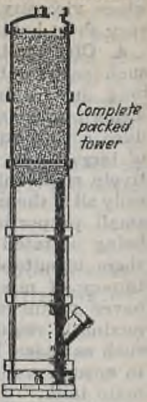


Fig. 6.—Absorption tower.

Paddle mixers, or arm mixers, are more widely used than any other type, because (1) they are oldest, best known, and first to be thought of; (2) often they can be homemade; (3) the first cost is usually quite low; (4) above all, on many kinds of work they are entirely satisfactory. For instance, for the mixing, or kneading, of heavy pastes or plastics (or doughs) the arm type (Fig. 17) is indispensable. However, where stratification may easily occur, as in the suspending of fairly heavy solids in light liquids or the mixing of light pastes or liquids of considerable viscosity, a paddle mixer, no matter how carefully it is designed, is comparatively inefficient both as to power consumed and as to quality of results.

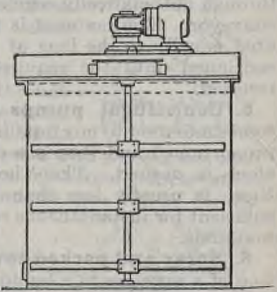


Fig. 7.—Simple paddle mixer.

7. Straight Arm or Blade Paddle Mixer. This is the most common form of mixer and may be either horizontal or vertical. The blades may be either flat or tilted to produce an up or down thrust on the liquid. It is worth noting that in the latter case the result is more nearly that of a propeller than a paddle. Badger, Wood, and Whittemore [*Chem. & Met. Eng.*, **27**, 1176 (1922)] experimented with blades tilted at 45 deg. and found that under the best conditions this mixer took three times as long as and 25 per cent more power than a propeller required to produce the same degree and speed of mixing in the same container (Figs. *Q* and *R*, p. 1560). Nevertheless, it was justly concluded that this paddle type was entirely satisfactory in many cases where the requirements were not too severe.

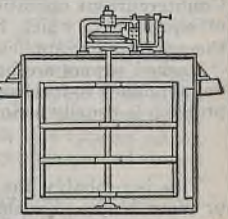


Fig. 8.—Gate mixer.

In the case of cylindrical tanks with unimpeded swirl, a prediction of the power input to flat-paddle agitators can be made by the use of the data of White and Sumerford [*Chem. & Met. Eng.*, **43**, 370–371 (1936)]. In the higher ranges of paddle size and/or speed, White found the power input to be related to the paddle dimensions, speed, and properties of the liquid by the following equation:

$$P = cL^3sN^3D^{1.1}W^{0.3}H^{0.6} \tag{1}$$

where P = power input, h.p.
c = power coefficient, non-dimensional.
D = vessel diameter, ft.
H = liquid depth, ft.
L = paddle length, ft.
N = paddle speed, r.p.s.
s = liquid density, lb. per cu. ft.
W = paddle width, ft.
Also z = absolute viscosity, lb. per ft. sec.

The power coefficient c they found to be related to a modified Reynolds number, just as the friction factor in pipe-line flow of fluids is related to the true Reynolds number. Figure *L* below is a plot of c *vs.* the modified Reynolds number, L^2Ns/z. This correlation they found to hold very well for tanks without baffles, having paddles with a length one-third or more of the tank diameter and with a width less than one-sixth the length. The shaft must be on the center line of the tank and the depth of liquid not more than 50 per cent greater than the tank diameter.

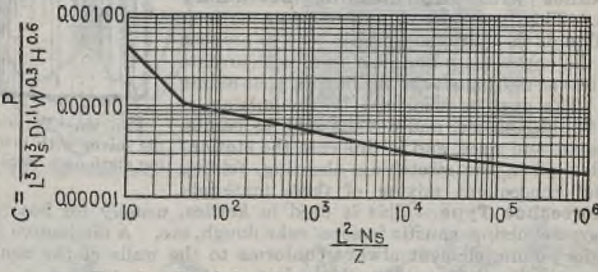


FIG. *L*.—Graph for determining power coefficient.

To find the power input to a given paddle agitator at a given speed, the modified Reynolds number must first be determined. By reference to Fig. *L* above, the power coefficient c for the system is determined, and from it and Eq. (1) the power input P is calculated.

Other correlations of power input have been made by Hixson and Luedeke [*Ind. Eng. Chem.*, **29**, 927–933 (1937)] and Büche [*Zeit. Ver. deut. Ing.*, **81**, 1065–1069 (1937)].

In designing a paddle agitator for a given purpose, a procedure is outlined by Büche (*loc. cit.*). The shape and speed of the agitator are first determined by small scale tests, and the modified Reynolds number $(L_L)^2N_Ls/z$ for the small tank is calculated. The plant-size vessel and agitator are then made geometrically similar to the laboratory model, and a speed N_P is chosen dependent on the laboratory Reynolds number and the linear scale factor, as follows:

$(L_L)^2N_Ls/z$	N_P/N_L
<1.0	1.0
$1.0 - 2000$	$(L_L/L_P)^{0.56}$
>2000	$(L_L/L_P)^{0.67}$

The Reynolds number in the plant equipment can then be calculated and the power input predicted by the use of Fig. *L* and Eq. (1). The basis for this method is Büche's finding that degree of agitation in similar systems is dependent upon power input per unit volume of liquid. Brothman and Kaplan [*Chem. & Met. Eng.*, **46**, 633–636, 639 (1939)] have also found that under like conditions the degree of mixing is related to the power input per unit volume and have plotted it *vs.* a "shear index," which is a measure of the work output of the paddle per unit input.

8. Gate Type. This type covers many designs of which Fig. 8 is an example. It is questionable whether or not a combination of horizontal, vertical, and sometimes diagonal blades improves the mixing, but it is very often used where structural strength is desired.

9. Paddles with Intermeshing Stationary Fingers. This type may be horizontal or vertical. In thin liquids the stationary fingers tend to prevent swirl of the entire mass and also to direct currents more or less at right angles to the fingers, thus aiding mixing. This type is also used in the mixing of heavier liquids, pastes, and doughs such as paints, starch paste, and sizes, and in this case the stationary blades aid in the stretching, shearing, folding over, and consequent mixing of these materials.

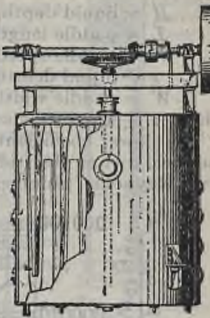


FIG. 9*a*.—Vertical paddle mixer with intermeshing stationary fingers.

10. Horseshoe Type. This is used in kettles, usually for heavy duty such as grease mixing, caustic fusions, cake dough, etc. A distinctive feature is that the mixing element always conforms to the walls of the container, sweeping or actually scraping them free from pasty or solid material that might otherwise cake upon them. It is particularly important to prevent this caking in mixtures which burn when locally overheated, or in cases where the walls must be kept clean to permit good heat transfer. Therefore this type (as well as other types mentioned later) is widely used in jacketed kettles or on furnace settings where the mix within the kettle is thick.

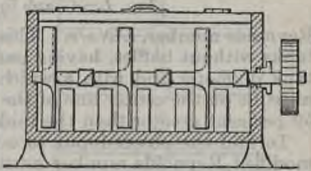


FIG. 9*b*.—Intermeshing fingers in horizontal tank.

11. Traveling Paddle. This mechanism is used for very large batches of slurries, such as cement slurry or paper pulp. The task is usually a matter of maintaining the material in suspension, and the sizes of the vats used are seldom less than 40 ft. long by 25 ft. wide by 20 ft. high and run up to more than 150 ft. long by 40 ft. wide by 30 ft. high. This type is also adapted to circular tanks.

A traveling mechanism, either paddle propeller or turbine, is the only one which will operate successfully on large quantities of slurries and pulps.

Types 12, 13, 14, and 17 which follow represent double-motion paddle or arm mixers and are used for pastes and plastics. In these cases there is little flow produced, and the double motion produces an extra amount of direct shearing, kneading, and folding action on the mass.

12. Rotating Pans with Offset Blades. *a*. The can revolves on a turntable and the offset paddles within it also revolve. It is in common use for mixing small batches of thick paint and ink pastes.

b. This machine is a very efficient modification of the same sort of double motion, also with offset, rotating blades and rotating pan. The plow-like blades, mounted with springs, fit the bottom of the pan very closely, and stationary scrapers help to feed the blades and also scrape the sides of the pan clean. It is used for mixing pastes or plastics, such as putty, for concrete mixing, and also for intimately mixing dry powdered or granular solids. For such materials it is beyond doubt faster than any other type. Where a kneading or a smearing action is desirable, as in putty making, mullers as in the pan mixer (see type 40) are used.

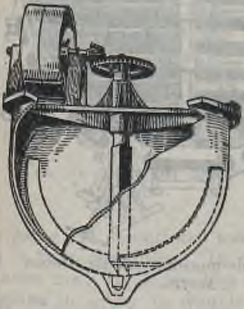


FIG. 10.—Horseshoe mixer.

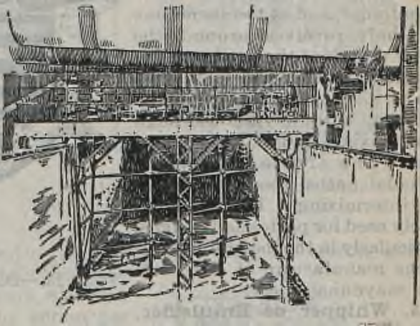


FIG. 11.—Traveling paddle mixer.

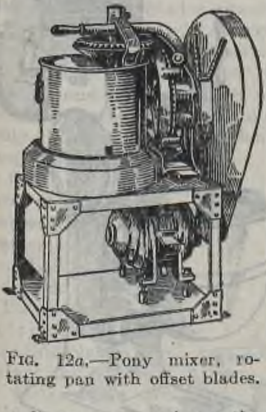


FIG. 12*a*.—Pony mixer, rotating pan with offset blades.


FIG. 12*b*.—Rotating pan mixer.

13. Double-motion Paddle. This type is used extensively for pasty materials such as adhesives, greases, and cosmetics and for ice-cream freezing. Two sets of blades rotate in opposite directions. The outer sweep is often provided with scraper blades which keep the container wall clean. This results in improved heat transfer, making it possible to heat or to cool batches in as little as one-quarter of the time necessary in vessels equipped with non-scraping agitators. This mixing action is probably not to be excelled for the type of work mentioned above.

14. Double-motion Paddle. This is not unrelated to the traveling paddle of type 11. A paddle rotates on a shaft which is located off center in the kettle or container, and at the same time the shaft revolves around the center or axis of the kettle. This planetary motion causes the action to visit every portion of the kettle in turn, giving thorough local mixing and carrying particles forward in overlapping cycloidal paths, thereby producing intermixing. This type is widely used for pastes and doughs, particularly in the food industries, in the manufacture of cake batters, mayonnaise, etc.

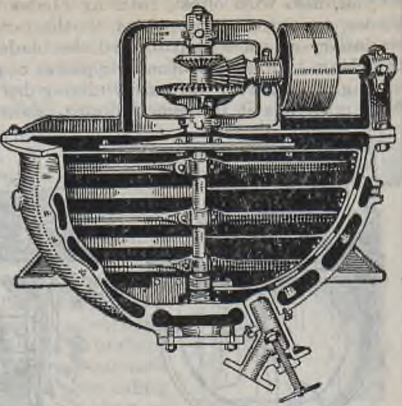


FIG. 13.—Double-motion paddle mixer in kettle.

15. Whipper or Emulsifier. A familiar example of this type is an egg beater. The device, whatever its form, is always run at high speed, and due to the actual **beating** together of the two fluids, a fine state of division, or emulsification, is produced. It is used extensively for the preparation of whipped cream (liquid and gas), mayonnaise (immiscible liquids), etc.

16. Air-lift Agitator. Air forces the slurry up the central tube to the overhead rotating distributor pipe. The slurry flows out and is distributed over the surface. The paddle at the bottom is also equipped with an air pipe along its entire length for freeing it when stuck in settled slurry.

This type is useful for maintaining large masses of slurry in suspension. The mixing action, if any is desired, is very slow. Sizes range as a rule from 20 ft. diameter by 12 ft. high, up to twenty times that volume.

17. The kneader, with two arms or blades rotating in opposite directions in a container with a divided trough or saddle, is used for mixing thick, plastic, gummy, doughy masses. Of all 40 types of mixers, type 17 is most nearly indispensable to its particular field of materials. Heavy "sigma" blades, slightly helical, rotating oppositely across the trough division, simultaneously effect transportation, kneading, tearing, stretching, folding. (See pp. 1572 to 1574 for fuller description of the action.) These blades

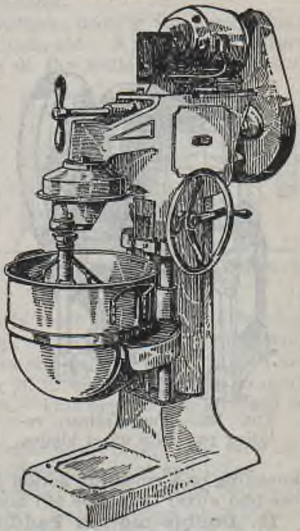


FIG. 14.—Double-motion paddle mixer (baker's type).

FIG. 15.—Whipper or emulsifier.

FIG. 16.—Air-lift agitator with scrapers.

or arms are sometimes toothed or serrated to intensify the tearing action, as in pulp shredders. In certain cases the blades may be more like figure 8's than sigmas in order to double the action per revolution. The two blades are often made so that they overlap to produce a better interchange of material from one blade and trough to the other. On the other hand, non-overlapping blades running at differential speeds may be employed, and in this case they are usually adjusted with slight clearance so that they clean each other in passing and also produce more positive shear, like scissors. For fine work, especially that which involves heat transfer, the blades are machine-fitted to the trough with as little as 0.001-in. clearance at working temperatures. For greater heat transfer, hollow blades are used.

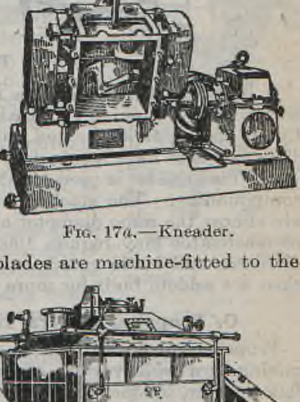


FIG. 17*a*.—Kneader.

Because of the difficulty of discharging plastic materials, the majority of kneaders are made to empty by tilting, the tilt being either hand- or power-operated. Other machines empty through a large door at the side of one trough. In cases where the mixture is sufficiently

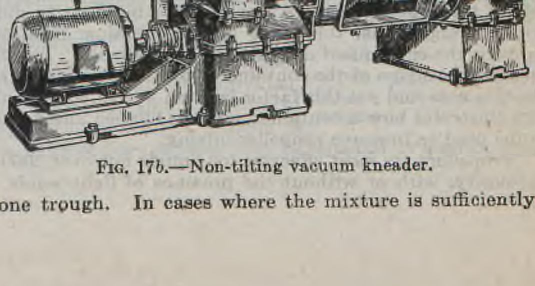


FIG. 17*b*.—Non-tilting vacuum kneader.

fluid or granular at the end of the process, valves in the bottom of each trough are best adapted for easy and rapid discharge.

a. This is a kneader for general purposes, bread dough representing the mean consistency. A 100-gal. machine requires from 5 to 80 h.p. A typical trough is 38 in. long by 32 in. wide by 28 in. high. The sigma blade shafts and the blades themselves vary from 3 to 7 in. diameter as power increases. Bread dough requires about 15 h.p. and a heavy asphalt-asbestos mastic about 60 h.p. Blade speeds of 20 to 40 r.p.m. are common. Where a differential speed is used, a ratio of 3:2 or even 6:7 is often employed. Machines of this class are not often built for more than 1000 gal.

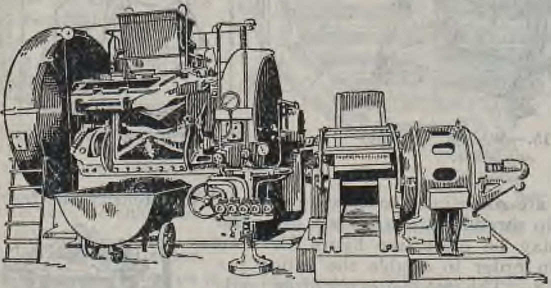


Fig. 17*c*.—Masticator.

b. This is a non-tilting type for vacuum operation. It is especially useful where evaporation is required, as in the preparation of powdered milk. Cored blades, steam-heated and machine-fitted to the trough are essential. This may also have a side door instead of bottom valves for discharge.

c. This kneader is used for the heaviest work, such as dispersion in rubber compounding. The machine is shown in tilted position. The blade shafts are almost the same diameter as the blades themselves. A 100-gal. disperser or masticator may require 100 to 200 h.p. The shafts may be 10 to 12 in. diameter and blade speeds up to 30 r.p.m. are common. Machines of this class are seldom built for more than 100 gal. capacity.

C. Propeller Mixers, Including a Few of the Helical Type

Propeller mixers furnish an inexpensive, simple, and compact means for mixing in a wide variety of cases. Their mixing action follows from the fact that the revolving helical blades constantly push forward what is to all intents and purposes a continuous cylinder of material, although "slip" induces currents which modify considerably this cylindrical form. Since the propeller causes a cylinder of material to move in a straight line, the shape of the container itself will govern the subsequent disposition of this stream. For this reason the shape of the container is particularly important in this case and yet this factor is often neglected. Figure 22 illustrates how a container may be shaped and a draft tube used to improve propeller mixing.

Fig. 18.—Propeller-type fan.

Propellers are most effective for liquids not over 2000 centipoises apparent viscosity, with or without the presence of light solids, though useful up to

4000 centipoises. With heavier solids, in flat-bottom containers especially, some difficulty is experienced in preventing settling, since it is practically impossible to direct the stream from the propeller to all parts of the tank in any simple manner.

The location of propellers within the tank influences the nature of the mixing, and the following types illustrate these various positions.

Figure S makes it possible to calculate the approximate horse-power consumption of any propeller, given size, speed, and pitch.

18. Propeller as a Gas Mixer. A propeller, disk, or fan (practically the same as the ordinary window ventilator fan) is used within a mixing chamber to give circulation and mixing of gases. It is also used for mixing gases under continuous-flow conditions.

FIG. 19.—Propeller mixer, push-pull type. (*Buffalo Foundry & Machine Co.*)

19. Propeller with Vertical Shaft. These are used in combinations of one, two, or more propellers on the shaft. They may all thrust upward or downward to meet special conditions; however, the push-pull combination is usually the most desirable for small tanks.

20. Portable-type Propeller—Clamped to Side. This type is compact and satisfactory and is very useful for portable work; for viscosities up to 300 centipoises the direct-connected, full-motor speed type is correct, but for higher viscosity the geared machine must be used. These units are made in sizes from ⅛ to 5 h.p.

FIG. 20.—Portable-type propeller mixer.

21. Propeller in Side of Tank. This type is usually located non-radially. The motion produced is a swirl which gradually brings the entire tank contents into the influence of the propellers. This motion is most useful for large batches of light liquid, such as gasoline or aqueous solutions, where great speed of mixing is not necessary. In such cases this type gives satisfactory blending up to 200,000 gal. capacity and is one of the best means of mixing very large tanks of light liquids. In these large tanks it is usually desirable to use two or more units at intervals around the periphery.

In the paper industry, propellers on horizontal shafts are used for circulation with incidental mixing of stock in large stock chests, built with one or more mid-feathers. The propeller may be freely installed in one of the aisles, or it may be located in a large hole in a partition placed across the aisle; somewhat like the fan, Fig. 18.

Side propeller agitators vary in size from ½ to 50 h.p., with propellers from 4 to 84 in. A good stuffing box and strong bearings must be provided.

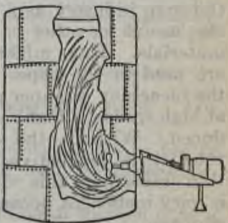


FIG. 21.—Propeller mixer. Side tank installation.

For blending liquids of considerable difference of specific gravity, a high velocity vertical stream is desirable, and for this purpose the propeller may be replaced by a centrifugal impeller.

22. Propeller in Draft Tube. One or more propellers are surrounded by a tube, which usually has small clearance from the propeller tips. The tube serves to guide fluid through the propeller, appreciably overcoming side slippage of currents. Providing the tank is well shaped, a thorough circulation at a rapid rate with consequent uniformity of mixing action takes place. For these reasons this type is probably the most efficient among propeller mixers and is related to type 24. It is used mainly on light or moderately viscous liquids where intimate mixing is desired.

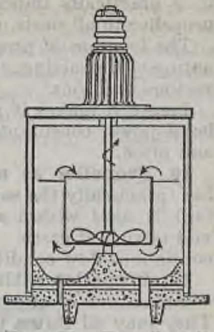


FIG. 22.—Propeller in draft tube.

23. Pug Mill. This machine is indispensable to the ceramic and related industries for securing a thorough mixing of very heavy clay masses, usually on a continuous basis. The unmixed or partially mixed ingredients are fed at one end of a trough or cylinder, usually enclosed to withstand heavy pressure, within which a series of very short and stout paddles are revolving. These paddles, tilted to approximate the form of propellers, transport the mixture gradually to the other end of the trough, cutting and kneading it constantly in transit. It is often discharged by extrusion through one or more holes. The entire operation is known as "pugging." Pug mills may be either vertical or horizontal. Because very heavy slips are handled, the power consumption is high.

24. Soap Crutcher. This consists of a continuous helix in a draft tube which fits closely around the screw. The course of the material is usually upward through the helix and downward on the outside of the tube. Its conveying action is well adapted to pastes of the consistency of soap, and it is universally used in the soap industry as a mixer. It is also useful for other pasty or fibrous materials. Huge mixers of this type are used in the paper industry for the bleaching of paper pulp at 16 to 18 per cent consistency. When operated at high speeds, a high rate of circulation and a vigorous mixing action are produced. Without the draft tube it is ineffective. On thin materials this device is very inefficient because of slippage.

FIG. 23.—Pug mill.

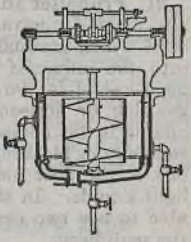


FIG. 24.—Soap crutcher.

FIG. 25.—Ribbon mixer.

25. Ribbon (Double-helical) Mixer. This is a satisfactory type for mixing powders, self-rising flour mixtures, talc, baking powder, etc. It is usually operated at moderate speeds. The mixing element consists of several vertical paddles and two helical ribbons, one a right-hand screw and the other a left-hand screw, so that the material is moved back and forth from one end of the container to the other, and also lifted vertically. The ribbon blades are rather narrow because the mixing of solids requires constant slippage off the blades to promote local turnover (eddies) to prevent packing of the material and to hold the power required to a reasonable figure. This type is also used for the mixing of some moderately thin pastes.

D. Turbines or Centrifugal-impeller Mixers

The turbine mixer is best described as one or more centrifugal pumps working in a tank against practically no back pressure. As is evident from Figs. *M* and 28, material enters the impeller axially through the central opening. The material is accelerated by the vanes and is discharged more or less tangentially from the impeller and at fairly high velocity. The most efficient type of turbine mixer employs a curved stationary deflecting-blade ring, which deflects these tangential currents to a radial direction. The entire direction change from vertical to horizontal and radial is thus accomplished smoothly with the smallest possible loss of kinetic energy, and, as a result, the radial currents are still traveling at high velocity when they reach the remote parts of the container. Thus, Rule 2 (p. 1530) is especially well fulfilled by the turbine type. Rule 1 is also fulfilled, since the discharge from the impeller is scattered along the radii in literally an infinite number of directions and the process is repeated many times a minute. The entire contents of the tank are kept in vigorous and well-directed motion.

FIG. *M*.—A typical turbine mixer.

When two or more impellers are used, the vertical currents are as shown in Fig. 28.

The power required by a turbine mixer is approximately one-thirtieth that required by an outside circulating centrifugal pump delivering the same volume of liquid, and the mixer impeller revolves at a moderate speed. For instance, a 36-in. rotor turns at about 75 r.p.m.

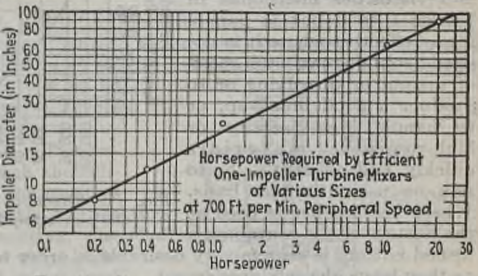


FIG. *N*.—Power requirement for turbine mixer.

Turbine mixers are especially useful for mixing viscous liquids or heavy slurries, for suspending heavy solids, for rapid dissolving, for good dispersions, and for mixing in irregularly shaped containers.

Figures *N*, *O*, and *P* show how the power consumption of turbine mixers varies with changes in size of turbines, peripheral speed, and viscosity of liquid being mixed.

26. Turbine Blower or Centrifugal Fan. This type will mix gases very intimately when the gases are supplied in the desired proportions on continuous work. It is also used for batch mixing of gases, being located inside or outside of the mixing chamber. It will handle large volumes with low power consumption.

27. Simple Turbine Mixer. This type is particularly desirable for the blending of liquids of low or medium viscosity, especially when set off center in the tank. It is also good for low and medium consistency of slurries and for medium consistencies of fibrous materials, such as paper pulp, in suspension.

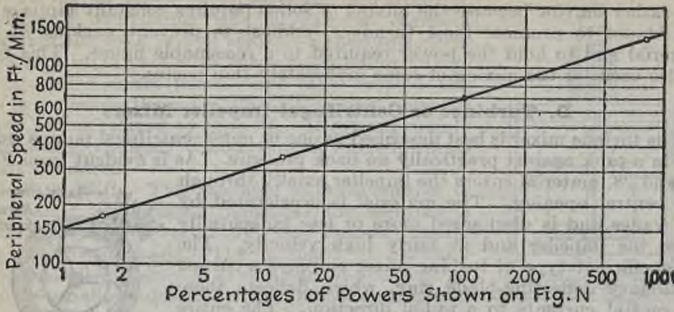


Fig. *O.*

28. Turbine Mixer with Stationary Deflecting Blades. With this type one or more impellers may be used. It is useful for all consistencies and viscosities mentioned in Tables 1 to 5 but is particularly effective where it is desired to obtain maximum speed of mixing or dissolving or to produce a good break-up, or to handle high-viscosity or high-consistency materials quickly and efficiently, or to mix on a continuous basis. Where an impeller is used at the bottom of the container, a dished bottom is always very desirable in order to start the currents upward as they leave the mixing element.

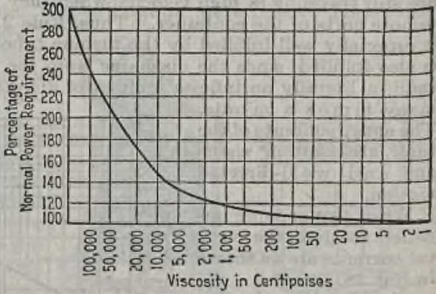


Fig. *P.*—Effect of viscosity of the material on the power requirement of turbine mixers.

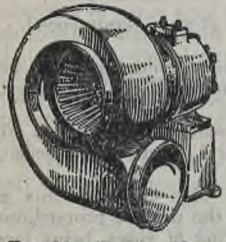


Fig. 26.—Turbine blower.

Fig. 27.—Simple turbine mixer.

Fig. 28.—Turbine mixer with stationary deflecting blades.

Type 28 is better than any other mixer for use in irregularly shaped containers, *i.e.*, rectangular tanks, horizontal cylindrical tanks, etc., because the radial flow penetrates to the extreme corners.

29. Turbo-disperser. This type is a new machine occupying a unique position since it fills a place between simple mixers and homogenizers or colloid mills. It consists of a centrifugal turbine impeller which rotates with a screen or perforated plate interposed between impeller and stationary deflecting blades. The blades come close to the screen, giving an area of high shear in which difficult dispersions are easily accomplished. The high degree of shear, the extruding action, and the high flow, all contribute toward the accomplishment of dispersions and the dissolving of types of material which are difficult to accomplish in simpler types of mixer.

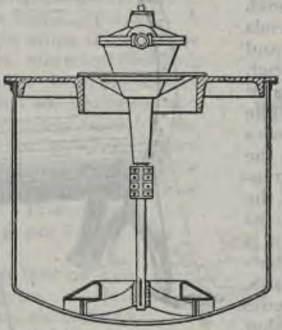


FIG. 29.—Turbo-disperser.

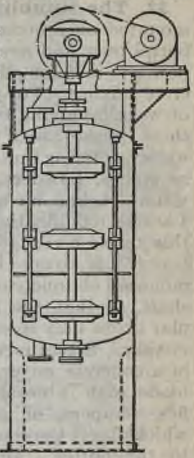


FIG. 30.—Continuous-flow turbo-mixer.

30. The continuous turbo-mixer has all the advantages of type 28 but, owing to the continuous flow and the division of the volume into three compartments in series, smaller equipment is allowable for a given production. The main stream has proportioned into it the solid, liquid, or gas with which it is to be treated and passes progressively from one compartment to the next, thus eliminating short circuiting. The tremendous circulation through the centrifugal impellers assures the greatest contact between the materials, so that the resultant rates of reacting, dissolving, or contacting are very high. The type of impeller and stator used in each stage is determined by the type of work to be done. A 200-gal. size will dissolve 100 tons of salt per day practically to saturation, while a 100-gal. machine is capable of mixing 15,000 gal. asphalt and naphtha per hour.

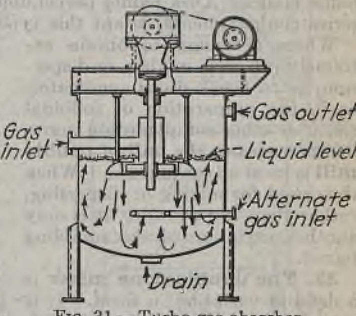


FIG. 31.—Turbo-gas absorber.

31. Turbo-gas absorber or oxidizer is used for promoting contact between gases and liquids. Hydrogenations, oxidations, chlorinations, purifications, etc., are greatly stimulated by the long gas path through the liquid under violent agitation. The constant distortion of the bubbles of the gas together with the continual exchange of liquid at the interface account for the high efficiency of this type. In some applications the gas is self-induced at the surface, while in others pressure gas is fed to the lower of a series of the absorbers on a single shaft.

E. Tumbling Mixers

32. The tumbling barrel is simple but useful. It consists of a barrel, mounted on a horizontal shaft and rotating with the shaft. Tumbling the barrel over and over mixes the contents. It is extensively used for mixing powders and for all concrete mixing. For types of work involving two or three phases with such widely differing materials as stones, powders, and water, it has no equal. Various modifications of this type exist. The barrel is sometimes mounted obliquely on the shaft, so that the irregular throw may speed the mixing. Sometimes, as in a concrete mixer, it is made with internal baffles, scrapers, or plows, which divert the contents to the outlet. Another variation has barrel rotating in one direction and paddle blades turning in the opposite direction. A modification used in the mixing of hair felt, for example, employs longitudinal baffle blades. Disk baffles perpendicular to the shaft, dividing the body into series compartments, adapt this type to continuous operation.

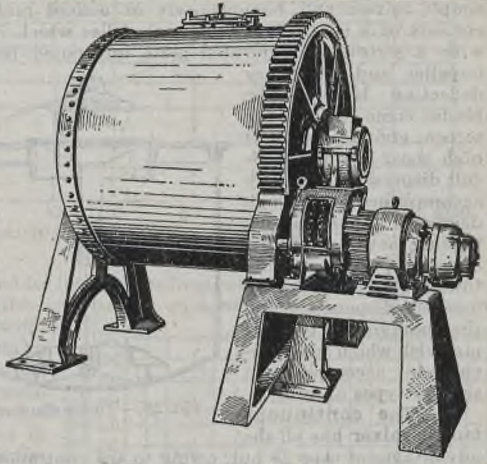


FIG. 32.—Tumbling barrel.

Where necessary to obtain extremely intimate mixing, or dispersion, or to break down aggregates, as in the preparation of colloidal sulfur or other compounding ingredients for latex, the **ball of pebble mill** is most advantageous. When thus used for mixing or dispersing, rather than for grinding, it is only another variation of the tumbling barrel.

33. The double-cone mixer is a definite variation in form. It is used for the rapid blending of solids only. It consists of a cylindrical ring to which are attached two cones, the whole rotating slowly on trunnions, end over end. The inside is usually polished and free from obstructions for easy cleaning. During rotation, the bottom cone is tilted to a point where the angle of repose of the contents is exceeded. The surface layers then roll down toward the opposite cone, followed quickly by the entire mass, which slips rapidly into the other cone, now near the bottom position. Striking

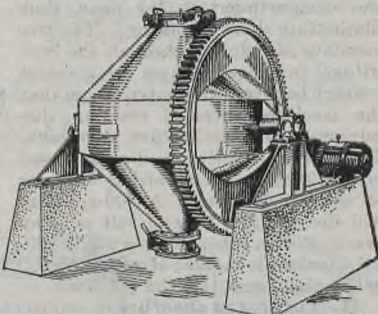


FIG. 33.—Double-cone mixer.

against the conical walls, much material is deflected toward the center and thence upward through the remainder of the mass. Because no two particles take parallel paths and, further, because there is a great difference in the velocities of various particles, homogeneity quickly results. Ten minutes is usually sufficient time for thorough blending of any materials. The mixer is quickly loaded or discharged, a positive-seating, quick-acting, dust-tight discharge valve being provided. A magnetic brake on the drive stops the machine in any position, and an electrical inching mechanism permits it to be brought slowly around to the correct point for charging and discharging. This type is widely used for the mixing of solid granules or powders where speed or great cleanliness is required, *e.g.*, for dry color standardization or for the blending of colored resin products. Power consumption is not over 1.5 h.p. per 1000 lb. of contents.

Fig. 34.—Mushroom mixer.

34. The mushroom mixer is the third distinct tumbling type. A flat, covered bowl is mounted on an inclined shaft and rotated. Three to eight heavy metal balls from 3 to 6 in. diameter are loaded into the mixer with the batch. Their function is to break down aggregates and to produce intimate mixing by shearing. This type is widely used for the standardization of dyestuffs and for the mixing of various pharmaceuticals, botanicals, and organics. It is easily cleaned and must be dust-tight.

F. Miscellaneous Types

35. The colloid mill is used where extremely fine dispersions are required. Most colloid mills are the same in principle, though they may differ in details of construction. As in Fig. 35, the materials to be dispersed are fed between a very rapidly revolving solid rotor and its casing, which it clears by 0.001 in. or less. The rotor may or may not be grooved, and it may or may not be conical. The material is subjected to intense shear and intense centrifugal force, and the combination acts to make excellent dispersions. The material is usually premixed in an ordinary mixer and this coarser dispersion is then reduced by passing it through the mill. Owing to the electric charge imparted

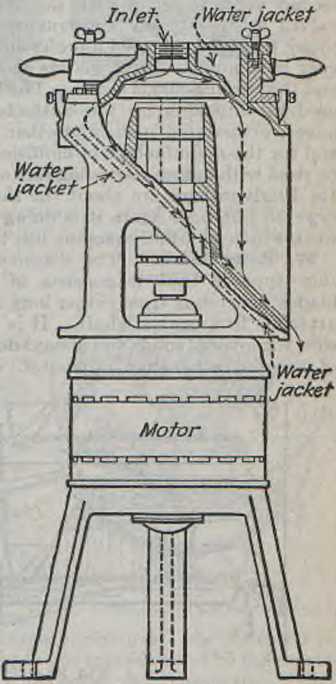


Fig. 35.—Colloid mill.

to the particles, and the small size to which they are reduced, emulsions can usually be made with very little stabilizer. Pigments can be dispersed in oils to the original ground particle size, but it is doubtful if actual grinding takes place. Colloid mills have the advantage of giving continuous flow but the disadvantage of a high first cost, high power requirements, and a heating effect on the material. On some types of work where the maximum degree of dispersion is required, nothing else has replaced them.

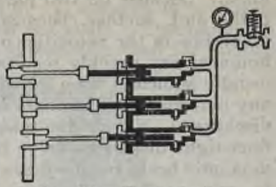


Fig. 36.—Homogenizer.

36. The homogenizer may be described as a positive high-pressure pump in which the pressure is released radially past a disk or valve which is tightly pressed against the end of the discharge pipe by means of a spring. Homogenizing is often done at pressures of 1000 lb. per sq. in. and above. On some products, a finer break-up is obtained by passing the material through a second valve in series with the first valve. The valves are commonly constructed of agate, but today there is evidenced a preference for the use of very hard non-corrosive metals such as Hastelloy and the chrome-nickel steels. The homogenizer is used for breaking up the butterfat in ice-cream mixes, evaporated milk, and other food products, and for the manufacture of emulsions. It cannot be used with materials having any abrasive action. Its disadvantages are about the same as those of type 35 but here again it is doing some types of work which no other machine has been able to do.

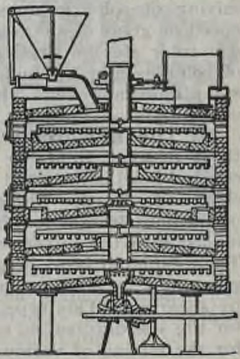


Fig. 37.—Rake mixer (Herreshoff furnace).

37. Rake Mixer. The distinctive feature of this type is that it consists of rake or plow blades depending from rather long arms which are attached to a central shaft. It is used to give a slow turnover of solids and always designed to give good vertical transportation of material rather than horizontal. The particular mixer illustrated in Fig. 37 is the Herreshoff furnace, used for roasting ores, the object being to bring fresh material constantly to the surface and, incidentally, to break up lumps.

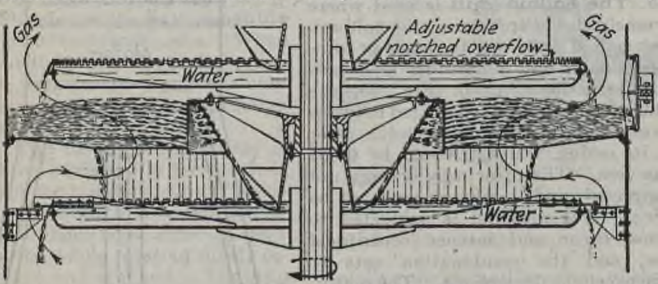


Fig. 38.—The Feld gas scrubber.

Many rake mixers are built with one set of rakes only, rather than the several superimposed sets shown in the figure. A rake mixer with very much longer and narrower vertical blades is used for turning over grain in malting operations.

38. The Feld scrubber or spray type operates by throwing a thin screen of liquid across the path of an oncoming gas for the purpose of removing certain materials from the gas, either to purify the gas or to dissolve it in the liquid. It is built up of several superimposed elements. Each of these elements, as shown in Fig. 38, consists essentially of a tray of liquid, around which the gas passes, and a rotating conical frustum. Liquid travels from the tray up the cone by centrifugal force and is discharged as a spray through which the upflowing gas must pass. Gases passing through this spray are thus brought into excellent contact with the liquid.

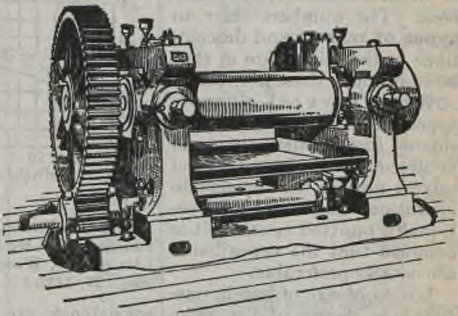


FIG. 39.—Mixing rolls.

39. Mixing Rolls. This type consists of two rolls, usually turning at different speeds, between which the materials to be mixed are passed. A kneading, tearing, stretching, folding, and shearing action is produced. They are used in certain cases where an exceptionally intimate mixture of a solid with a liquid is desired, *e.g.*, in printing-ink manufacture. They are also used for the heaviest types of work in which mixing is possible, *e.g.*, mixing fillers into rubber and blending rubber stocks. The rolls are often corrugated to afford a better grip on the material.

FIG. 40.—Pan mixer.

40. Pan mixer, sometimes known as putty chaser, edge runner, or muller mixer; this type is widely used for the manufacture of putty, *i.e.*, whiting mixed with linseed oil, for clay mixtures, and for other similar operations on plastic and on semidry materials, such as foundry sand. It is also used for the intimate mixing of dry materials, where the breakdown of aggregates or the coating of one solid particle with other solid particles is desired. Built on an ancient grinding principle, having one or more large wheels (or mullers) rolling around in a pan, together with scraper blades or plows, it combines a kneading, grinding, and mixing action, giving thereby very intimate mixtures. As the speed of rotation is usually slow, the power required is not excessive. The rollers are usually steel, though sometimes stone; they are more often heavy than light. Since the mullers have a wide face, there is constant twisting or shear on the line of contact between the muller face and the material next to the bottom of the pan. The scrapers, or knives, rotating with the rollers around the central axis, deflect the material into the path of the rollers and also scrape the sides and bottom of the pan.

In 5 ft. diameter pans the mullers may be 30 in. diameter, and may weigh from 100 to 400 lb. each. There are some putty mixers with 10-ft. pans and 6 ft. diameter mullers.

FITTING THE MIXER TO THE OPERATION

In Tables 1 to 5, mixing problems have been classified according to the materials to be mixed. Every combination of gaseous, liquid, and solid material is considered and the types of mixers best suited to handle the combination are indicated by numbers. The numbers refer to **types of mixers** and descriptions of the mixers are in that section under these numbers.

When several alternate types are given, that type considered least satisfactory will be given first. Many types of mixers now in use for certain operations have been intentionally omitted because it is believed that the types listed are usually preferable.

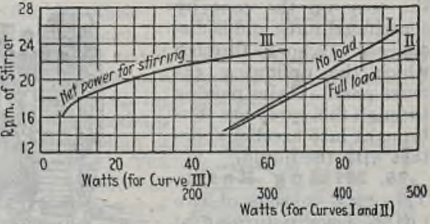


FIG. Q.—Power consumed vs. r.p.m. for 4 ft. 6 in. paddle in a 5- by 5-ft. tank. Low viscosity liquid. (*See Badger, Wood, and Whittemore, Chem. & Met. Eng.*, **27**, 1176.)

Limits of size of batch, particle size, degree of dispersion, consistency, etc., are also given, **although it must be realized that these limits are in no way fixed,** and there are no rigid rules covering any of this work. The limits are empirical but appear to be entirely reasonable in the light of experience.

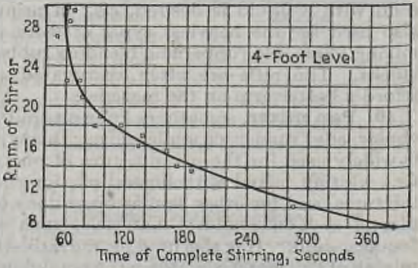


FIG. R.—Effect of r.p.m. on mixing speed. Conditions as in Fig. Q. (*Badger, Wood and Whittemore, Chem. Met. Eng.*, **27**, 1176.)

The general type only can be indicated. For questions in individual cases regarding power, exact size, price, etc., of mixer, the manufacturers should be consulted.

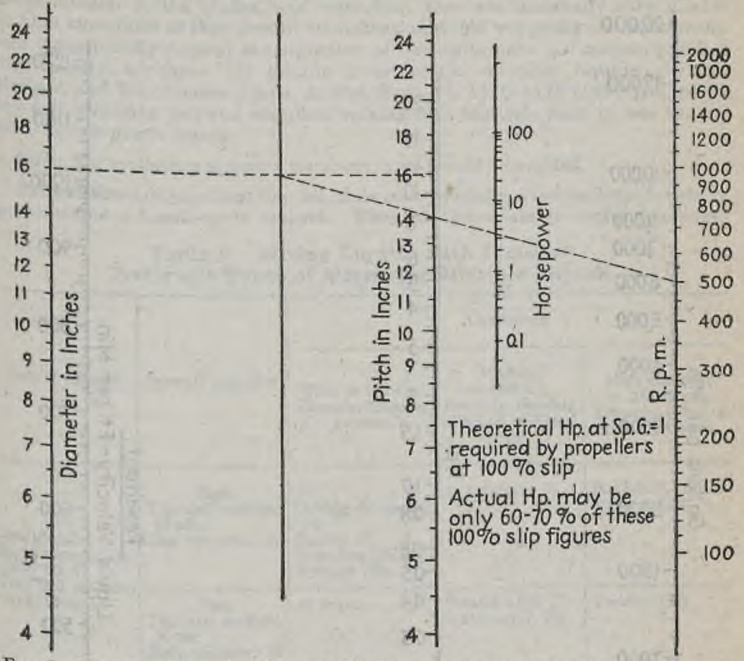
Though it would be desirable, it is impossible to give any satisfactory figures or tabulations of comparative power requirements, as this is determined, not by general cases, but by each individual case. Figures *N* (p. 1553), *O* and *P* (p. 1554), *Q* (above), *R*, *S*, and *T* offer some general information along these lines.

In using Fig. *S* to determine the proper size motor for a propeller installation the indicated horse power can usually be multiplied by the factor 0.6 to get the actual power required. This represents the pumping efficiency of the propeller and will be smaller with the two-blade and larger with the four-blade type.

1. Mixing Gases with Gases

Mixers of jet (1), injector (2), baffle-column (3), propeller (18), or turbine (26) types are universally used for gas mixing.

The mixing of gases is not ordinarily considered a difficult operation. Gases are readily moved and can be made to flow together without elaborate equipment. Nearly all mixing of gases is done as a continuous process by any of the above-mentioned types, but more especially by types 1 and 2, which are the simplest forms.



Theoretical Hp. at Sp.G.=1 required by propellers at 100% slip

Actual Hp. may be only 60-70% of these 100% slip figures

FIG. 8.—Nomograph for checking propeller horse-power requirements at specific gravity 1 and viscosity of water.

Where batch mixing is desired, especially in cases where great differences in specific gravity exist, a mechanical type such as a propeller (fan) within the container is advisable (type 18).

2. Mixing Liquids with Liquids (See Tables 1 and 2)

1. As the viscosity increases, the size of the so-called "small" batch becomes smaller, so that at 200,000 centipoises we should not expect to handle over 5000 gal. per batch. For intermediate points, construct a centipoise-gallons graph using 100 centipoises, 20,000 gal., and 200,000 centipoises, 5000 gal. Connect these two points by a straight line and interpolate.

2. The same considerations apply to all other limits of quantity and degree here given. There are no hard and fast rules to be applied. They will vary for different cases. The limits are given here as reasonable suggestions.

3. Batch emulsification or dispersion is not so much a question of the speed with which the operation is completed as it is of ultimate result.

4. Continuous dispersions must always be done rapidly.

Comparison of Mixers for Liquids with Liquids.

Compressed air, a makeshift method, is sometimes satisfactory for thin and medium liquids where slow blending or coarse break-up is desired.

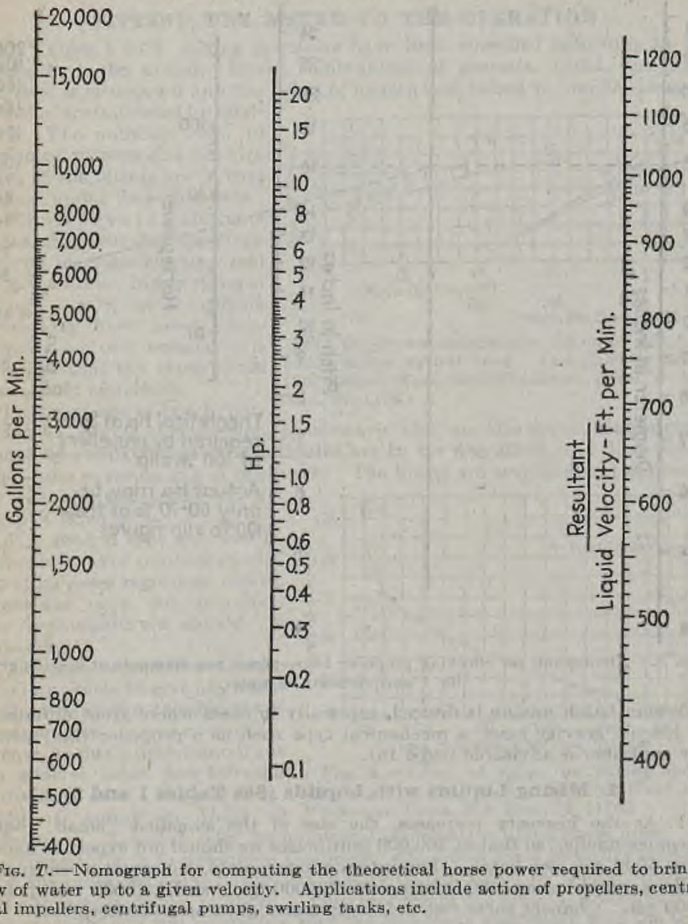


FIG. *T.*—Nomograph for computing the theoretical horse power required to bring a flow of water up to a given velocity. Applications include action of propellers, centrifugal impellers, centrifugal pumps, swirling tanks, etc.

Even then, obvious disadvantages are the danger of oxidation, loss of vapors, and extremely high power cost.

Outside circulation offers a method of mixing liquids in tanks without putting a mechanism in the tank. Heavy liquids can be pumped from the bottom and distributed over the top. Power costs are higher than for internal mixing, but the pumps are usually already present for other purposes and hence are available without extra cost. This kind of mixing is always slow.

Paddles will give slow mixing and comparatively coarse break-up in all cases of batch liquid-and-liquid mixing. Paddles do not give good mixing perpendicular to the blades, and, therefore, they are unusually slow mixers at high viscosities as they permit stratification either vertically or horizontally and consequently neglect some portion of the container. Of course, paddles are usually adequate for mixing non-viscous, miscible liquids. Wood, Badger, and Whittemore [*Chem. & Met. Eng.*, **27**, 1176–1179 (1922)], employing salt solutions, showed complete mixing in a 600-gal. tank in less than 1 min. at low power input.

NOTE: For continuous processes the above types should be avoided.

Propellers are excellent for all thin and medium viscous liquids where not too fine a break-up is desired. They are particularly useful for large-

Table 1. Mixing Liquids with Liquids*
Preferable Types of Mixers for Miscible Liquids

Basis of operation	Speed of operation	Consistency		
		Thin, to 100 c.p. Example: blending gasoline	Medium, to 2500 c.p. Example: blending lube fuel, or Diesel oils	High viscosity, to 200,000 c.p. Example: blending pyroxylin bases
Small batch High viscosity, to 5000 gal. Thin and medium, to 20,000 gal.	Slow Thin and medium, 15 min. High viscosity, 1 hr.	Air Outside circulation (4) Paddles (9) Propellers (20, 21) Turbines (28)	As for thin	Paddles (9, 10, 12) Turbines (28)
	Fast Thin and medium, 30 sec. High viscosity, 10 min.	As above	Propellers (20, 21) Turbines (27, 28)	Turbines (28)
Large batch High viscosity, to 20,000 gal. Thin and medium to 200,000 gal.	Slow Thin and medium, 15 min. to 3 hr. High viscosity	As above	As for thin	
	Fast Thin and medium, 2 to 30 min.	Propellers (21) Turbines (27, 28)	As for thin	
Continuous Any desired gallonage through-put	Slow Up to 6 hr. as in sewage treatment	Air (special design) Paddles (7, 9) Turbines (28)		
	Fast 1 to 60 sec.	Injectors (2) Orifice column (3) Pumps (5) Propellers (22) Turbines (30)	Pumps (5) Turbines (30)	Turbines (30)

* The least desirable mixer is listed first.

quantity blending of light liquids. Their mixing action is rapid. They are, therefore, useful in both batch and continuous work. The first cost is moderate. They are not very useful for high viscosities or for very fine break-up.

Turbines will mix all the classes listed in Tables 1 and 2 and probably excel all types for mixing viscous liquids, and, with the exception of the colloid mill and homogenizer, they probably excel all types for producing fine dispersions. This is particularly true of the turbo-disperser (30). Furthermore they are the fastest of all types, and their power consumption is moderate. Because of the ability of a properly designed turbine-type mixer to approximate instantaneous mixing it is the most satisfactory type for continuous work. As a partial offset to these advantages their structure is more complicated than the other types referred to, and their first cost is usually somewhat higher.

Jets, Orifices, and Pumps. On certain continuous operations these types are used with about equal success. Jets and orifice columns will provide a fairly coarse break-up with economy, but the finer break-up comes with high pressure drop and consequently high power. These types operate best

Table 2. Mixing Liquids with Liquids*
Preferable Types of Mixers for Immiscible Liquids

	Consistency					
	Thin, to 100 c.p.			Viscous, to 200,000 c.p.		
	Degree of dispersion			Degree of dispersion		
Basis of operation	Coarse (visible droplets) Example: caustic wash of gasoline	Fine (invisible droplets) Example: acid treatment of gasoline	Finest (Brownian movement) Example: ice-cream mix Commercial emulsions	Coarse (visible droplets) Unusual case	Fine (invisible droplets) Example: H_2SO_4 treatment of heavy lube oil	Finest (Brownian movement) Example: malt extract with cod-liver oil
Small Up to 1000 gal.	Paddles (9, 10) Outside circulation (4) Air Propellers (19, 20) Turbines (28)	Whippers (15) Propellers (20, 21, 22) Turbines (28)	Whippers (15) Propellers (20, 21, 22) Turbines (28)		Paddles (13, 14) Turbines (28)	Turbines (28) Paddles (13, 14) Turbo-disperser (29)
Large Up to 20,000 gal.	Outside circulation (4) Air Propellers (19, 21) Turbines (28)	Propellers (20, 21) Turbines (28)	Propellers (20, 21) Turbines (28)		Turbines (28) Paddles (13, 14)	
Continuous Any desired gallonage throughput	Nozzle (2) Orifice (3) Propellers (22) Turbines (30)	Orifice (3) Nozzle (2) Pumps (5) Propellers (22) Turbines (30)	Colloid mill (35) Homogenizer (36)		Pumps (5) Turbines (30)	Turbines (30) Turbo-disperser (29) Colloid mill (35)

* The least desirable mixer is listed first.

at one capacity and the results obtained are variable when this is changed. The centrifugal pump offers fine break-up but is difficult to control.

Colloid mills and homogenizers will produce the maximum degree of dispersion and, although not strictly mixers, are included here because they represent the apparatus which will produce the smallest particle size on most materials. They are not very satisfactory on high viscosities. They use a large amount of power, ranging from 20 to 50 h.p. per 100 gal. per hr.

3. Mixing Liquids and Gases (Table 3)

Notes on Table 3.

1. Reactions between gases and liquids always require mixing.

2. The liquids are assumed of low viscosity (say, under 100 centipoises). Cases where viscosities are higher are too rare and need not be treated here.

3. Batch reactions under pressure may require time for completion. All other cases assume a very rapid reaction, taking place before the gas escapes from the system.

4. In continuous systems the best results are generally obtained by countercurrent flow, though sometimes parallel flow is used.

5. In many cases where high efficiency is desired, two or more towers, turbo-absorbers, etc., are used in series.

6. Equipment can be obtained to handle any desired quantities of gas and liquid, except in the case of very high pressure work. Here the size is limited by the mechanical strength of the containers.

Table 3. Mixing of Liquids and Gases*
Preferable Types of Mixers for Intimate Mixing

One or more liquid phases and one gas, with or without solids present	Batch (both liquid and gas)	High pressure, up to 2000 lb. per sq. in. Example: hydrogenation	Turbo-absorber (31) Rolling bomb
		Low pressure or open, up to 50 lb. per sq. in. Example: chlorination	Bubbling gas through tank Recirculation through towers (6) Recirculation through nozzles (2) Turbo-absorber (31)
	Continuous (both liquid and gas)	High pressure (as above) Example: washing sulfur out of natural gas	Injector (2) Towers (6) Bubble columns Continuous turbo-mixer (30)
		Low pressure or open (as above) Example: sulfite liquor preparation	Cascades Injector (2) Continuous turbo-mixer (30) Towers (6)
One or more liquid phases and two or more gases, with or without solids present	Liquid-batch (gas continuous)	Pressure or open Example: removing CO_2 from air	Bubbling gas through tank Turbo-absorber (31) Recirculation through towers (6) Recirculation through nozzles (2) Recirculation over cascades Scrubbers (38)
	Both continuous	Pressure or open Example: washing illuminating gas with water	Cascades Injector (2) Towers (6) Bubble columns Scrubbers (38)

* The least desirable mixer is listed first.

Comparison of Types. 1. For autoclave work, or for time reactions such as hydrogenation, amidation, and other cases where pressure is required, it has been repeatedly demonstrated that the **turbo-absorber** is excellent because it sucks large volumes of the gas down through its impellers and distributes this gas to all parts of the container in a finely divided state. Compared to other types its repair and maintenance costs are low, since its comparatively slowly rotating vertical shaft gives little packing trouble at the stuffing box.

2. The **continuous turbo-mixer and absorber** has also a very high absorption rate on continuous work. because of the fine dispersion and long path of travel of the gases. However, it has the disadvantage that this path of travel is not so long as it is, say in a 40-ft. tower, though the tower is more expensive to construct. To overcome this, when necessary, mixers may be used in series.

3. **Bubbling** is good only where a very rapid absorption takes place as in the preparation of sodium hypochlorite from chlorine bubbled through caustic soda solution.

4. Where large volumes of gas are passed through small volumes of liquid, as in all scrubbing operations (see liquid, batch gas, continuous) the turbo-absorber and bubbling gas through a tank are not very useful methods. High efficiency is obtained in the **Feld scrubber** and in towers, especially in **bubble towers.**

5. Where solids are present, or are formed during the reaction, towers and nozzles should be avoided. This is especially true in cases where the solid accumulates on nozzles or on tower parts by crystallization or precipitation.

4. Mixing Liquids and Solids (Tables 4 and 5)

Special Cases. The tables are based on conditions involving one liquid and one solid.

In cases where one liquid phase and two or more solids are to be mixed, the tables will also serve if we bear in mind that we must choose the mixer that will best mix the solid of highest gravity. An example of this is found in the manufacture of abrasives.

In cases where two or more liquid phases and one or more solids are to be mixed, the degree of dispersion desired in the liquid phases is usually the governing factor, *e.g.*, metal polishing preparations. But this is not always true, especially if a very heavy solid is present. For example, in the preparation of aniline from nitrobenzene, iron borings, and HCl solution, we have two liquid phases and a very dense solid phase. Here the turbine type is clearly indicated for most efficient action.

Notes on Tables 4 and 5. The mixing of liquids and solids is by far the greatest and most complex category commonly encountered. The number of combinations that may result is infinite. Therefore in this class, least of all, can anyone impose definite limits on size of batch, consistency of mixture, degree of break-up, time of reaction, etc. Certain points can be indicated, however, and certain examples can be given that may help others to get a better picture of their own particular situation.

1. **Small scale** in this division is as follows: 25,000 gal. at thin consistency, and less than 10 per cent of that amount at very high consistency (as 100,000 centipoises apparent viscosity); with **large scale,** anything over the above. For **intermediate points,** construct a centipoise-gallon graph, connecting the points 1 centipoise 25,000 gal. and 100,000 centipoises 2500 gal., and interpolate.

2. The specific gravity of solids plays an important part in determining the type of mixer. The specific gravity of the liquid also must be considered, though it is not so important because it usually varies much less than that of the solid. For instance, common liquid extremes are gasoline, etc., at 0.7 sp. gr. and sulfuric acid at 1.84, whereas solids often vary from 0.9 to 7.0. Note that the limits given in the table refer to the **sp. gr. difference** between liquid and solid.

Comparison of Types.

Paddles, when correctly designed, are satisfactory for liquids of thin and medium consistency and for low-gravity solids, except for very rapid or intimate mixing such as in rapid solution or in the preparation of fine dispersions. Where applicable, paddles are economical because (*a*) of their low first cost and, (*b*) when run at low speeds, their power requirements are moderate. They will produce intimate mixtures of thick materials of all gravity differences, if sufficient time is allowed. For this, the power consumption is very high. They are also satisfactory for slow dissolving of fibrous, crystalline, and amorphous solids and for maintaining fibrous materials in fairly uniform suspension. They are not at all useful for maintaining high-gravity solids in suspension in liquids of thin and medium consistency as they have not enough sustained lifting power, and they are also impractical with fibrous material of high consistency, as the torque becomes too great.

In suspending granular solids in a liquid, White and Sumerford found [*Chem. & Met. Eng.*, **43**, 370–371 (1936)] that at a given paddle speed the best suspension is obtained with a paddle length slightly less than half the tank diameter, without baffles. The clearance of the paddle from the bottom should equal the paddle width. For a given power input, the optimum size of paddle would be slightly smaller than this. The superiority of this size of paddle over others was found to be independent of tank size, paddle speed, and size and amount of sand.

Propellers are very useful over a wide range of liquid and solid mixtures. They are satisfactory for suspension and for intimate, though not the most intimate, mixing of materials of thin and medium consistency of all kinds in batches of all sizes. They are not satisfactory on large batches of heavy materials because of size limitation, nor are they good for intimate mixtures of high consistency except those of fibrous solids. They are faster than paddles for most dissolving operations. The first cost of the various types of propellers is comparatively low, but their power requirement is moderate.

Turbine mixers surpass all other types in speed of mixing and dissolving and in intimacy of mixing. Their power consumption is not excessive, being as a rule considerably lower than that for propellers. They are entirely satisfactory on materials of all consistencies or apparent viscosities here considered and on batches of all sizes. Because of their speed and thoroughness they are the best type for continuous mixing or dissolving operations. As a partial offset to these advantages, their first cost is somewhat higher than that of paddles or propellers, although this is not true in all cases. Being based on a centrifugal principle, and hence a true disperser, or scatterer, the turbine type (especially in the turbo-disperser style) is very efficient for intimate dispersions of all sizes in batch or continuous operations. This is also true of dissolving processes.

Traveling mixers are useful for huge batches, where nothing else could be used (for example in tanks 100 ft. long by 40 ft. wide by 30 ft. high), for they accomplish sufficient turnover of the contents with rather low power

Table 4. Mixing of Liquids and Solids.* Preferable Types of *Mixers* for Small-scale Batch Mixing

Consistency	Character of solids	Examples S.S. = simple suspension I.D. = intimate dispersion D. = dissolving	Simple suspension or Rough dispersion or Slow precipitation or Slow leaching	Intimate mix or dispersion or fast precipitation or fast leach		Dissolving	
				Ordinary	Maximum	Slow	Fast
Thin Having an apparent viscosity up to 100 c.p. when placed under high rate of shear	Low-gravity difference (not over 1)	S.S. $CaCO_3$ in water I.D. Wax in water (for polish) D. Salts in water	Paddles (7, 8, 9, 10) Outside circulation (4) Air Propellers (19, 20, 21)	Paddles (9, 10, 11) Propellers (19, 20, 21, 22) Turbines (27, 28)	Propellers (22) Turbines (28) Turbo-disperser (29)	Outside circulation (4) Paddles (9, 10, 13, 14)	Turbines (28) Propellers (21, 22)
	High-gravity difference (over 1)	S.S. Sand in water I.D. Pigment washing D. Metals in acid	Propellers (19, 21) Turbines (27, 28)	Propellers (20, 21, 22) Turbines (27, 28)	Propellers (22) Turbines (28) Turbo-disperser (29)		Turbines (28) Propellers (21, 22)
	Fibrous (as cellulose)	S.S. 2% paper pulp in water I.D. Mixing same with dye D. Nitrocellulose in solvents	Air Outside circulation (4) Paddles (7) Propellers (19, 21) Turbines (27, 28)	Paddles (7, 9) Propellers (19, 21) Turbines (27, 28)	Propellers (19, 21, 22) Turbines (27, 28) Beaters		Turbines (28) Propellers (21, 22)
	Low-gravity difference (not over 1)	S.S. $Mg(OH)_2$ in water I.D. Clay in lube oils D. Gum dissolving	As for thin	As for thin	Tumbling barrel (32) Propellers (19, 22) Turbines (28) Turbo-disperser (29)	Tumbling barrel (32) Propellers (19, 20, 21, 22)	Turbines (28)

Medium Having an apparent viscosity up to 2500 c.p. when placed under high rate of shear	High-gravity difference (over 1)	S.S. HgI_2 in alcohol I.D. Lead pigment in linseed D. Litharge in NaOH	As for thin	As for thin	Ball mill (32) Propellers (19, 22) Turbines (28) Turbo-disperser (29)		Turbines (28)
	Fibrous	S.S. 5% paper pulp in water I.D. Mixing same with dye D. Nitrocellulose in solvents	As for thin	As for thin	Propellers (22) Beaters Turbines (27)		Turbines (28)
	Low-gravity difference (not over 1)	S.S. Bentonite in water I.D. Slate powder in asphalt D. Rubber in gasoline	Paddles (8, 9, 10) Propellers (19, 20, 21, 22) Turbines (27, 28)	Paddles (9, 12, 13, 14) Turbines (27, 28)	Ball mill (32) Paddles (9, 12, 13, 14 Turbines (28) Turbo-disperser (29)		Turbines (28)
Thick Apparent viscosity over 2500 c.p. and up to 200,000 c.p. when placed under high rate of shear	High gravity difference (over 1)	S.S. Flat wall paint I.D. Lead pigment in lacquer D. Plasticizer in leather dopes	Propellers (19, 20, 21, 22) Paddles (8, 9, 10) Turbines (27, 28)	Paddles (9, 12, 13, 14) Turbines (27, 28)	Ball mill (32) Paddles (9, 12, 13, 14) Turbines (28) Turbo-disperser (29)		Turbines (28)
	Fibrous	S.S. 7% paper pulp in water I.D. Mixing same with dye D. Nitrocellulose with solvents	Propellers (20, 21)	Propellers (21)	Paddles (9)		Turbines (28)

---

* The least desirable mixer is listed first.

Table 5. Mixing of Liquids and Solids.* Preferable Types of Mixers for Large-scale Batch Mixing

Consistency	Character of solids	Examples S.S. = simple suspension I.D. = intimate dispersion D. = dissolving	Simple suspension or Rough dispersion or Slow precipitation or Slow leaching	Intimate mix or dispersion or fast precipitation or fast leach		Dissolving	
				Ordinary	Maximum	Slow	Fast
Thin (see Table 4)	Low-gravity difference (not over 1)	S.S. Sugar clarification I.D. $Al(OH)_3$ precipitation D. Ice in water	Air Outside circulation (4) Paddles (7, 8, 9) Propellers (19, 21, 22) Turbines (27, 28)	Paddles (9) Propellers (22)	Turbines (28) Propellers (22)		Turbines (27, 28) Propellers (21, 22)
	High-gravity difference (over 1)	S.S. Leaching phosphate rock I.D. Blanc-fixe precipitation D. Sodium sulfate	Propellers (21, 22) Turbines (27, 28)	Propellers (22) Turbines (28)	Turbines (28)		Turbines (27, 28)
	Fibrous (as cellulose)	S.S. 2% paper pulp in water I.D. Mixing same with alum D. Nitrocellulose in solvents	Air Paddles (7, 8, 9) Propellers (21, 22) Turbines (27, 28)	Propellers (22)	Turbines (28)	Outside circulation (4) Paddles (7, 9) Propellers (21, 22) Turbines (27, 28) Traveling paddles (11)	Turbines (27, 28)
Medium (see Table 4)	Low-gravity difference (not over 1)	S.S. Starch washing I.D. Clay blunging D. Leaching diatomaceous earth	Paddles (7, 8, 9) Propellers (19, 21, 22) Traveling paddles (11) Air lift (16)	As for thin	As for thin		Turbines (27, 28) Propellers (21, 22)
	High-gravity difference (over 1)	S.S. Metallurgical slurries I.D. Cyanide process (gold) D. Barytes bleaching	Turbines (27, 28) Air lift (16)	As for thin	As for thin		Turbines (28)

---

* The least desirable mixer is listed first.

Table 5. Mixing of Liquids and Solids*

Preferable Types of Mixers for Large-scale Batch Mixing—(*Concluded*)

Consistency	Character of solids	Examples S.S. = simple suspension I.D. = intimate dispersion D. = dissolving	Simple suspension or Rough dispersion or Slow precipitation or Slow leaching	Intimate mix or dispersion or fast precipitation or fast leach		Dissolving	
				Ordinary	Maximum	Slow	Fast
Medium (see Table 4) (*Continued*)	Fibrous (as cellulose)	S.S. 5% paper pulp in water I.D. Mixing same with bleach D. Cellulose acetate in acetone	Paddles (7, 9) Propellers (21, 22) Turbines (27) Traveling paddles (11)	As for thin	As for thin	Outside circulation (4) Paddles (7, 9) Propellers (21, 22) Turbines (27, 28) Traveling paddles (11)	Turbines (28)
Thick (see Table 4)	Low-gravity difference (not over 1)	S.S. Separate digestion sewage sludge I.D. Casting slip D. Grain mash	Paddles (7, 8, 9) Turbines (27, 28) Air lift (16) Traveling paddles (11)	Paddles (9) Turbines (27) Rake mixer (37) Traveling paddle (11)		Paddles (7, 9) Turbines (27, 28) Air lift (16)	
	High-gravity difference (over 1)	S.S. Cement slurries I.D. Asphalt filling D. Leaching copper oxide	Paddles (9) Turbines (27, 28) Air lift (16) Traveling paddles (11)	Paddles (9) Turbines (27, 28) Traveling paddles (11)			
	Fibrous	S.S. 7% paper pulp in water I.D. Asbestos in asphalt D. Cellulose acetate in acetone	Propellers (19, 21, 22) Traveling paddles (11)	Paddles (7, 9) Propellers (19, 21, 22) Traveling paddles (11)			

---

* The least desirable mixer is listed first.

consumption. They are particularly useful on cement slurries, paper pulp, and other similar materials.

Outside circulation has a limited usefulness and has been discussed before under Liquids and Liquids.

Air agitation should be used only as an expedient where nothing else is possible, as it gives very unsatisfactory mixing on liquid and solid mixtures.

The air-lift agitator (Dorr) is good for large batches of material, especially high-gravity slurries, where it accomplishes a slow, thorough turnover of this material.

The ball mill, or tumbling mixer with balls, is mentioned because of its ability to produce very intimate dispersions, especially in the paint industry. This is not necessarily a grinding action but simply a separation of the flocs of pigment particles by impact. Hence it performs an intimate mixing operation. The ball mill is also occasionally useful for slow solution of nitrocellulose and gums in solvents.

Continuous Mixing of Liquids and Solids

The foregoing tables indicate the mixers to be used for large- and small-batch mixing. What mixers should be used for continuous work?

For the case of the continuous mixing of simple suspensions where approximately uniform distribution, etc., is satisfactory, use the same equipment as for batch, for both large and small scale.

For all the other cases, fast action—as nearly instantaneous as is possible—is implied, and usually the smallest possible container with the greatest possible through-put is the most desirable. Turbines are usually the most satisfactory for all cases of continuous mixing where intimate mixing or dispersion, fast dissolving, or precipitation is required, because they produce the longest path of travel and the greatest amount of recirculation within a container of given size and within a given holding time. Type 30 should be used where the flow and required holding time allow, but the usual practice for high flow rates is the use of two or more separate tanks in series. **Countercurrent flow** offers many advantages in some applications and can often be accomplished in turbine installations without the use of intermediate pumps. Many continuous-flow operations require the use of almost exact quantities of the ingredients, making it necessary to find some means of accurate proportioning.

Proportioning. For dry feed both constant volume and constant weight feeders are available while for liquids reliance is often placed on meters and pumps. Specialized equipment from which exact proportioning may be obtained has been developed. In all cases it is essential that no entrained air or gas reach the proportioning device, for it cannot discriminate between gas and liquid in its measurement of uniform volumes.

5. Mixing of Pastes, Plastics, and Doughy Masses

Strictly speaking, all materials or mixtures which possess the properties of plasticity (pseudoplasticity, inverted plasticity) as discussed on pp. 1533–1539, fall into this class. However, from a practical viewpoint, some of these materials are so thin that they must be considered in the liquid-liquid, or the solid-liquid class, and have already been discussed under 3 and 4. Therefore, it may be said that pastes, plastics, and doughy masses are those materials or mixtures, whose consistency, or apparent viscosity, ranges from 200,000 centipoises to several million centipoises. They include Nos. 4 to 8 in the list given on p. 1540, and also such mixtures as greases, bentonite solutions, dough, putty, and countless others. In evaluating their consistency, a Gardner Mobilometer may be used.

The most difficult task in the whole field of mixing is presented by these thick, doughy, sticky materials. Their yield points are usually high, and this means that substantial force must always be applied before any shear, or motion, takes place. For the same reason, the flow of these materials is limited, and the mixing is achieved through a *stretching*, a *folding*, a *kneading*, or a *tearing* action, or the most desirable combination of these actions. The object of these actions is stated exactly in Rules 1 and 2 on p. 1530. Since there is little flow, the particles of one constituent must be *forced* between the particles of other constituents until the whole mass is in a thoroughly mixed condition and all surfaces of solid particles are thoroughly wetted. Furthermore, every particle within the container must be brought to the place where it may best be subjected to this force.

As the consistency (apparent viscosity) of these pastes and plastics becomes greater, ranging from 200,000 centipoises to several million centipoises, increasingly heavier mechanisms must be used. In general, however, except for types 39 and 40, they will all be found to be multibladed paddle, or arm, types. The following types are arranged in the order of the consistencies they are capable of handling 7, 9, 25, 12*a*, 13, 14, 12*b*, 40, 23, 17*a*, 17*c*, 39.

In comparison with mixers for more fluid mixtures, the sizes of these heavy-duty machines are usually small, *i.e.*, seldom over 1000 gal. and usually smaller. Nevertheless, a large amount of time and power is consumed for the operation. For example, a machine of type 17*a*, designed to mix 300 gal. of a plastic, modeling clay, requires 75 h.p. and an hour to do the work. The dispersion type (17*c*) often requires 200 h.p. per 100 gal., for example, on rubber compounding. The main thing is that this machine is thoroughly successful at its task. Types 12*b* and 40 would also satisfactorily mix a batch of modeling clay.

Analysis of Mechanical Actions Involved. In order to analyze the mechanical actions involved in mixing materials of this class, consider the preparation of bread dough in a mixer of type 17*a*. The raw materials are a limited amount of water or milk, other minor ingredients, and flour which is full of air. The water must displace the air and wet the entire surface of each particle of flour. When completely mixed, both materials must be uniformly distributed with respect to each other in that form which is known as dough. The mixing elements and the container itself must be able to perform various functions. First, the elements must *transport* material from one end of the container to the other, and back again. The design makes this possible, because the mixer arms, or blades, usually of the sigma form, are not parallel to the axis upon which they rotate, but are pitched so that material is pushed back and forth. Second, the elements must *knead* the material by pressing it against the walls of the container and against contiguous material. This pressure tends to force water between the flour particles, an action which displaces the air which has been present. It also causes shear, which brings a new relationship to different portions of the partially mixed material. Third, as the mixer arms rotate, they *tear* loose portions of the mass, carrying these portions to other parts of the container, thus redistributing the contents. Fourth, as the mass becomes coherent, *stretching* takes place. That is, the mixer arm grips a portion of the material, stretching it as a rubber band is stretched. The tension to which this portion of material is subjected, with the concomitant compression, which occurs at right angles to the other force and is analogous to a kneading action, is one of the chief factors in working the water into the flour. Fifth, the stretched material is then *folded* over on itself or on fresh material, and thus a realignment of material is brought about. These actions occur repeatedly until the whole is mixed.

Each of these actions depends for its effectiveness on the amount of shear or transportation it produces. A combination of all these actions is the usual thing in mixing pastes, plastics, and doughy masses, and all the types listed here are capable of a similar performance on suitable consistencies.

6. Mixing of Solids with Solids (also Solids with Gases)

The problems encountered depend on relative size, shape, and gravity of particles. When two kinds of solid particles to be mixed are the same size but of different gravity, naturally the heavier seek the bottom. When of the same gravity but of different size, the smaller particles seek the bottom. So also do round, smooth particles, while the jagged or polyhedral ones seek the top of the mass. In mixing solids, these natural separating tendencies must be overcome, and this is invariably done by some means which lifts material from the bottom to the top of the mass, the resulting voids being filled from above by gravity. Simultaneously the means must also produce horizontal transportation in at least two opposite directions.

Types 4, circulating system; 12*b*, rotating pans with offset blades; 25, the helical ribbon mixer; 32, the tumbling barrel; 33, the double-cone mixer; 34, the mushroom mixer; 37, the rake mixer; and 40, the putty chaser, or pan mixer, are used. The rake mixer and the tumbling barrel are also used for mixing solids with gases.

For mixing of batches larger than 1000 cu. ft. (as in the large-scale preparation of molding powders), circulating systems or large tumbling barrels are preferable, though no ideal method has yet been discovered. Circulating systems represent a mechanical quartering method and therefore are slow. In such sizes tumbling barrels, because they do not produce good end-to-end flow of material, even when provided with internal baffles, give incomplete mixing unless excessive time is consumed.

For batches of ordinary size, or up to 1000 cu. ft., the double-cone mixer is the most successful type yet devised. Large or small batches are mixed in 15 min. or less. Power is low, wear is negligible, and discharge is rapid and leaves a smooth inside surface which is easily brushed clean. Thus it is useful where extreme cleanliness is required or where the mixture is colored, as in the mixing of dry colors. The ribbon mixer is the most common type because it is older and somewhat lower in first cost but is slower than the cone mixer for the same power consumption and is not quite as thorough nor is it as easily cleaned. The tumbling barrel is less effective than either of the two just mentioned.

Whenever necessary to obtain extremely intimate mixing, involving the breaking down of aggregates or the coating of one solid material with another, as in the standardizing of dyestuffs with salt or the coating of resin granules with color, a mixer producing a smearing or shearing action must be used. The types with mullers, as 12*b* and 40, are good. A tumbling barrel containing a few balls or pebbles is often satisfactory. The mushroom mixer with balls is a modified type of tumbling barrel finding special use in the dye and pharmaceutical industries. The usual blending requires 1.0 to 1.5 h.p. per 1000 lb. of solids.

THE TRANSMISSION OF POWER TO MIXERS*

Prime Movers. In recent years the tendency has been for individual drives, wherever possible, to replace the line-shaft method of power trans-

---

* See section on the Mechanical Transmission of Power, pp. 2494–2524.

mission. Neat installations, flexible in operation and easily maintained at low cost, may be made. Individual drives, therefore, are heartily recommended wherever working conditions permit. The use of totally enclosed and explosion-proof motors makes this possible today in places where formerly it was not to be considered because of dirt, water, or flammable materials.

In many mixing operations the exact degree of break-up or dispersion, as well as other desired results, is controlled largely by the speed of the mixer. To obtain the proper adjustment, a variable-speed motor is often used. Steam turbines have also been employed with excellent effect for this purpose. Air motors and hydraulic motors have been introduced for variable-speed operation.

Power Transmission to Mixer. It is then necessary to transmit the power to the mixer shaft so that it will run at the required speed. (Individual drives only will be considered.)

The following methods are usually used (numbers refer to illustrations in section, Types of Mixers):

1. Direct connected (20). This gives full motor speed.
2. Gear-reduction units with built-in or separate motors (7, 17*a*, 21, 22).
3. Connected through multi-V-belt (30).
4. Connected through chain (12*a*).
5. Connected through flat belt with pulley (10, 15).

Of all these methods the gear reduction unit has gained the widest popularity, and justly so. With reliable prime movers, it has no disadvantage. The first cost is moderate; it is compact and easy to maintain and to operate. However, in cases where the horse power is above 60 or where severe starting shock may be experienced, a V-belt or chain drive is preferable.

In the transmission of power to a mixer, variable speed is often obtained by the interposition of a Reeves drive, a Link-belt PIV drive, or other similar device between motor and mixer.

The above considerations apply to cases where the mixing elements are mounted on either horizontal or vertical shafts. However, where vertical shafts are used, the vertical motor-reducer unit is preferred to the right-angle drive unit in present practice where head room permits.

FROM LABORATORY TO PLANT*

Research work on a new process or product usually starts in the laboratory, using glass rods for mixing. As a second step, the glass rods are bent and attached to the laboratory mixer. Then, having shown promise of being chemically sound, the process has justified the construction of a pilot plant in which it will be given the opportunity of proving its value under conditions approaching commercial practice.

If properly laid out, the pilot plant should be capable of producing the information necessary for translating to full commercial operation. This information will involve such points as

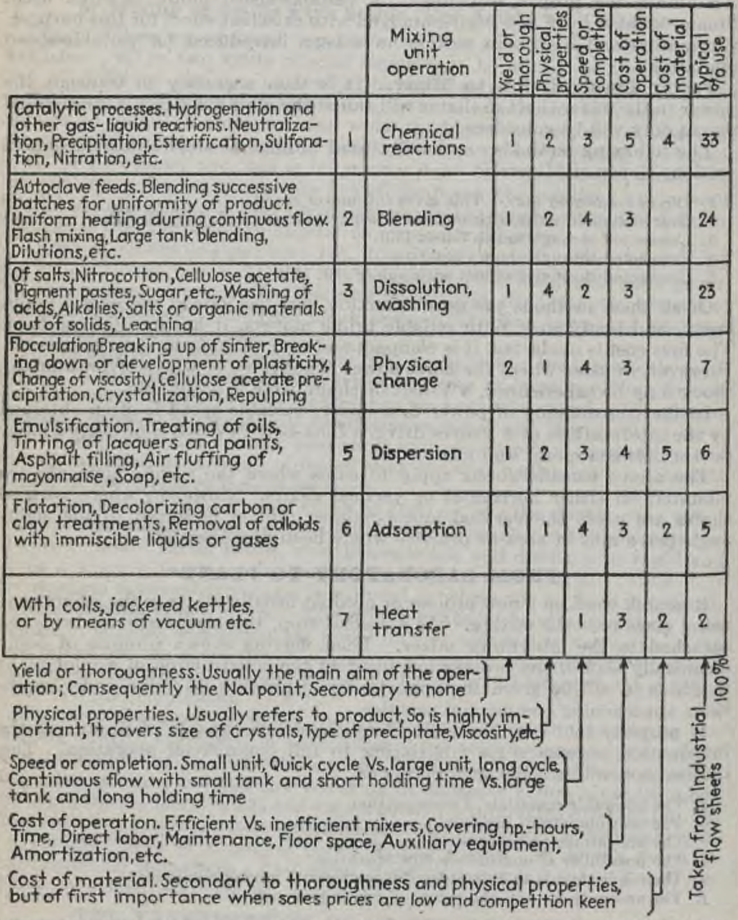
a. The allowable materials of construction.
b. The unit operations involved.
c. The limitations on batch size; or
d. The possibility of continuous flow operation.
e. The requirements to be met by the commercial mixing equipment.
f. The necessity for and the extent of heating or cooling.

At the very start of pilot plant design it is necessary to take into account the objectives of the mixing operations, such as

---

* Taken in part from an article published in *Ind. Eng. Chem.*, **30**, 489 (1938) through courtesy of the publishers and coauthor E. J. Lyons of the Turbomixer Corp.

1. Chemical reaction.
2. Blending.
3. Dissolution or washing.
4. Physical change.
5. Dispersion.
6. Adsorption.
7. Heat transfer.



		Mixing unit operation	Yield or thorough	Physical properties	Speed or completion	Cost of operation	Cost of material	Typical % use
Catalytic processes. Hydrogenation and other gas-liquid reactions. Neutralization, Precipitation, Esterification, Sulfonation, Nitration, etc.	1	Chemical reactions	1	2	3	5	4	33
Autoclave feeds. Blending successive batches for uniformity of product. Uniform heating during continuous flow. Flash mixing, Large tank blending, Dilutions, etc.	2	Blending	1	2	4	3		24
Of salts, Nitrocotton, Cellulose acetate, Pigment pastes, Sugar, etc., Washing of acids, Alkalies, Salts or organic materials out of solids, Leaching	3	Dissolution, washing	1	4	2	3		23
Flocculation, Breaking up of sinter, Breaking down or development of plasticity, Change of viscosity, Cellulose acetate precipitation, Crystallization, Repulping	4	Physical change	2	1	4	3		7
Emulsification. Treating of oils, Tinting of lacquers and paints, Asphalt filling, Air fluffing of mayonnaise, Soap, etc.	5	Dispersion	1	2	3	4	5	6
Flotation, Decolorizing carbon or clay treatments, Removal of colloids with immiscible liquids or gases	6	Adsorption	1	1	4	3	2	5
With coils, jacketed kettles, or by means of vacuum etc.	7	Heat transfer			1	3	2	2

Yield or thoroughness. Usually the main aim of the operation; Consequently the No.1 point, Secondary to none

Physical properties. Usually refers to product, So is highly important, It covers size of crystals, Type of precipitate, Viscosity, etc.

Speed or completion. Small unit, Quick cycle Vs. large unit, long cycle. Continuous flow with small tank and short holding time Vs. large tank and long holding time

Cost of operation. Efficient Vs. inefficient mixers, Covering hp.-hours, Time, Direct labor, Maintenance, Floor space, Auxiliary equipment, Amortization, etc.

Cost of material. Secondary to thoroughness and physical properties, but of first importance when sales prices are low and competition keen

Taken from Industrial flow sheets 100%

FIG. *U*.—Evaluation chart. Starting with a series of typical industrial flow sheets, the mixing operations are placed in seven classes of unit operations. Five mixing factors are then arbitrarily evaluated with respect to each unit operation. (See Table I.)

The physical characteristics of the materials such as viscosity, and specific gravity, and their variations should be available. The mixers in the pilot plant should have sufficient flexibility for the determination of the factors which will affect the ultimate choice of the type of mixing equipment. This flexibility will allow the determination of the importance of

A. High recirculation rate of entire tank contents per minute	1–6
B. At least one point of high shear	1, 2, 5, 6
C. General high level of velocity throughout tank	1, 2, 3, 6, 7 (coils)
D. Violent scrubbing effect	1, 3, 5, 6
E. Instantaneous blending or dispersion of added material through large volume of tank contents	1, 2, 3, 5, 6
F. High velocity over tank walls	2, 3 (viscous), 5 (viscous), 7 (jacketed)
G. High velocity over entire tank bottom	1–4, 6, 7
H. High-volume low-velocity	1–5
I. Uniform overflow (continuous operation)	1, 2, 4–7
J. Low short-circuiting (continuous operation)	1–6
K. Selective overflow (continuous operation)	1, 3
L. Horse power per unit of volume	1–6
M. Freedom from swirl	1–6

The figures after each of the above factors refer to the operations that may be affected.

To get such flexibility requires the availability of variable speed on the mixer shaft; ease of disassembly for changing mixer units; excess tank volume so that points *A*, *C*, *D*, and *H* can be settled; provision for baffling; and means for making power readings. This latter provision in conjunction with the variable-speed feature allows the determination of the lowest power per unit of volume with which satisfactory results can be secured, often a great help when the commercial plant is being designed and an estimate of operating costs is being made. Power should be read in terms of peripheral mixer speed for turbines and paddles and in projected stream velocity for propellers.

All mixing units should be accurately weighed before use so that such factors as abrasion and corrosion can be evaluated in terms of replacement cost of equipment and contamination of product.

It costs but little more to set up the pilot plant with these provisions, and this extra cost may declare dividends when the commercial plant is finally installed. The operations will be thoroughly understood and whether you elect to build your own equipment or whether you call in the engineer from the mixer manufacturer for recommendations, the final results will more than justify the extra expense.

HEAT TRANSFER IN MIXERS

Efficient heat transfer in kettles and tanks is very closely related to the amount of mixing provided for these same containers. Where plant operations are being scientifically handled, it is axiomatic that good agitation is not only desirable, but absolutely necessary, to increase heat transfer and to prevent local overheating or cooling.

Heating or cooling of tanks or kettles is employed

1. To control the speed and extent of a chemical reaction, such as in the sulfonation of an oil, nitration, catalytic reaction, etc.
2. To accomplish a physical change, such as evaporation, pasting, crystallization, dissolution or melting, to change consistency or to promote blending.

3. To aid in dispersing one material in another.
4. To restore the desired temperature to product.

For heating, the means most generally encountered in the chemical industry today are: (1) direct or indirect firing; (2) water or steam; (3) hot oil; (4) Dowtherm (diphenyl or diphenyl oxide or a mixture of these), liquid, or vapor; (5) mercury vapor; (6) electric heaters, contact, immersion, or radiant.

For cooling, the means employed are: (1) air; (2) an evaporant, such as liquid ammonia; (3) water or brine; (4) oil; (5) Dowtherm; (6) vacuum; (7) direct ice addition.

In every case the largest problem is not the transfer of heat through the metal itself, even in the case of a relatively poor conductor like lead. It is rather the transfer of heat through the residual film on *both* the outside and inside wall of the metal separating the medium from the product. For good heat transfer, these films must be thin. The conductivity of most of the media mentioned above, as well as of most of the materials being mixed, is only a very small fraction of the conductivity of the metals of the container; therefore a much greater difference of potential (in terms of heat) is required to force the heat through $\frac{1}{100}$ or $\frac{1}{50}$ in. of film than to force it through $\frac{1}{2}$ in. of metal (see Sec. 7 on Heat Transfer for detailed discussion). It cannot be too strongly stressed that in designing a good unit, every feasible means for removing both films should be considered, so that the highest desired over-all rate of heat transfer, herein called K, can be obtained. K = over-all B.t.u./(sq. ft.)/(°F.)/(hr.).

A higher coefficient, K, results, in practice, in

1. An improved product because of prevention of local overheating.
2. Longer life of equipment for same reason.
3. A faster process (see Figs. *V* and *W*).
4. Less square feet of heat transfer surface.
5. Saving of fuel due to reduction of heat losses.
6. Less medium to circulate, with consequent saving of power.

Satisfactory rates are obtained by maintaining a substantial velocity of liquid or vapor along both heat-transfer surfaces. For liquids these speeds may economically range from 200 to 400 ft. per min., and for vapor, much higher. In coils these velocities are fairly easily attained—a reason why coils are at times preferred to jackets. Where the heat-transfer medium is a liquid, jackets with baffles and preferably with spiral baffles, should be employed if a high rate is important. For further improvement the space between container shell and jacket shell should be narrow, *i.e.*, 1 to 3 in.; otherwise the rate of heat transfer may be surprisingly poor. For example, in a case where a viscous milk product was being cooled in a jacketed tank, a plain, unbaffled jacket gave a K of 35, whereas the same unit, with jacket spirally baffled to maintain a cold-water velocity of 200 ft. per min. gave a K of 55, using the same amount of cooling water at the same inlet temperature. Figure 13 shows a kettle with baffled jacket. In fired kettles, best results are obtained by arranging the setting so that hot gases sweep around the bottom and sides of a kettle in a gradually ascending spiral, for this causes the cooler gas film against the kettle to be greatly attenuated.

The finish on the metal also influences the K, so, where practical, both the inside and the outside of the metal should be smooth and free from pits and ridges.

Inside the vessel the surface film must be removed by some sort of agitation or by scrapers. The choice of method depends upon the consistency of the mixture. Where the boiling of thin liquids is being carried on, ebullition will often provide satisfactory agitation. However, in every other instance, whether heating or cooling, good agitation is definitely advantageous. Sluggish action at the wall means an insulating layer of viscous liquid or film of solid through which the heat will diffuse but slowly. Once a solid film begins to form, it grows rapidly in thickness, and heat transfer becomes progressively worse.

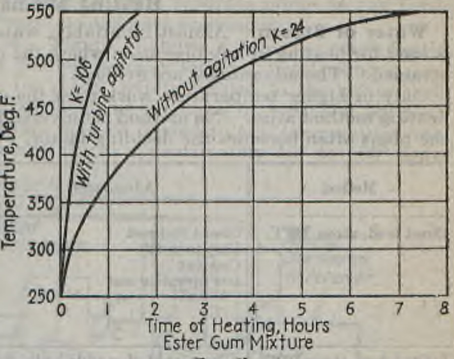


FIG. *V*.

When the vessel contents are of reasonably thin consistency, a type of mixing element having large pumping capacity is desirable. Propellers or preferably turbines are best. They should be located so that their stream spreads over the entire heated or cooled area as uniformly as possible. Thus the highest velocities are obtained where they will be most effective. Propellers as shown in Fig. 21 are satisfactory. However, turbine types are greatly preferable, especially when the viscosity is appreciable. Their discharge is directed radially and uniformly at the walls, sweeping these at all points with high velocity currents of liquid. In heating materials which are not thinned by heat or in cooling materials which congeal, even turbines become less effective as film removers at thicker consistency. Although the mixing may seem excellent there is nevertheless a stagnant layer at the wall, its thickness bearing a very definite relationship to the viscosity of the material (see definition of viscosity p. 1533). For higher consistencies, therefore, greater heat transfer is obtained when the film is removed by *scraping* the walls which can be done by means of hinged or spring scraper blades affixed to agitator arms (see Fig. 13). In cases involving very sluggish materials, a double-motion mixing element, as in Fig. 21, or a combination of paddles with turbines, is desirable to transport material from the center of the vessel to the walls.

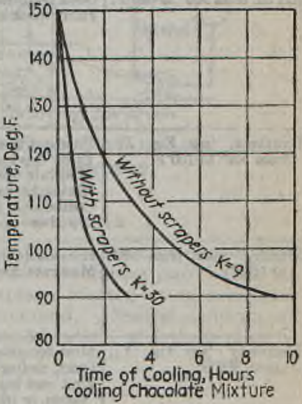


FIG. *W*.

Figures *V* and *W* show the effect of good agitation on film removal with consequent improvement of heat transfer. For instance, Fig. *V* shows the time of heating on ester gum in a 1000-gal. kettle from 250°F. to the top temperature of 550°F. In this case the temperature on the outside of the kettle was around 700° to 750°F. Without agitation a coefficient of heat transfer of approximately 24 is obtained. Satisfactory agitation raises the K to 106, approximately a fourfold improvement. Figure

W illustrates the cooling of a chocolate mixture in 1000-gal. batches from 150° to 90°F. with non-scraper and with scraper type of stirrer. With a jacket cooling water of 70° to 75°F., triple efficiency is noted when scrapers are used.

Heating Means

Water or Steam. Almost invariably, water or steam in jackets or coils is used for heating in any operation where the desired temperature may be so attained. The advantages are evident.

Only in higher temperature work does the question of proper selection of heating method arise. No method is universally applicable. The location of the plant often becomes the deciding factor.

Method	Advantages	Disadvantages
Direct fired, above 300°F.	Lowest first cost Easy to install Compact Low operating cost	Fire hazard Local overheat of product Cannot cool quickly Short life High maintenance Lag on heating and cooling Constant supervision
Indirect fired, above 300°F.	Some control of outside kettle-shell temperature First cost less than Dowtherm, hot oil, and mercury, but more than electricity Lowest operating cost	Fire hazard Cannot cool quickly Elaborate setting construction Lag on heating and cooling Constant supervision
Hot oil, from 300° to 600°F.	Good temperature control Fairly quick cooling	High first cost Much auxiliary equipment Decomposition of oil and inflammability No higher heat efficiency than direct or indirect, ±50% Constant supervision Liquid system only
Dowtherm (see Fig. *X*), from 300° to 700°F.	Excellent temperature control Fairly quick cooling Moderate operating cost Reasonable maintenance Combustible but non-flammable Liquid or vapor system	Higher first cost Much auxiliary equipment Steady loads—60 to 65%; intermittent loads—±50% Periodical inspection only with good controls
Mercury vapor, from 600° to 1200°F.	Reasonable temperature control Moderate operating cost	Highest first cost Much auxiliary equipment Expense of mercury Constant supervision High maintenance
Electricity (see Fig. *Y*), above 300°F.	Most accurate temperature control Quick cooling First cost lower than hot oil, Dowtherm, or indirect fired Clean, compact, instantaneous Highest over-all heat efficiency Lowest fire hazard No danger from leaks or collapsing of jacket or coils Easiest and cheapest to install and easily movable Least attention while running Lowest maintenance cost	First cost higher than direct Operating cost usually high

Circulating Systems. Figure *X* illustrates a typical circulating system employing Dowtherm vapor for heating the kettle and Dowtherm liquid for the subsequent cooling. In this case, condensate returns to the boiler by gravity. Where this is not feasible, a return pump is employed. Cooling Dowtherm is handled from the receiver by a separate pump to the kettle and thence back through the water-cooled heat exchanger.

Electrically Heated Kettle. Figure *Y* shows a typical electric kettle. Heat is supplied by means of flexible, spirally formed, metal ribbons, centrally located inside of hoods (or pipes) solidly welded to the kettle shell, the ribbons being prevented from touching the pipes by means of porcelain spacers. The elements themselves may be heated to any temperature from 300° to 1300°F., producing temperatures approximately 50° to 100° lower within the kettle itself. Radiation is prevented by means of a double steel jacket carrying 6 in. of insulation. Between the insulating jacket and the tank is a substantial air space with water sprays located at intervals for instant cooling. A blower for air cooling is often used. Electric elements are usually arranged in two or three banks for zone heating, and these may be run separately or together.

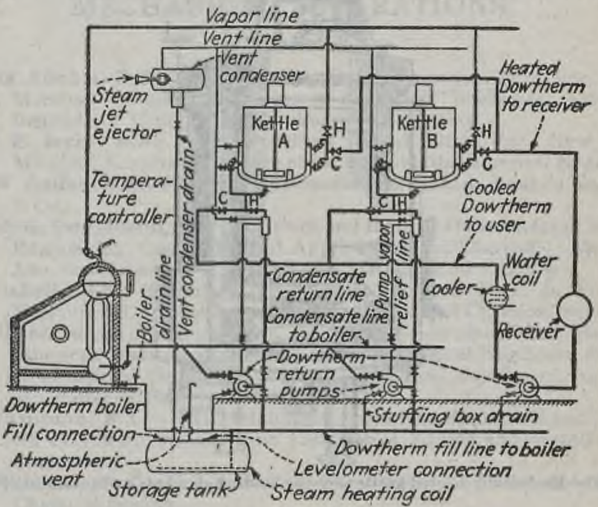


FIG. *X*.—Dowtherm system for heating kettles.

Cooling Means

A surprising amount of cooling is still done in some chemical industries by the direct addition of crushed ice, but in many others this method is outlawed by the attendant dilution. A satisfactory substitute may be seen in the chilling of lubricating oil through the addition of liquid propane and its subsequent evaporation. This agent boils off at room temperature, but low temperatures may be attained by the application of vacuum, thus lowering the boiling point. By the application of suitable vacuum, water itself becomes the evaporant, and increasing use is being made of steam jet ejectors for this purpose. Most of the large distilleries in this country are using this method for cooling corn mashes part way to the fermenting temperature. Much water also is being cooled by this method for industrial purposes and for air conditioning. Mixing is vital in such units, for the liquid to be cooled must be brought to the surface where the static head is zero, in order to attain the temperature justified by the reduced pressure. Much greater use will be made of this means of temperature reduction in industry, especially where the water supply is too warm in summer to be effective for cooling purposes.

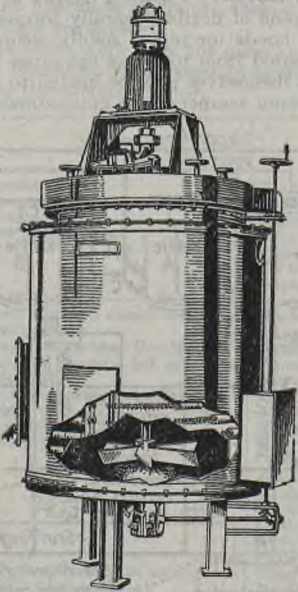


FIG. *Y*.—**Electrically heated kettle showing heating elements and insulating jacket.**

The other cooling means do not need especial mention since their use is well established.

SECTION 15

MECHANICAL SEPARATIONS

BY

Anthony Anable, B.S., Engineer, The Dorr Company, Inc., New York; Member, American Institute of Chemical Engineers, American Institute of Mining and Metallurgical Engineers.

Donald F. Irvin, B.S., Engineer, Oliver United Filters, Inc., New York; Member, American Institute of Mining and Metallurgical Engineers.

Hugh W. Bellas, B.S., Engineering Department, E. I. du Pont de Nemours & Co.

Kenneth H. Donaldson, B.S., Professor and Head of Department of Mining Engineering, Case School of Applied Science, Cleveland; Member, American Institute of Mining and Metallurgical Engineers.

C. P. Cabell, B.S., Technical Advertising Assistant, E. I. du Pont de Nemours & Co.; Member, American Institute of Chemical Engineers.

J. L. Gillson, Sc.D., Geologist, E. I. du Pont de Nemours & Co.; Member, American Institute of Mining and Metallurgical Engineers, Society of Economic Geologists; Fellow, Geological Society of America, Mineralogical Society of America.

Robert Ammon, B.S., Chief Metallurgist, American Zinc, Lead and Smelting Co.; Member, American Institute of Mining and Metallurgical Engineers.

J. W. Stillman, Ph.D., E. I. du Pont de Nemours & Co.; Member, American Chemical Society.

Warren L. McCabe, Ph.D., Professor and Head of Department of Chemical Engineering, Carnegie Institute of Technology, Pittsburgh; Member, American Institute of Chemical Engineers, American Chemical Society.

A. E. Flowers, M.E. in E.E., M.M.E., Ph.D., Engineer-in-charge of Development, The DeLaval Separator Company, Poughkeepsie, N.Y.; Member, American Society of Mechanical Engineers, American Institute of Electrical Engineers, American Society of Testing Materials, American Physical Society.

Evald Anderson, B.S., Late Technical Director, Western Precipitation Corp. Los Angeles; Member, American Institute of Chemical Engineers, American Physical Society, American Chemical Society.

C. E. Lapple, Chem. E., Chemical Engineer, Engineering Department, E. I. du Pont de Nemours & Co.; Member, American Institute of Chemical Engineers.

O. C. Ralston, Ph.D., Chief Engineer, Nonmetallic Minerals Division, U. S. Bureau of Mines.

CONTENTS

PAGE

CLASSIFICATION

SEDIMENTATION

FILTRATION

MISCELLANEOUS METHODS OF SIZING AND CONCENTRATION OF MATERIALS

SAMPLING

CRYSTALLIZATION

CENTRIFUGES

SEPARATION OF DUSTS AND MISTS

CLASSIFICATION

BY ANTHONY ANABLE

DRAINING

Definition. Draining may be defined as the dewatering action taking place when moist particles of a granular substance are placed on, or advanced over, a flat or inclined surface. Dewatering by centrifugal machines and by filters accomplishes the same purpose but is discussed separately under Centrifugation and Filtration.

Purpose. It is the purpose of draining to separate, in so far as is possible by gravitational force alone, the two constituents of the mixture—the solid portion and the liquid portion—so that each may subsequently be subjected to such further more complete and generally more expensive separation treatment as may be required, such as, in the case of the solids, centrifugation, filtration, or drying and, in the case of the liquid, clarification or filtration.

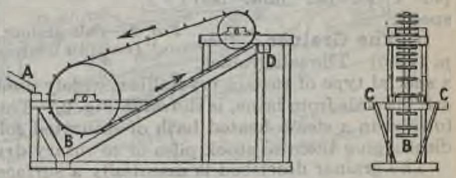


FIG. 1.—Drag conveyor.

Draining, by itself, is seldom a complete treatment, as it does not result in the production of a crystal-clear solution and a bone-dry solid. Due, however, to its relative cheapness, it is an approved preliminary unit operation, preparing the two constituents of the mixture so as to meet the requirements of the additional unit operations which follow.

Equipment Used. a. The Drag Conveyor. The simplest type of mechanical apparatus for draining is the drag conveyor shown in Fig. 1. This device consists of a shallow, inclined, rectangular box, equipped with an endless belt or chain to which scrapers are attached. The belt or chain is carried on two sets of pulleys or sprockets, located respectively at the upper and lower ends of the inclined box. The shafts upon which the pulleys or sprockets are mounted lie with their axes at right angles to the long axis of the box and at such an elevation above the inclined bottom that the scrapers just clear the bottom as they are drawn along from the lower to the upper end.

The mixture of solids and liquids enters through a trough at the lower end. The heavy granular solids settle to the bottom, are picked up by the scrapers, and are dragged up the incline, out of the solution, over the draining deck and discharged from the upper end of the box. Solution entrained with the solids partially drains in the process and flows back into the lower portion of the box.

The solution from which the heavy solid particles have settled overflows across a weir at the lower end of the box or through a pipe. Finely divided solids, contained in the feed mixture and settling more slowly than the granular portion, remain suspended in the solution and pass off with it as an overflow product. Being a "home-made" device no data on size, capacity, or power are available.

Drag classifiers are generally about 3 ft. to 5 ft. 6 in. wide, set at slopes ranging from 2 to 6 in. per ft., the steeper slope corresponding to the coarser separations. The belts are equipped with wood or steel blades on 12- to 18-in. centers and are driven at 12 to 40 ft. per min.

Capacity is roughly proportional to width and belt speed and roughly inversely proportional to the blade spacing. Data from commercial installations indicate that when making coarse separations at from 28 to 48 mesh, a capacity of 5 to 7 tons per 24 hr. may be secured per foot of tank width per 1 ft. per min. belt speed. When making finer separations around 100 to 200 mesh, the capacity may be as low as 1.5 to 2.5 tons per 24 hr. per ft. of tank width per 1 ft. per min. belt speed.

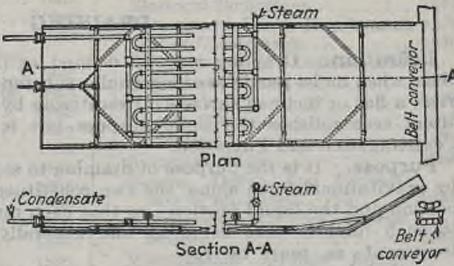


FIG. 2.—Salt grainer. (*Courtesy of Badger and Baker, "Inorganic Chemical Technology," McGraw-Hill.*)

b. The Grainer. (See p. 1786.) The salt grainer, a special type of surface crystallizer widely used in the manufacture of coarse salt crystals from brine, is shown in Fig. 2. This machine collects the crystals formed in a steam-heated bath of saturated solution and drains them before discharging them to stock piles or to rotary dryers.

The grainer described is essentially a surface crystallizer and the draining obtained is merely incidental to mechanical discharge. Rake speed is adjusted to suit crystal formation independent of drainage conditions, and subsequent drainage in stock piles or by filters or centrifuges is used.

The grainer tank, constructed generally of concrete, is 100 to 150 ft. long, about 12 to 18 ft. wide, and about 2 ft. deep. It is filled with brine and equipped with steam coils which warm the contents slightly below the boiling point to the point of surface crystallization.

A reciprocating mechanism actuated by a steam or water piston travels slowly back and forth over the bottom of the tank and over the drainage deck at the discharge end. Raking blades are hinged to the mechanism at regular intervals, being arranged in such a manner as to swing on the hinges with a feathering motion. The blades lie parallel to the tank bottom and to the drainage deck on the return stroke and then immediately take a position at right angles to the bottom on the forward or crystal-advancing stroke.

The salt crystals are pushed along the tank bottom at regular intervals, eventually emerging from their bath of mother liquor and being subjected to a period of draining on the inclined deck. At intervals of a few weeks the grainer is shut down and cleaned, as scale forms rapidly on the coils and impurities collect in the liquor to a degree endangering the purity of the finished salt.

The capacity of a grainer 150 ft. long by 18 ft. wide by 22 in. deep is 11.6 to 14.5 tons per day (Badger and Baker, "Inorganic Chemical Technology," p. 15, McGraw-Hill, New York, 1928).

c. The Dorr Classifier. The Dorr classifier is shown in Fig. 3. It consists of a settling box of wood, concrete, or steel, in the form of an inclined trough, with or without lining of rubber, lead, or special metals, with the upper end open, in which are placed mechanically operated rakes or scrapers which carry the quick-settling granular material to the point of discharge at the open end. Each rake is carried by two hangers, one at the discharge end and the other at the overflow end. Special covering or special metals may be used with corrosive solutions.

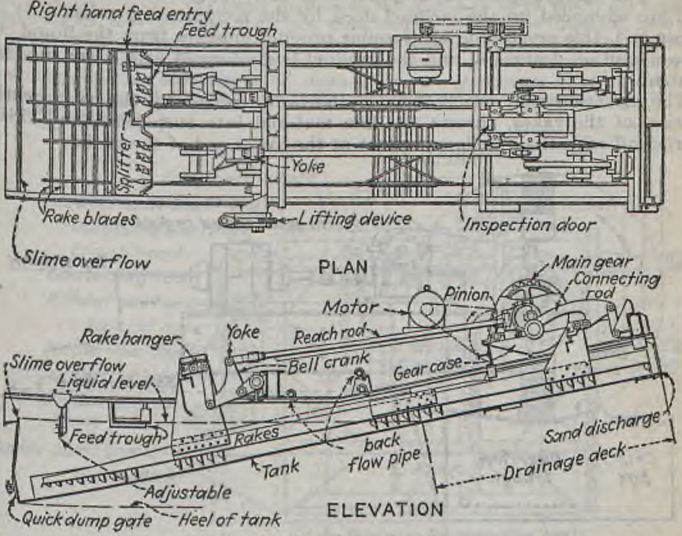


Fig. 3.—The Dorr classifier.

The rakes are lifted and lowered vertically by the hangers by the action of the head motion, transmitted through eccentrics and cranks. The horizontal motion is obtained directly from the head-motion crank. The rakes can be raised several inches at the lower end by a lifting device and operated

Table 1. Dorr Classifiers

Type of mechanism	Width	Max. length	Min. length	Aver. conn. h.p.*
The Dorr FR—light duty For small open-circuit operation where the sand is small compared to the overflow	1'6"	18'4"	12'0"	1½
	2'0"	23'4"	15'0"	2
	3'0"	23'4"	15'0"	2
	4'0"	23'4"	15'0"	3
The Dorr FH—intermediate duty For small closed-circuit operations with relatively light circulating loads	4'0"	30'0"	18'4"	3
	5'0"	30'0"	18'4"	5
The Dorr F—normal duty For the average or normal closed-circuit operation—the general run-of-mill classification job	6'0"	30'0"	18'4"	7½
	8'0"	30'0"	18'4"	7½
	12'0"	30'0"	18'4"	10
	16'0"	30'0"	18'4"	15
The Dorr FX—heavy duty For the heavy and really tough closed-circuit operation, where the circulating load is high, the sands are heavy, and the service is exacting	5'0"	31'6"	24'0"	10
	6'0"	31'6"	24'0"	10
	7'0"	31'6"	24'0"	15
	8'0"	31'6"	24'0"	15
	12'0"	31'6"	24'0"	20
	14'0"	31'6"	24'0"	25
	16'0"	31'6"	24'0"	25

* Average connected horse power is the power rating of the motor supplied for direct drive and is 25 to 50 per cent in excess of the actual power required to drive the classifier on a job of average severity.

in that, or any intermediate, position. This allows the classifier to be started readily when nearly filled with solids after an unexpected shutdown.

The feed enters continuously from a distributing trough near the overflow end. The heavy, quick-settling particles sink to the bottom of the tank and are advanced up the inclined deck by the reciprocating rakes. At the upper end, this crystalline or granular product emerges from the liquid, the excess solution drains off, and the product is discharged with a low moisture content.

The agitation near the bottom of the tank, caused by the reciprocating motion of the rakes, throws the fine material into suspension and this is carried off with the overflow product at the lower end of the tank.

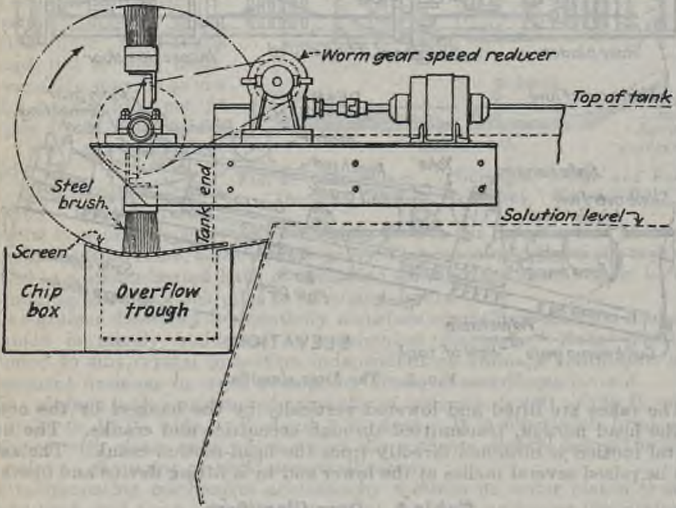


Fig. 4.—Chip-removing device on Dorr classifier.

Adjustments. All other things being the same, the mesh of separation between discharge and overflow solids is determined by rake speed, overflow dilution, and slope of tank bottom. The greater the rake speed, the lower the dilution; and the steeper the slope, the coarser is the separation.

Example 1. With quartz of 2.7 sp. gr., suspended in water, a 100-mesh separation corresponds to 16 to 20 raking strokes per minute, 4 or 6 to 1 overflow dilution, and a tank slope of 2½ to 2¾ in. per ft. A 20-mesh separation with the same materials corresponds to 27 to 32 strokes per minute, 1.5 to 1 dilution, and slope of 3½ to 4 in. per ft. Specific gravity and nature of solids, specific gravity and viscosity of solution and working temperature affect adjustments to such an extent that the above figures must only be considered as average, subject to change as great as 100 per cent in certain cases, particularly in the handling of chemicals.

Capacity. All other things being the same, such as speed, dilution, slope, separation, and materials handled, both overflow and raking capacity are proportional to classifier width. The finer the material to be raked, the slower must be the raking speed and the less the tank slope, both of which have the effect of reducing the capacity.

Example 2. When draining quartz of 2.7 sp. gr. from a solution of 1.0 sp. gr., making such a separation that the finest particle in the discharge is 20 mesh, the draining capacity is 750 tons per 24 hr. for a 3-ft. wide Simplex heavy Dorr Classifier, 1500 tons for a 6-ft. wide Duplex heavy machine, and 2000 tons for an 8-ft. wide Duplex heavy machine. When draining the same materials, but making such a separation that the finest particle in the discharge is 100 mesh, slope and speed must be reduced, giving draining capacities of 400, 800, and 1050 tons per 24 hr. for, respectively, 3-, 6-, and 8-ft. wide classifiers.

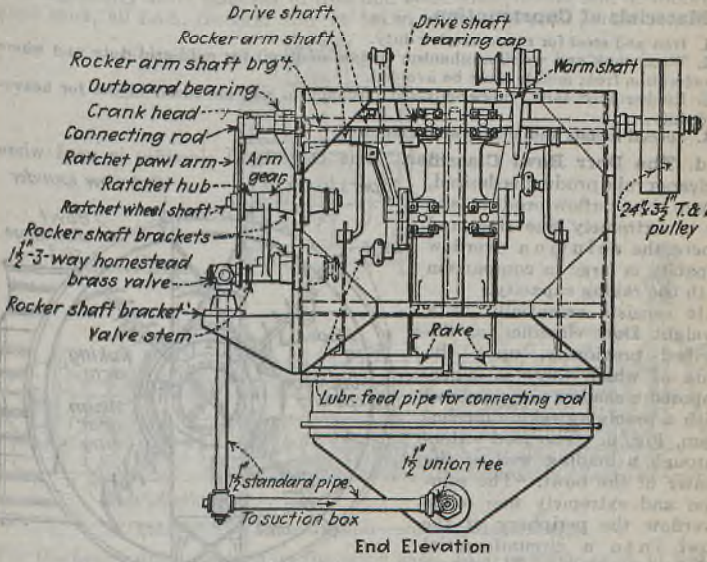


FIG. 5.—Suction box on Dorr classifier.

Chip-removing Device. In cases where the material to be drained is contaminated by chips of wood or other substances floating on the surface of the solution, a chip-removing device is supplied with the classifier, as shown in Fig. 4. Adjacent to the solution-overflow weir of the classifier, two troughs are placed. The one nearer to the weir is fitted with a screen to retain the chips carried over in the overflow. The one farther away serves as a receptacle for the chips which are brushed across the screen by a scraper.

Suction Box. The suction box, shown in Fig. 5, may be supplied with the classifier in order to reduce the moisture of the material below that point which is possible with ordinary gravitational draining.

It consists of a metal box attached to the under side of the drainage deck and equipped with a suitable screen or canvas to replace the portion of tank bottom directly above it. One port of a three-way valve is attached to a pipe leading from the bottom of the box, another port of the valve is connected to a source of vacuum, and the third is connected to a source of low-pressure air.

This valve is operated mechanically from the classifier mechanism by a pawl-and-ratchet device. Adjustments are provided so that the periods between vacuum and blowback may be varied at will. Filtrate is collected in a receiver on the vacuum line between the suction box and the vacuum pump. Where the filtrate consists of a saturated solution, salts precipitated on the canvas or screen may have to be removed by water or steam washes instead of with low-pressure air. The suction box generally reduces moisture in drained material 50 to 75 per cent.

Materials of Construction.

1. Iron and steel for non-corrosive duty.
2. Wood tank and wood mechanism (below solution) for mild-acid duty and where discoloration from metals must be avoided.
3. Rubber-lined tank, rubber-covered mechanism, and hard-lead blades for heavy-acid duty.
4. Special metals and alloys, where required.

d. The Dorr Bowl Classifier. The Dorr bowl classifier is used when a cleaner rake product is desired, where the overflow product is to be of extremely fine size, and where the solution overflow capacity is large in comparison with the raking capacity.

It consists essentially of a straight Dorr classifier, as described previously, upon the tank of which there is superimposed a shallow circular bowl with a revolving raking mechanism, Fig. 6. The feed enters through a loading well at the center of the bowl. The solution and extremely fine solids overflow the periphery of the bowl into a circumferential collecting launder. The coarse solids settle to the bottom of the bowl, are raked to the opening in the center of the bowl, and gravitate through this opening into the reciprocating-rake compartment below.

The coarse solids are advanced up the sloping bottom of the classifier and on to the draining deck from which they are presently discharged by the reciprocating rakes. Wash water, introduced near the center of the reciprocating-rake compartment, flows countercurrently with respect to the coarse solids and, after passing through the opening in the bowl bottom, leaves the classifier as a portion of the overflow product.

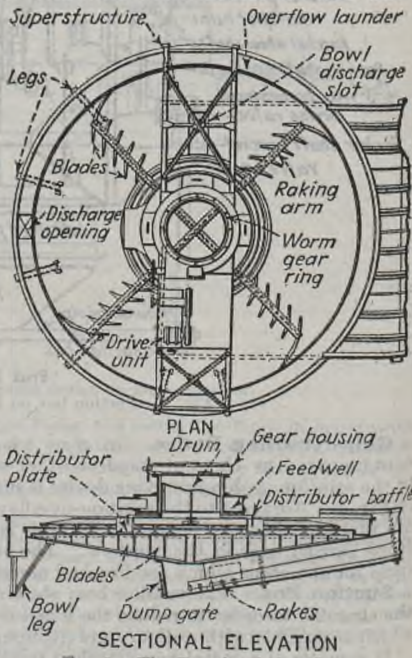


Fig. 6.—The Dorr bowl classifier.

Sizes. Bowls may be used with all types and sizes of Dorr classifiers, as listed previously. Bowls range in diameter from 3 ft. for use with a 15-in. wide classifier to 28 ft. for use with the larger units.

Power consumptions for the bowl alone, which must be added to that of the classifier with which it is used, range from less than ¼ h.p. for the small units to 2 to 3 h.p. for the largest.

Costs. Owing to the various types and sizes of Dorr classifiers furnished, as well as the variety of materials of construction which may be used for acid-resistant duty, sales prices per unit of sand-raking capacity vary widely. For rough and very preliminary estimating purposes only, the net sales price of a light-duty Dorr classifier of iron and steel construction and including a steel tank, all f.o.b. factory, may be taken at $350 to $500 per ft. of width. For the heavy-duty machine on the same basis, $450 to $950 per ft. of width may be used. Acid-resistant construction, such as wood, lead, rubber-covered steel, hard-lead or special alloys, materially increases the price.

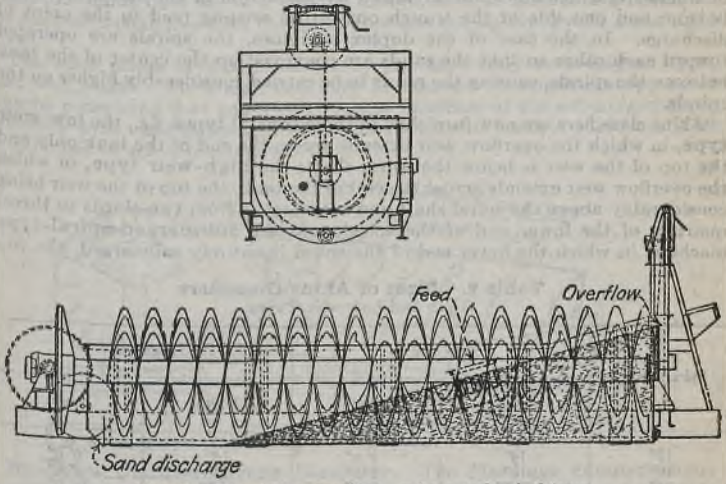


FIG. 7.—The Akins classifier—submerged-spiral type.

The sales prices of Dorr bowl classifiers vary even more than those of the regular, single-stage Dorr classifiers, since a wide range of bowl sizes may be used with each size and type of classifier. Accordingly, for rough and preliminary estimating purposes, it may be considered that a Dorr bowl classifier may cost from 50 to 200 per cent in excess of the price of the regular classifier to which the bowl is attached, the higher figure corresponding to the use of large bowls with narrow classifiers, and the lower figure covering cases where the bowl diameter is only slightly greater than the classifier width.

e. The Akins Classifier. The Akins classifier (Fig. 7) consists of an inclined trough or tank enclosing one or more revolving helixes, commonly called "spirals." The trough is equipped at its lower end with an overflow weir, and a hopper for collecting the overflow product which consists of the fine solids and water. The sand product is discharged at the upper end of this trough by the revolving spiral.

The sand-raking mechanism is made up of double spirals mounted on heavy hollow shafts, spirals being continuous from the overflow weir to a point above the feed entrance. At a point where the inner edges of the spiral blades emerge from the pool, a number of interruptions or depressions on the inner edge of the spiral blades are provided for a short distance to facilitate drainage of slimes and liquid back into the pool. Near the upper or sand-discharge end of the spiral similar interruptions or depressions are provided in the outer edge of the spiral flights. In any single-spiral machine, the sand load is conveyed up one side of the tank, leaving a drainage channel between the spiral and the tank on the opposite side.

The feed enters through a feed box at one, or, in some cases, both sides of the trough. The heavier, coarser solids settle to the bottom, and in the case of single-spiral machines are advanced by the action of the spiral along the bottom and one side of the trough out of the settling pool to the point of discharge. In the case of the duplex machines, the spirals are operated toward each other so that the sands are conveyed up the center of the tank between the spirals, causing the sands to be carried considerably higher on the spirals.

Akins classifiers are now furnished in three general types, *i.e.*, the **low-weir type,** in which the overflow weir extends across the end of the tank only and the top of the weir is below the spiral shaft; the **high-weir type,** in which the overflow weir extends across the end of the tank, the top of the weir being considerably above the spiral shaft and submerging from two-thirds to three-quarters of the lower end of the spiral; and the **submerged-spiral-type** machine, in which the lower end of the spiral is entirely submerged, the top

Table 2. Sizes of Akins Classifiers

High- and Low-weir Types

Spiral, diam.	Low weir		High weir	
	Width	Length	Width	Length
12″	14″	5′ 6″	14″	6′ 6″
16″	18″	5′ 6″	18″	8′ 6″
24″	2′ 3″	9′ 6″	2′ 3″	11′ 0″
30″	2′ 9″	11′ 0″	2′ 9″	13′ 0″
36″	3′ 3″	13′ 0″	3′ 4″	15′ 0″
45″	4′ 0″	14′ 0″	4′ 0″	17′ 3″
54″	4′ 9″	18′ 0″	4′ 9″	20′ 6″
60″	5′ 5″	19′ 6″	5′ 5″	22′ 0″
72″	6′ 8″	21′ 0″	6′ 8″	24′ 0″
78″	7′ 2″	22′ 0″	7′ 2″	24′ 0″

Submerged-spiral Types

Spiral, diam.	Single		Duplex	
	Width	Length	Width	Length
12″	14″	10′ 0″	26¾″	10′ 0″
36″	3′ 4″	22′ 8″	6′ 5½″	22′ 8″
48″	4′ 5″	26′ 0″	8′ 6½″	26′ 0″

of the overflow weir being from 12 to 24 in. above the lower end of the spiral and extending across the end of the tank and along both sides of the tank for a distance of from 2 to 5 ft., depending upon operating conditions.

Akins classifiers are generally set at an inclination of the tank of from 3 to 3½ in. per ft., although in some instances they are operating at inclinations up to 4 in. per ft. The speed at which the classifier spiral is generally driven is from 2 to 5 r.p.m., although it can be operated as low as ½ r.p.m. and up to 10 or 12 r.p.m. The power required will depend largely upon the size of the classifier and the sand-raking load, varying from less than 1 h.p. on the small sizes, up to 5 or 6 h.p. on the large machines with large sand-raking loads.

The tanks can be steam- or water-jacketed for temperature control, and where the fluids used are of a volatile nature, such as ammonia or gasoline, the tank or trough in which the spiral operates can be fitted with a gas-tight cover.

The sales prices of Akins classifiers vary materially according to type, size, and length. For rough and very preliminary estimating purposes, it may be considered that heavy-duty Akins classifiers of the submerged type, complete with steel tanks and lifting devices, f.o.b. factory, cost from $500 to $750 per ft. spiral diameter.

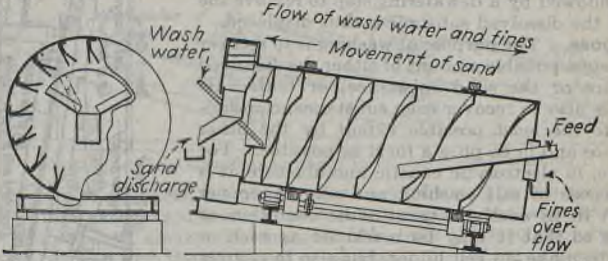


FIG. 8.—Principle of operation of the Hardinge countercurrent classifier.

Hardinge Countercurrent Classifier. The Hardinge countercurrent classifier (Fig. 8) is a slowly rotating drum, on the inner surface of which is located a spiral attached to the drum and revolving with it. The material to be classified is fed in at one end above the pulp level, and, as the classifier rotates, the coarser particles that settle out are moved forward by the spiral and are repeatedly turned over in a forward motion, releasing any fines mixed with them. The fines, with any wash water added, overflow through an opening at the opposite end of the classifier. The sand or oversize is dewatered and elevated by buckets to a higher elevation, so that the classifier may be operated in closed circuit with any suitable type of grinding mill and without the use of auxiliary conveyors or other equipment. The coarser settled solids are continually being turned over and washed by the countercurrent action of wash water.

Two steel tires near the end of the drum support the rotating drum on four rollers mounted on two parallel shafts, which in turn are driven by sprockets and chain from a variable speed motor.

Some of the uses of this classifier are for the closed-circuit grinding of various materials, dewatering sand, and the washing and scrubbing of various granular materials.

In addition to being supplied in standard iron and steel construction, it is also supplied in the chemical industry in stainless steel construction and with rubber lining for the handling of corrosive mixtures.

An interesting operating feature is that the classifier will start up under full load after several hours shutdown. The classifier is supplied in sizes varying from 18 in. to 10 ft. in diameter, and from 4 to 30 ft. in length.

The exact classification desired is obtained by varying the slope of the classifier, by varying the speed, or by varying the size of the overflow opening.

WASHING (GRANULAR MATERIAL)*

Definition. Washing of granular substances may be defined as the displacement of dissolved substances, adhering in solution form to a solid substance, by one or more applications of water or other suitable displacing agent, each application being followed by a dewatering step to remove the bulk of the dissolved substances thus displaced.

Purpose. The purpose of washing is to recover the greatest possible amount of either the dissolved substance or the solid substance, or both. Its object is also to recover each substance, contaminated to the least possible extent by the other substance and in as pure a form as possible. For example, in electrolytic caustic manufacture, it is the purpose of salt washing not only to recover the salt in as nearly a caustic-free condition as possible so that it may be marketed as such or reused to make up cell liquor, but also to recover the caustic soda solution diluted by wash solution to the least extent in order to promote good evaporator economy.

Equipment Used. The Dorr multideck classifier is the only standard, self-contained unit utilizing the alternate administrations of a displacing solution and of a draining process for the washing of soluble substances from insoluble granular substances. Centrifugation is applicable to the washing of such substances, but, since it is not classed under draining operations, it is discussed elsewhere (p. 1808).

The Dorr multideck classifier consists of a series of two or more Dorr classifier mechanisms connected together and driven from a common driving mechanism. A single tank is used, divided into from two to six individual washing compartments

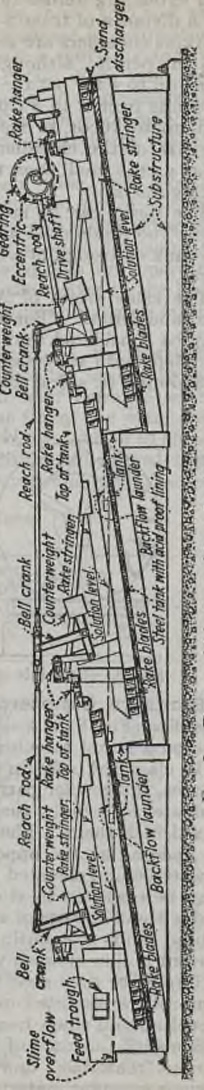


FIG. 9.—Dorr multideck washing classifier—acid-resisting type.

---

* See article on Countercurrent Decantation (p. 1643) and on Filtration (p. 1653) for washing of finely divided materials.

and drainage decks. As shown in Fig. 9, the various compartments are connected by backflow launders, located on the outside of the tank, so that wash solution may flow continuously through the series from the discharge to the feed end of the tank. The solid substance is advanced mechanically by the reciprocating rakes from the feed to the discharge end of the tank.

As shown in Fig. 10, two or more Akins classifiers may be arranged in tandem for countercurrent washing. The direction of flow of wash solution and solids is the same as in the Dorr washing classifier. As each washing operation is carried out in a separate classifier instead of (as in the case of the Dorr) in individual compartments of a single machine, external pipes or troughs are used for transferring the product between the different units.

Operation. The mixture of a solid substance and a solution is introduced at the overflow end of the tank, the washed solids are drained and discharged at the other end, and wash water or solution is introduced in the last compartment. The wash solution flows in a direction countercurrent to the solids and thus becomes progressively enriched in soluble salts until it finally overflows from the first compartment in relatively concentrated form.

In a similar manner the solids, being advanced toward the last compartment, come in contact with wash solutions progressively less concentrated. The mild agitation caused by the reciprocating rakes in each compartment assures efficient displacement of the soluble constituents in the solids which have just been discharged from the preceding compartment, and the draining on the inclined deck assures the removal of the bulk of the solution before the solids are remixed with the next weaker solution in the following compartment.

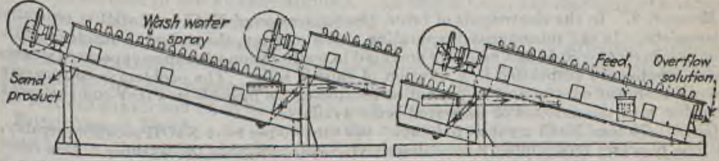


Fig. 10.—Akins classifiers arranged for multiple-stage washing.

Adjustments are as given for single-stage classifiers except that the slope is not variable. Rake speed determines the time the solids are retained in each washing compartment, and hence the capacity is generally determined by the chemical requirements of time for displacement of dissolved material rather than by the mesh at which the separation is to be made.

The greater the volume of the wash solution, the more complete is the displacement of dissolved material and the less concentrated is the solution overflowing the first compartment. When both high purity of washed solids and high concentration of solution are required, a larger number of compartments are required in order to increase the number of washes which may be given, by the permissible volume of wash solution, which is necessarily limited by the desired strength of the solution.

Capacity. Same as given for single-stage classifier of same type and width. Capacity may vary up to 100 per cent from figures given, which are quartz (2.7 sp. gr.) in water, due to variations in specific gravity, viscosity, temperature, and time required for displacement of dissolved substances. High speeds (greater than 12 r.p.m.) are seldom used on account of the time element in the displacement washing.

Accessories. Chip-removing devices and suction boxes may be supplied, these being identical with those discussed above. Steam coils may be placed in the compartments or in the solution launders between compartments to maintain proper temperature.

Materials of Construction. These are the same as for the single-stage Dorr classifier discussed on p. 1594.

Washing Calculations and Practical Examples. The method of washing crystalline or granular substances utilized in a multideck classifier is known as **continuous countercurrent decantation** (abbr. C.C.D.). The over-all washing efficiency, concentration of solutions in all compartments, and purity of final product may be determined mathematically, if data are available on feed characteristics, moisture content to which the solids will drain, wash water permissible, etc. The method of calculation is best illustrated by an actual example on a typical set of conditions.

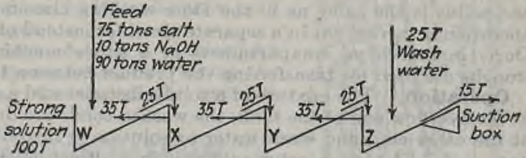


Fig. 11.—Washing classifier flow sheet.

Example 3. In the electrolysis of brine, the conversion of NaCl to NaOH is generally incomplete. In the subsequent evaporation of the solution, the sodium chloride crystallizes out, is recovered in salt catchers operated in conjunction with the evaporator stages, and is discharged suspended in a solution of caustic soda. The problem is to separate the salt from the caustic and to wash it as completely as possible in a four-compartment classifier with the amount of saturated brine available.

Given. 75 tons NaCl crystals per day. 100 tons 10 per cent NaOH solution per day. 25 tons of water (containing 8 tons dissolved NaCl) available for washing. One four-compartment washing classifier. One suction box mounted on last deck. Salt drains to 25 per cent moisture without vacuum. Salt drains to 15 per cent moisture with vacuum.

Problem. To find concentrations of solution in all compartments, washing efficiency, and percentage NaOH in washed salt.

Procedure. Draw diagrammatic flow sheet (Fig. 11 above) letting W, X, Y, and Z represent pounds of NaOH per ton water in each compartment and letting T represent tons of water in circulation at each point.

Equating pounds of dissolved NaOH out of and into each compartment, set up the following simultaneous equations:

$$\begin{aligned} 100W + 25W &= 35X + 10 \times 2000 && (1) \\ 35X + 25X &= 25W + 35Y && (2) \\ 35Y + 25Y &= 25X + 35Z && (3) \\ 35Z + 15Z &= 25Y + 25 \times 0 && (4) \end{aligned}$$

Solving: By substitution:

W = 194.51 lb. NaOH per ton water
X = 123.39 lb. NaOH per ton water
Y = 72.58 lb. NaOH per ton water
Z = 36.29 lb. NaOH per ton water

Summarizing.

Compartment	Solution Strength, % NaOH*
1	8.86
2	5.81
3	3.50
4	1.78

$$\text{Recovery of NaOH} = \frac{100 \times 194.51}{10 \times 2000} \times 100 = 97.26 \text{ per cent}$$

$$\text{NaOH in washed salt} = \frac{15 \times 36.29}{(75 \times 2000) + (15 \times 36.29)} \times 100 = 0.36 \text{ per cent}$$

* Salt in saturated brine-wash solution neglected.

From the preceding example, it should be obvious that the same method of calculation may be applied for finding other unknowns, *viz.:*

a. To find the minimum amount of wash solution which will give a washed product containing not more than a specified percentage of soluble impurities, when given the number of washing compartments to be used and the moisture content to which product will drain with and without suction box.

b. To find the minimum number of washing compartments required to give a specified percentage of soluble impurities in the washed product, when given the maximum permissible amount of wash solution and the moisture content to which the product will drain with and without suction box.

Other Chemical Applications. *a.* The washing of sodium phosphate crystals free from mother liquor.

b. The removal of finely divided clay or bond from artificially prepared abrasives such as carborundum, aloxite, etc.

c. The washing of phosphate rock to remove clay and fine sands.

LEACHING

Definition. Leaching may be defined as a process of removing, by the application of a solvent, that constituent of the substance being treated which is readily soluble in the solvent applied. In this article, the leaching of coarse, granular substances is alone considered.

Purpose. It is the purpose of leaching to recover the greatest possible amount of the soluble constituent in as concentrated a solution as can readily be handled by the succeeding extraction process, such as precipitation of the soluble materials or evaporation and crystallization.

Equipment Used. As is true in the case of washing, the Dorr multideck classifier is the only standard self-contained unit applying the principle of continuous-countercurrent flow of the substance to be leached and the solvent, to the leaching of granular substances. Discontinuous methods include the use of percolation vats, filter-bottom tanks, etc., but as these are not classed as draining operations in this article, they are not discussed here. The leaching of finely divided substances is discussed in the article on Countercurrent Decantation (p. 1643 and on p. 1215).

The Dorr multideck classifier used for leaching is described completely in the preceding section on Washing.

Operation. The substance to be leached is introduced into the first compartment at the overflow end of the tank, the residue remaining after leaching is discharged from the last compartment, solvent is introduced into the last compartment, and concentrated solution containing the soluble constituent passes off across a weir in the first compartment. The solvent flows in a direction countercurrent to the solids and thus becomes progressively enriched in the soluble constituent of the treated substance until it finally overflows from the first compartment in a relatively concentrated form.

In a similar manner, the substance to be leached being advanced toward the last compartment comes in contact with weaker and weaker solutions and is thus progressively impoverished in soluble constituents. The mild agitation set up in each compartment assures efficient penetration of solvent and good leaching of the substance discharged from the preceding compartment. The draining on the inclined deck assures the removal of the bulk of the enriched solvent before the substance is retreated with the next weaker solvent in the ensuing compartment.

Leaching Calculations and Practical Examples. The method of calculating the extraction, washing efficiency, concentration of solutions, etc., in a leaching or dissolving operation is similar to that used in washing calculations. The unknowns may be calculated from data available on feed characteristics, moisture content to which the residue will drain, the amount of lixiviant or solvent that may be used, etc.

Example 4. A calcine containing soluble copper is to be leached. The rate at which the copper dissolves having been determined experimentally and a multideck classifier having been selected to give the required time of contact in each compartment, it is desired to know the probable extraction of copper, the concentration of the solutions in the various compartments, and the percentage of copper remaining in the residue after treatment. The amount of wash water that may be used is limited by the capacity of the subsequent evaporator installation.

Given. One hundred tons of calcine containing 5 per cent (10,000 lb.) of soluble copper mixed with 100 tons of water; one six-compartment multideck classifier equipped with suction box; 50 tons of available wash water; and a residue that drains to 25 per cent moisture or to 15 per cent moisture with the aid of a suction box.

The following rates of dissolution of copper have been determined experimentally:

Pounds dissolved before reaching classifier	2000
Pounds dissolved in first compartment	4000
Pounds dissolved in second compartment	3000
Pounds dissolved in third compartment	1000
Pounds dissolved in fourth, fifth, and sixth compartments	0

Problem. To find the concentration of the solutions in all six compartments, the overall extraction efficiency, and copper content of the final residue.

Procedure. Calculate the amount of solids raked in each compartment, allowing for the dissolution of soluble copper that takes place in each. Calculate the amount of water advanced with the raked solids in each case, allowing for 25 per cent moisture in the first five compartments and 15 per cent in the last one.

Compartment A

$$\text{Solids raked} = 100 - \frac{(2000 + 4000)}{2000} = 97 \text{ tons}$$

Water in raked solids = 32.33 tons

Compartment B

$$\text{Solids raked} = 97 - \frac{3000}{2000} = 95.5 \text{ tons}$$

Water in raked solids = 31.83 tons

Compartments C, D, and E

$$\text{Solids raked} = 95.5 - \frac{1000}{2000} = 95 \text{ tons}$$

Water in raked solids = 31.66 tons

Compartment F

Solids raked = 95 tons
Water in raked solids = 16.76 tons

Draw the diagrammatic flow sheet shown in Fig. 12 and assign to each flow line figures to show the amount of water (not solids) in circulation at that point. Water enters at two points: 100 tons with the calcine in the first compartment, and 50 tons as wash water in the last compartment. The balance of the water in circulation is either overflow from compartments or moisture contained in the raked residues.

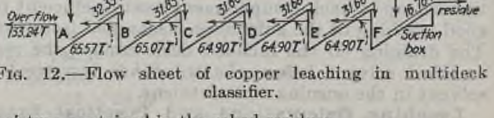


FIG. 12.—Flow sheet of copper leaching in multideck classifier.

Let A, B, C, D, E, and F represent the pounds of dissolved copper per ton of water in the respective compartments of the classifier. Then set up the following six simultaneous equations by equating the pounds of copper entering and leaving each compartment.

Equate pounds of dissolved copper entering and leaving each compartment as follows:

Compartment A	$133.24A + 32.33A = 65.57B + 6000$
Compartment B	$65.57B + 31.83B = 32.33A + 65.07C + 3000$
Compartment C	$65.07C + 31.66C = 31.83B + 64.90D + 1000$
Compartment D	$64.90D + 31.66D = 31.66C + 64.90E$
Compartment E	$64.90E + 31.66E = 31.66D + 64.90F$
Compartment F	$64.90F + 16.76F = 31.66E$

Simplifying,

$$\begin{aligned} 165.57A &= 65.57B + 6000 \\ 97.40B &= 32.33A + 65.07C + 3000 \\ 96.73C &= 31.83B + 64.90D + 1000 \\ 96.56D &= 31.66C + 64.90E \\ 96.56E &= 31.66D + 64.90F \\ 81.66F &= 31.66E \end{aligned}$$

Solving by substitution,

$A = 74.25$ lb. copper per ton water
$B = 96.10$ lb. copper per ton water
$C = 61.05$ lb. copper per ton water
$D = 28.60$ lb. copper per ton water
$E = 12.70$ lb. copper per ton water
$F = 4.92$ lb. copper per ton water

Checking calculations,

Total copper in classifier overflow $= 133.24 \times 74.25$	$= 9890$ lb.
Total copper in classifier discard $= 16.76 \times 4.92$	$= 82.5$
	9972.5 lb.
Error due to neglected decimals in slide-rule computations	$= 27.5$
Total copper in feed to classifier	$= 10{,}000$ lb.

$$\text{Recovery of copper} = 100 - \frac{16.76 \times 4.92}{133.24 \times 74.25 + 16.76 \times 4.92} \times 100$$

$$= 100 - \frac{82.5}{9972.5} \times 100 = 99.173\%$$

$$\text{Copper content of leached residue} = \frac{16.76 \times 4.92}{95 \times 2000 + 16.76 \times 4.92} \times 100 = 0.043\%$$

The same method of calculation may be used to find the minimum amount of water required for leaching to give a specified extraction or to find the minimum number of compartments to give a specified extraction with a specified amount of leaching solution.

CLOSED-CIRCUIT GRINDING (See p. 1932)

Closed-circuit wet grinding or classified grinding, introduced and perfected in the metallurgical industry, has been successfully applied to such chemical engineering operations as the grinding of lithopone, the grinding of phosphate rock with weak acid in phosphoric acid manufacture, the preparation of water-floated whiting, the wet grinding of cement slurry, abrasives, etc. A brief résumé of this subject may therefore properly be included in a discussion of classification and mechanical classifiers.

Definitions. [Dorr and Marriott, Importance of Classification in Fine Grinding, *Trans. Am. Inst. Mining Met. Engrs.*, Milling, pp. 109–154, 1930.] **Open-circuit grinding** is a method of comminution aiming to secure the desired reduction in particle size by a single passage of the material through the mill. **Closed-circuit grinding** is a method of comminution in which a partially finished mill discharge is separated by a classifier into a finished overflow product and an unfinished rake product which is returned to the mill for further grinding. **Overflow** is the comparatively finer, more slowly settling portion of the mill discharge which is carried over the tail board, or lip, of the classifier by the flow of water. **Rake product** is the comparatively coarser, more rapidly settling portion of the mill discharge which is discharged from the classifier by the mechanical action of the rakes. In closed-circuit grinding, the rake product is frequently referred to as *the circulating load* in that it travels in the mill-classifier circuit until reduced to overflow fineness. **Mill** is the generic term used to describe grinding mills, whether ball, pebble, or rod mills.

Purpose. It is the purpose of closed-circuit grinding to center the sizing of the ground product at one point, the classifier, so that the mill, now responsible only for grinding, may be fed at such a rate and loaded in such a manner that it may operate at maximum efficiency. As will be shown later, a judicious loading of the mill, unlimited by size specifications for the mill discharge, permits large savings in power, less wear on liners, and, less wear on grinding mediums, which three items determine largely the unit cost of grinding.

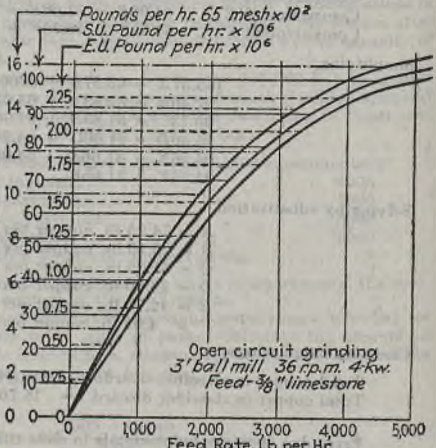


FIG. 13.—Relation between rate of feed and work done in a rotary mill.

Theory. Theory and practice concur in the hypothesis that the work done in a rotary mill increases with rate of feed. Work done is measured by the actual reduction in particle size from feed to discharge and may be measured by the tons of material of a specific size actually produced in a given time or by the more refined methods of Kick and of Rittinger, described later (see p. 1605).

Table 3. Relation between Rate of Feed to Ball Mill, Work Done, and Unit Power Consumption*

Feed rate, lb. per hr. of ⅜-in. limestone	Finished product in discharge, lb. per hr. of 65-mesh limestone	% Finished material in mill discharge	Rittinger's work units (S.U.)	Kick's work units (E.U.)	Kw.-hr. per ton of 65-mesh limestone
1,000	600	60	40 000.000	900 000	13.3
2,000	970	48.5	60 000.000	1,500.000	8.25
3,000	1,200	40	90 000 000	2,100,000	6.67
4,000	1,400	35	100,000,000	2,250,000	5.70
5,000	1,650	33	110,000,000	2,350.000	4.85

* Courtesy University of Minnesota.

This relationship is brought out in Table 3 and the accompanying graph (Fig. 13) based upon studies conducted at the Mines Experimental Station of the University of Minnesota, under the direction of E. W. Davis. It will be seen that the work done in the experimental mill increases as the feed is increased progressively to five times its original value, regardless of which of these methods is used to compute work, and in practice it has been found that the work curve is still increasing even when the rate of feed is increased to more than ten times the open-circuit capacity of the mill.

Work Units. Authorities appear divided as proponents of the Kick and the Rittinger methods of computing the work done in cylindrical mills. Rittinger's method is based upon the hypothesis that the work done in crushing is proportional to the increased surface produced, a function of the square of the diameter of the particle, and the Rittinger unit of measurement is called a "surface unit" (S.U.). Kick, on the other hand, claims that the work done is inversely proportional to the change in volume of the particle, a function of the cube of the diameter, and the Kick unit of measurement is known as an "energy unit" (E.U.). Practical plant operators prefer to measure work by the amount of material of a specific size actually produced, as this may readily be determined from screen analyses of feed and of discharge and the rate of new feed to the mill.

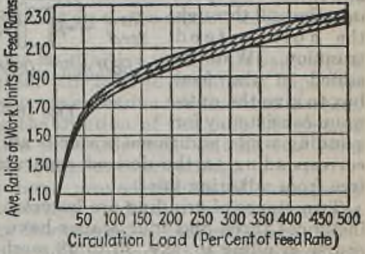


FIG. 14.—Relation between circulating load and work done in cylindrical mill.

Open- vs. Closed-circuit Grinding. The fact that capacity increases without a corresponding increase in power may be attributed to the more rapid elimination of fines, the reduction of uneconomical overgrinding, and the increased amount of material which may be exposed to the cascading action of the balls at one time. Theory indicates that uneconomical, open-circuit grinding is a result of hampering the work of the mill by the imposition of a specification for the fineness of discharge, which results in overgrinding of the bulk of the product in order that all may pass a given sieve size and in a tendency for the accumulated fines to act as a cushion, damping the effective impact of the balls upon the unfinished material.

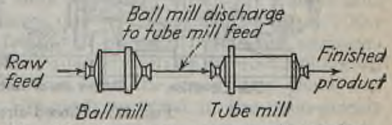


FIG. 15.—Two-stage open-circuit grinding.

In closed-circuit grinding, the classifier builds up the feed to the mill to the optimum value by returning unfinished oversize to the mill for further comminution. Furthermore, it grades the mill discharge so that only material of finished size may escape from the circuit as an overflow product. As shown in Fig. 14, the work done in a cylindrical mill increases with the amount of the circulating load or rake product.

Closed-circuit Grinding Equipment. Figure 15 is a diagram representing a two-stage open-circuit grinding installation, and Fig. 16 represents a two-stage closed-circuit grinding installation. In the case of the latter, it is to be noted that the amount of the circulating load in both ball- and tube-mill circuits is several times the amount of the new feed and finished product.

The closed-circuit grinding equipment consists of a cylindrical wet-grinding mill, a mechanical classifier, and two launders or troughs, the one conveying the mill discharge to the feed end of the classifier and the other conveying the oversize from the classifier to the feed box of the mill. The mill is equipped with a spiral scoop feeder, which picks up the new feed and the classifier discharge from its feed box and introduces them into the mill through the hollow feed trunnion. Water is added in the feed box to give the optimum consistency for grinding while additional water is added at the classifier to give the dilution corresponding to the desired separation and for washing the rake product free from adhering fines.

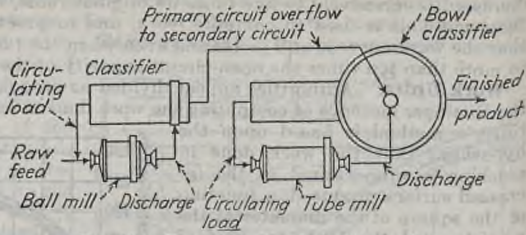


FIG. 16.—Two-stage closed-circuit grinding.

Two stages of grinding are becoming increasingly common while in certain instances three and four stages have been adopted profitably. In general, coarse grinding to, say, 35 to 48 mesh is carried out in ball mills (relatively large in diameter and short in length) loaded with balls ranging in size from 5 to 2 in. in diameter. Single-stage classifiers are generally used with ball mills.

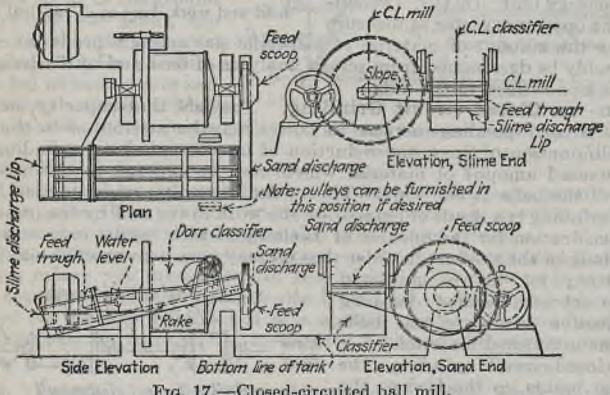


FIG. 17.—Closed-circuited ball mill.

Fine grinding to 100 to 325 mesh is generally accomplished in tube mills (relatively small in diameter and great in length) loaded with balls ranging in size from 2 to ¾ in. Bowl classifiers are generally used with tube mills.

The general arrangement of closed-circuit grinding equipment is shown in Fig. 17, representing a primary closed-circuited ball mill for relatively coarse grinding, 35 to 48 mesh, and Fig. 18, representing a secondary closed-circuit tube mill for relatively fine separations, 100 to 325 mesh.

CIRCULATING LOAD LIMITS

It has already been pointed out that the circulating load may profitably be built up to several times the new feed to the mill. That being the case, the question is frequently asked, "What circulating load ratio should be carried?" This is really an ambiguous question and one that will lead to great difficulty if not properly considered. What is actually required for good grinding is a proper loading of the mill. To bring this about, there should always be added to the new feed the correct amount of circulating load from the classifier to give the optimum tonnage through the mill. When the problem is considered in this manner, it takes on a new aspect, as the following will show:

In every grinding unit there is available an effective volume through which feed will pass, which, if properly utilized, will lead to maximum capacity. However, if the tonnage of total feed becomes excessive for the total volume, then the mill will "choke." Changing the dilution of the pulp in the mill will not remedy the congestion, for the available mill volume has been taxed beyond capacity, and the stoppage is due to overloading.

Thus, as data from actual practice have proved, the total tonnage of feed through the mill and not the circulating load ratio is the important consideration in obtaining best efficiency from closed-circuit grinding. Concerning actual mill loadings, a review of several important grinding applications which used the heavy-duty Dorr classifier in the circuit has shown that the total tons of feed to the grinding unit has fallen in the range of 14 to 20 tons per day per cu. ft. of mill volume.

Size Control. Increasing recognition is being given to the fact that in many cases the utility of a finely ground product is dependent upon, not one, but two things. These are (1) its degree of subdivision and (2) the size distribution of its particles. In other words, it is not enough to specify simply that a product be ground to, say, 90 per cent minus 200 mesh. Several samples of the same product may all pass 90 per cent through a 200-mesh screen; yet, if not ground under exactly the same conditions, these samples may have entirely different distributions of particle sizes. Hence there may be widely varying characteristics in the aggregate.

As a rule a properly adjusted closed-circuit system comes nearer to producing the mesh distribution desired than any other method. This is true because the closed-circuit system allows one to "bunch" the grind. For example, if we wish to grind a 1/4-in. feed to 97 per cent minus 200, open-circuit grinding will produce a much greater quantity of this as minus 325-mesh superfines than will closed-circuit grinding. At the same time there will certainly be considerably more plus 150-mesh material in the open-circuit product.

Fineness of Grinding. The operating data tabulated below are from the portland cement industry and give a comparison of size distributions and fineness of grinding obtained by grinding in open and closed circuit. At the plant where these results were obtained, a year's operation with open-circuit grinding gave an average fineness of 89 per cent minus 200 mesh and 5 per cent plus 100 mesh. Under closed-circuit conditions the average fineness has been increased to 97.5 per cent minus 200 mesh and nothing on 100 mesh, while at times, when handling only 50 tons per hour per mill, the fineness has been held at 99.9 per cent minus 200 mesh and 95 per cent minus 325 mesh.

The table below shows the distribution of sizes as determined by screen analyses, elutriation tests, and microscopic examinations of particle sizes.

Table 4. Comparison of Size Distribution

	Mesh	Open circuit, % + % cum.		Closed circuit, % + % cum.	
Sieve No.:					
1	20	1.0	1.0		
2	28	0.25	1.25		
3	35	0.50	1.75		
4	48	0.50	2.25		
5	65	0.75	3.00		
6	100	2.00	5.00		
7	200	6.00	11.00	2.6	2.6
8	325	(Not recorded)		8.4	11.0
Elutriation jar:					
1	340*	10.7	21.7	16.6	27.6
2	540*	7.8	29.5	5.9	33.5
3	820*	3.4	32.9	6.7	40.2
4	1400*	8.4	41.3	8.0	48.2
Overflow..............	....	58.7	....	51.8	

* Average particle size determined by microscopic examination.

A study of these particle-size determinations indicates (1) an elimination of stray oversize, coarser than critical size, (2) a substantial reduction in plus 200-mesh material, and (3) less superfine as indicated by final elutriator overflow.

Another size analysis that shows how the substitution of closed circuit for open circuit results in a reduction in plus 150-mesh material and also in the superfines, is shown in Table 5. This size analysis was made by means of an Oden sedimentation balance, a device for making very accurate size determinations in the micron range. The open-circuit analysis is of a month's composite sample of cement slurry ground by the open-circuit method at a well-known cement mill. The closed-circuit sample was taken from closed-circuit grinding operations at the same mill.

Table 5. Oden Size Analysis of Cement

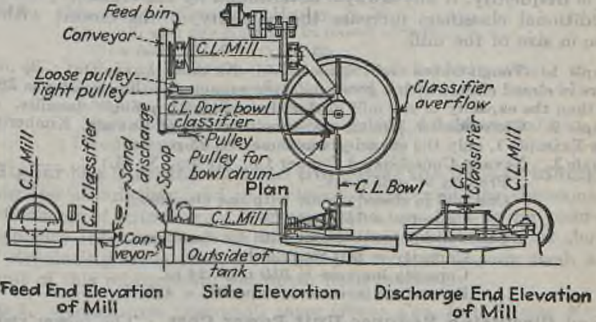
Diameter, microns	Equivalent sieve mesh	Open circuit, % cum.	Closed circuit, % cum.
295	48	0.32	
208	65	0.68	
147	100	2.18	0.08
104	150	4.10	0.38
74	200	7.85	3.75
53	325 dry	13.8	15.6
42	325 wet	15.9	19.15
40	......	19.3	23.3
30	......	23.5	31.0
25	......	27.6	34.7
20	......	33.1	39.4
15	......	39.9	44.9
10	......	48.5	51.7
7.5	......	53.4	57.0
6.0	......	57.6	60.5

Savings. At the cement plant from which the above data were obtained the following savings in grinding power and grinding media consumption were recorded:

Table 6. Savings in Power and Grinding Media

Consumption per bbl.	Open circuit	Closed circuit	Saving, %
Power, kw.-hr.	5.5	3.01	45.3
Primary media, lb.	0.086	0.057	33.7
Secondary media, lb.	0.321	0.112	65.1

Automatic Control. A point worth noting is that in closed-circuit grinding the classifier gives a control that automatically takes care of such small fluctuations in character and rate of feed as would otherwise cause variations in the finished product. Harder feed, for example, simply increases the circulating load.



Feed End Elevation of Mill   Side Elevation   Discharge End Elevation of Mill

Fig. 18.—Closed-circuited tube mill.

Metallurgical Operating Data

The operating data given below, taken from metallurgical practice, are arranged to emphasize certain important facts relating to closed-circuit grinding.

EXAMPLES OF CLOSED-CIRCUIT GRINDING IN MINING INDUSTRIES*

Closed Circuiting Increases Mill Capacity. When grinding a substance so that all of it shall pass a screen of a given size, the capacity of the mill may be increased by closed circuiting it with a classifier.

Example 1. Mill = 6 by 20 ft.
Feed = −6 mesh.
Product = 8 per cent + 100 mesh.
Capacity, open circuit = 144 tons, 24 hr.
Capacity, closed circuit = 240 tons, 24 hr.
Capacity increase = 96 tons, 24 hr.

---

* Anable, Closed Circuit Fine Grinding and What It Should Accomplish in the Cement Industry, *Rock Products*, Jan. 5, 1929.

Percentage increase in capacity = 66.
—Taggart, "Handbook of Ore Dressing," p. 455, Wiley, New York, 1927.

Example 2. Wright-Hargreaves Gold Mines, Kirkland Lake, Ont.:
Product = −200 mesh.
Capacity, open circuit = 190 tons, 24 hr.
Capacity, closed circuit = 275 tons, 24 hr.
Capacity increase = 85 tons, 24 hr.
Percentage increase in capacity = 44.7.

Example 3. Phelps-Dodge Corporation, Morenci, N. M.:
Product = −65 mesh.
Capacity, open circuit = 89 tons, 24 hr.
Capacity, closed circuit = 174 tons, 24 hr.
Capacity increase = 85 tons, 24 hr.
Percentage increase in capacity = 95.5.

Classifier Determines Capacity, Not Mill. The capacity of the closed circuit is frequently, if not always, determined by the classifier, not the mill; *i.e.*, additional classifiers increase the capacity of the circuit without any increase in size of the mill.

Example 1. Tough-Oakes Gold Mines, Ltd, Kirkland Lake, Ont.: By using two classifiers in closed circuit with a given mill, the capacity of the circuit was 28 per cent greater than the capacity of the mill in closed circuit with a single classifier.

Example 2. Consolidated Mining and Smelting Co. of Canada, Kimberly, B. C.: Same as Example 1, only the capacity was increased 35 per cent.

Example 3. Nevada Consolidated Copper Co., Hurley, N. M.:
Product = −65 mesh.
One mill in closed circuit with one classifier:
Capacity = 150 tons, 24 hr.
One mill in closed circuit with six classifiers:
Capacity = 800 tons, 24 hr.
Capacity increase = 650 tons, 24 hr.
Percentage increase in capacity = 433.

Closed Circuiting Reduces Unit Power Cost. The power required to drive a given mill remains practically constant regardless of tonnage fed; and, accordingly, increased capacity, due to closed circuiting, results in diminished power costs per ton finished product.

Example 1. Lake Shore Mines, Kirkland Lake, Ont.:
Percentage closed-circuiting increased capacity = 44.70.
Power consumption was reduced 10 per cent (due, no doubt, to better balance with heavy feed).
Percentage reduction in power per ton of finished product = 37.

Example 2. Lucky Tiger Mine:
Mill = 5 by 14-ft. tube.
Feed = 11 to 20 per cent + 20 mesh.
Finished product = −100 mesh.
Power was 47 h.p. throughout tests.
Capacity, open circuit = 22 tons, 24 hr.
Capacity, closed circuit = 37 tons, 24 hr.
Tons of −100 mesh, per h.p.-hr., open circuit = 0.016.
Tons of −100 mesh, per h.p.-hr., closed circuit = 0.055.
Percentage unit power cost reduced = 71.
—Taggart, "Handbook of Ore Dressing," p. 456, Wiley, New York, 1927.

Example 3. Mill = 6 by 20-ft. tube.
Feed = −6 mesh.
Finished product = 100 per cent + 100 mesh.
Capacity, open circuit = 144 tons, 24 hr.

Power consumption, open circuit = 75 kw.
Tons of 10 per cent + 100 mesh per h.p.-hr. = 0.0565.
Capacity, closed circuit = 240 tons, 24 hr.
Power consumption, closed circuit = 65 kw.
Tons of 10 per cent + 100 mesh per h.p.-hr. = 0.1087.
Percentage unit power cost reduced = 48.
—Taggart, "Handbook of Ore Dressing," p. 455, Wiley, New York, 1927.

Closed Circuiting Reduces Wear on Liners and Balls. Through a better loading of the mill with coarse classifier oversize, the abrasion of metal liners and the consumption of grinding mediums are greatly reduced.

Example 1. Lake Shore Mines, Kirkland Lake, Ont.:
Consumption of steel per ton finished product:
Open circuit = 6.5 lb. per ton.
Closed circuit = 3.2 lb. per ton.
Percentage reduction in steel loss = 51.

Example 2. Chino Copper Company, Hurley, N. M.:
One mill and one classifier in closed circuit:
New feed = 150 tons, 24 hr.
Ball and liner wear = 3.2 lb. per ton finished product.
One mill and six classifiers in closed circuit:
New feed = 240 tons, 24 hr.
Ball and liner wear = 1.5 lb. per ton of finished product.
Percentage reduction in steel loss = 53.

The Higher the Circulating Load the Lower the Unit Grinding Cost. Increasing the circulating load of a closed-circuit mill has the same effect on the efficiency of grinding as increasing the new feed to an open-circuit mill. Capacity increases and all unit costs decrease throughout the entire range of circulating loads from 0 to 1100 per cent of the new feed, and the upper limit of this relationship has never been reached.

Example 1. Quoting from Oughtred's paper (*Trans. Can. Inst. Mining Met.*, 1928, p. 310) on the Sullivan concentrator of the Consolidated Mining and Smelting Company, Kimberly, B. C.: "An abnormally high circulating load is maintained, consistent with the mechanical limitation of the machines. Normal circulating load at the present time ranges from 1000 to 1100 per cent, or an equivalent of 3000 tons of sand per standard classifier."

Example 2. Quoting from the report of high-circulating-load tests at the Nevada Consolidated Copper Company, Hurley, N. M.: "Only the structural limitations of Section 7 prevented us from obtaining the ultimate capacity of a ball mill in these tests, but we learned enough to discover that we could reduce the cost of producing −65-mesh flotation feed from around 20 cts. to about 5 cts. per ton by using all the classifiers on a single mill. We definitely learned that 3500 tons per day is not too great a feed for the above conditions."

(Note: The ball mill referred to was 7 ft. in diameter by 10 ft. long.)

Summing up, it may be stated that in the field of metallurgy, both theory and practice agree on the desirability of closed-circuiting wet-grinding mills with classifiers, since closed-circuit operation increases capacity, reduces unit power cost of grinding, cuts down the wear on liners and grinding mediums and in general permits the mill to be loaded to its maximum output of material of a specified size. The above advantages have the cumulative effect of reducing the unit cost of grinding, controlling accurately the size of the maximum particle in the finished product, and limiting the tendency to overgrind a large portion to semicolloidal size.

Operating Costs. The unit power consumption for fine closed-circuit grinding at metallurgical plants varies somewhat, depending upon the degree to which the ore is crushed before grinding and the fineness of the overflow from the classifiers.

Studies made at eight plants have yielded the data presented in the table below. All these mills crushed their ore to ¼ to ½ in. before grinding. Expressing the power consumption in terms of tons of 100-mesh and tons of 200-mesh material actually produced eliminates any error which otherwise might be introduced by different classifier settings. Most of these plants are making separations at from 48 to 65 mesh.

Table 7. Unit Power Consumption for Fine Grinding in Western Copper Plants*

Plant	Kw.-hr. per ton, 100 mesh	Kw.-hr. per ton, 200 mesh
A	7.90	10.80
B	8.53	9.73
C	8.95	11.66
D	9.20	11.69
E	10.10	13.10
F	10.61	14.56
G	10.74	14.80
H	11.93	11.69
Average........	9.74	12.25

* Anable, Cement Industry Investigates Metallurgical Grinding Methods, *Eng. Mining J.*, vol. 129, No. 4, p. 188, Feb. 24, 1930.

The gradual wearing out of mill liners and steel balls represents an appreciable item of the operating cost. Table 8 below gives some practical operating data on this point.

Table 8. Consumption of Liners and Grinding Media at Closed-circuit-grinding Installations in Metallurgy

	Primary circuit	Secondary circuit	Total
Lake Shore Mines, Kirkland Lake, Ont..................	.....	.....	3.2 lb. per ton
Chino Copper Company, Hurley, N. M.................	.....	.....	3.2–1.5
Cananea Consolidated Copper Co., Sonora, Mex.........	0.459	1.283	1.742
Phelps-Dodge Corporation, Bisbee, Ariz.................	.....	.....	3.72

The cost of replacing media and liners amounts to 80 to 90 per cent of the maintenance cost of cylindrical mills. **Grinding media,** such as **balls, rock,** or **flint pebbles,** wear much more rapidly than the steel-mill liners, probably in the ratio of about 3.5 to 4.5:1. **Forged-steel balls** cost from 2.5 to 3.25 cts. per lb., the higher price being for the smaller sizes. **High-carbon steel rods** sell at 0.25 ct. per lb. over the base price for merchant bars which usually range from 2.5 to 2.75 cts. per lb. **Manganese-steel liners** cost from 8.5 to 11.5 cts. per lb., depending upon the size and weight of the individual castings, the higher price being for the small, light castings.

Cement-slurry Grinding

In this industry, open-circuit two-stage grinding has been standard practice for many years. Combination or compartment mills are generally used. The mill is divided into two portions by a vertical, slotted grid, the primary compartment being loaded with 2- to 5-in. balls and the secondary with ¾- to 1-in. balls. Recently, however, many wet-process cement mills have adopted two-stage closed-circuit grinding, either using the existing combinations or compartment mills or preferably separate ball mills. Four mills of this type are in operation and several more under consideration. The following description relates to the mill of Universal Atlas Cement Company at Leeds, Ala., as reported by Counselman in *Tech. Pub.* 1096, *Am. Inst. Mining Met. Engrs.*, 1939.

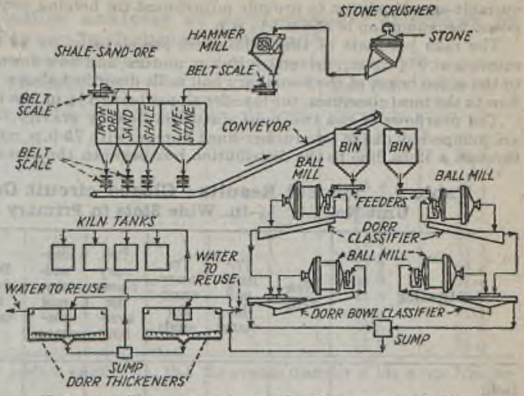


FIG. 19.—Two-stage closed circuit cement grinding

CLOSED-CIRCUIT GRINDING CEMENT RAW MATERIALS AT LEEDS

After several years study, the Universal Atlas Cement Co. decided in 1937 to rebuild its plant at Leeds, Alabama. The entire old plant, which was to operate during the new construction, was then to be scrapped with the exception of two kilns. The quarry was to be mechanized, new and larger kilns installed, with the most modern type of equipment all through the plant.

Two-stage, closed-circuit grinding (Fig. 19) was decided upon to reduce 1-in. material to 200 mesh. In selecting the ball mills, it was felt desirable to have the primary and secondary mills of exactly the same size, so that all parts would be interchangeable. After study of the grindability tests, mills were specified to be 9 ft. diameter inside the liners by 8 ft. long from the head liner to the grid plate. Throat and discharge trunnions are 22 in. and 24 in. clear diameter. The feed scoops are 4 ft. 0 in. radius. The discharge grates of the primary mills have slots ⅞ in. wide inside, tapering to 1 in. outside, and the plates are 1½ in. thick. On the secondary mills the tapered slots are ⅜ in. wide inside by ½ in. wide outside, same plate thickness. These slots extend inward for 13 in. from the inside of the liners toward the axis of the mill, the central portion being blanked off. All of the material passing through the mill must therefore pass through these slots near the periphery, so that these are low-level mills. Internal radial lifters inside the discharge head raise the mill product to the discharge trunnion, and are so arranged that there is no spillback through the slots. The mills have water-cooled, babbitted bearings.

The primary mills rotate at 19 r.p.m. and are fed heat-treated steel balls 4 in. in diameter, of about 500 Brinell. The secondary mills run at 17 r.p.m. and 2-in. balls are fed as make-up. Mixed charges were put in the mills at the start. Motors are 350 h.p., 180 r.p.m. synchronous, 2200 volt, capable of 100 per cent overload at 40°C. temperature rise.

The discharges from the primary mills flow by gravity at a slope of $1\frac{3}{16}$ in. per foot to 8 ft. by 25 ft. 6 in. classifiers with 20 h.p. motors. Actual power consumption is about 12 h.p. Sands are conveyed to the primary ball-mill scoop boxes by vibrating launders having 7½ h.p. motors.

The overflows of the primary classifiers go by gravity, in a launder having a slope of 2 in. per foot, to the bowl classifiers, which are 16 ft. wide by 39 ft. long by 24 ft. in diameter, and any finished material overflows without getting into the secondary ball mills. The rakes of these machines are driven by 20-h.p., constant-speed motors, actually taking 13 h.p. with normal load. The bowl is driven by a 10-h.p. motor with a variable-speed reducer to provide adjustment for helping control the fineness. Actual power consumption is about 6½ h.p.

The rake products of these classifiers are elevated by 12 ft. diameter sand wheels, rotating at 9½ r.p.m., driven by 10-h.p. motors, and flow down a slope of 4½ in. per foot to the scoop boxes of the secondary ball mills described above. The ball-mill discharges flow to the bowl classifiers, the launder slopes being 1⅝ in. per foot.

The overflows of the two bowl classifiers flow by gravity to a sump, and from there are pumped by one of two rubber-lined pumps, with 75-h.p. motors at 2500 gal. per min., through a 12-in. line to the distribution box between the two thickeners, the underflows

Table 9. Typical Results—Closed-circuit Cement Grinding

Unit No. 1.* ⅞-in. Wide Slots in Primary Mill Grates

Opening	New feed	Primary mill discharge	Primary classifier sands	Primary classifier overflow	Secondary mill discharge	Bowl classifier sands	Bowl classifier overflow	Primary circulating load	Secondary circulating load
Inch:									
+1	2.6								
+¾	5.4	0.3	1.2						
+½	10.7	3.7	3.5						
+⅜	23.0	6.8	7.1						
Mesh:									
+ 4	45.7	13.2	16.1						
+ 5	48.3	14.5	18.2						
+ 8	62.5	21.9	32.1	....	.....	.....	....	214	
+ 14	74.0	37.2	58.7	1.6	.....	0.4	....	166	
+ 28	84.5	61.2	86.5	18.6	2.1	7.4	....	168	351
+ 48	89.0	75.0	94.0	37.4	15.1	26.1	....	198	340
+100	92.2	81.6	96.2	52.1	48.8	64.0	....	202	342
+200	94.0	85.4	97.0	61.0	78.9	92.6	13.0	210	351
+325	....	.....	.....	....	.....	.....	22.4		
Specific surface	....	.....	.....	....	.....	.....	3792	Best	Best
Tons per hour	54	164	110	54	187	187	54	avg.	avg.
H_2O, per cent	2.3	23.8	24.5	50.1	35.6	23.4	83.9	203	346

Unit No. 2. ⅜-in. Wide Slots in Primary Mill Grates

Opening	New feed	Primary mill discharge	Primary classifier sands	Primary classifier overflow	Secondary mill discharge	Bowl classifier sands	Bowl classifier overflow	Primary circulating load	Secondary circulating load
Inch:									
+1	2.6								
+¾	5.4								
+½	10.7	.....	0.2						
+⅜	23.0	0.6	0.9						
Mesh:									
+ 4	45.7	3.5	7.8						
+ 5	48.3	4.0	8.9						
+ 8	62.5	9.2	18.9	....	.....	.....	....	95	
+ 14	74.0	20.6	41.4	0.6	.....	.....	....	96	
+ 28	84.5	45.0	76.3	13.6	1.0	4.9	....	100	
+ 48	89.0	60.5	89.1	32.6	8.0	22.6	....	98	223
+100	92.2	71.3	93.3	48.4	37.4	62.1	....	104	196
+200	94.0	76.6	95.2	58.9	69.2	91.5	10.1	95	210
+325	....	.....	.....	....	.....	.....	19.2		
Specific surface	....	.....	....	....	.....	.....	4093	Best	Best
Tons per hour	50	100	50	50	102	102	50	avg.	avg.
H_2O, per cent	2.3	32.8	21.6	55.2	34.2	30.5	84.5	100	203

* Approximately 6000 lb. less ball load than No. 2 unit.

of which are pumped to feed tanks ahead of the kilns. Five tables of operating data follow, giving typical closed circuit grinding results as of Nov. 19, 1938.

Table 10. Sedimentation Analyses of Bowl-classifier Overflows by Wagner Turbidimeter Method*

Particle size	Unit No. 1	Unit No. 2
Mesh:		
+200	13.0	10.1
+325	22.4	19.2
Microns:		
−60 +55	23.4	20.1
+50	25.5	22.6
+45	28.8	25.6
+40	31.2	28.3
+35	33.2	31.2
+30	37.3	33.2
+25	40.9	37.4
+20	44.8	40.7
+15	49.6	45.7
+10	56.6	52.4
+ 7.5	61.5	58.0

* It is assumed, for specific surface calculations, that the average diameter of the minus 7.5μ particles is 2.6μ.

Table 11. Tonnages and Power Consumptions

	Test data, tons per hr.		Power readings, kw.	
			Unit No. 1	Unit No. 2
Belt-scale settings:		Primary ball mills	290	310
Limestone	91.0	Secondary ball mills	305	305
Shale	15.0	Primary classifiers	9	9.5
Sandstone	2.2	Bowl-classifier rakes	10	9.5
		Bowl-classifier bowls	3.8	3.8
		Thickener, rotation	1.5 for both units	
		Kw-hr. per barrel	3.58	4.0

Table 12. Power Consumptions per Barrel* of Cement

Equipment	Kw.	Kw.-hr. per bbl.
Primary mill	290	3.44
Secondary mill	305	
Primary classifier	9	
Bowl classifier:		
Rakes	10	0.14
Bowl	3.8	
Thickener	1.5	
Total power	....	3.58

* 625 lb. of raw materials—limestone, shale, and sandstone—are required to make a barrel of finished cement.

POWER REQUIREMENTS

In addition to more uniform grinding, one of the principal advantages of closed-circuit grinding is saving in power. At most plants having open-circuit wet grinding in a two-compartment mill, the power required for the mills alone for raw-grinding enough material to make a barrel of cement (the relative grindability of the mix being almost the same as at Leeds) amounts to about 7.5 kw.-hr. At Leeds, when grinding in one unit 54 tons per hour, equivalent to 173 bbls., the power readings were 3.58 kw.-hr.,—a saving of 3.92 kw.-hr. per barrel, or, on a 4000 bbl. a day basis, a saving of 15,700 kw.-hr. per day.

The power consumption of 3.58 kw.-hr. per barrel, for reduction from 1 in. to 200 mesh, is equivalent to 11.45 kw.-hr. per ton. This can be broken down to a consumption of 5.55 kw.-hr. per ton, for reduction from 1 in. to 14 mesh, and 5.90 kw.-hr. per ton for reduction from 14 to 200 mesh. This compares very favorably with metallurgical practice.

Whiting, Lithopone, and Abrasive Grinding

Whiting (French, Cuban, English, or domestic chalk), lithopone (chemically precipitated $BaSO_4.ZnS$) and abrasives (Aloxite, Carborundum, etc.) used to be ground and sized by relatively crude intermittent methods. The substance was first wet ground to a fineness approaching that desired for the final product. The charge was withdrawn from the mill in the form of a thin paste or slurry, and the suspended solids were then sized by diluting the paste or slurry with water and allowing it to flow through a series of tanks of various sizes, the first in the series being the smallest and the last, the largest. In certain cases, especially in the abrasive industry, as many as a dozen or more tanks are frequently used. The material overflows from each tank into the next succeeding one and, owing to different settling periods provided, the coarsest and most quickly settling material settles in the first and smallest tank, the finest and most slowly settling material in the last and largest tank, and the intermediate sizes are collected in the tanks between the two extremes.

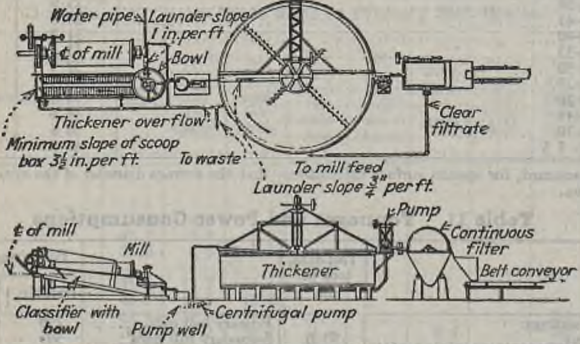


FIG. 20.—Wet closed-circuit grinding of whiting.

In certain cases, the materials are dry ground and sized pneumatically. Dry treatment, however, is outside the scope of this section.

Whiting Classification. The tendency toward the production of but a single grade or fineness of product in preference to variously sized products led to the development of the arrangement shown in Fig. 20 for the preparation of −300-mesh, water-floated whiting. Crushed chalk is ground in a tube mill in closed circuit with a bowl classifier. Classifier overflow is thickened in a continuous thickener, thickener discharge is dewatered on a continuous vacuum filter, and filter cake is dried in a dryer of the rotary-kiln type. In certain plants, mullers, *i.e.*, round or square tanks with revolving stones running on their edges upon the bottom, supplement or entirely replace the cylindrical mills.

Tables 13 and 14 give certain operating data from two installations of this general type.

Table 13. Plant A. 31 Tons Whiting per Day

	Dilution of water to solid	Screen analysis—cumulative %					
		+65	+100	+150	+200	+325	−325
Bowl-classifier feed	6.1:1	0.9	1.7	3.5	5.0	7.8	92.2
Bowl-classifier overflow	10.1:1	....	....	....	0.5	2.5	97.5
Bowl-classifier rake product	1:1	11.7	22.4	41.6	58.4	73.3	26.8
Thickener discharge	1.07:1	....	....	....	0.5	2.5	97.5
Vacuum-filter discharge	0.33:1	....	....	....	0.5	2.5	97.5

Table 14. Plant B. 40 Tons Whiting per Day

	Dilution water to solid	Screen analysis—cumulative per cent					
		+65	+100	+150	+200	+325	−325
Bowl-classifier feed	12.6:1	..	1.4	1.8	2.3	4.8	89.7
Bowl-classifier overflow	16.7:1	..	....	....	0.6	4.1	95.9
Bowl-classifier rake product	......	11	24.2	50.3	70.9	86.4	13.6
Thickener discharge	1.1:1	..	....	....	0.6	4.1	95.9

Lithopone Classification. Finished lithopone, after calcination and quenching, is generally reduced to the desired fineness by wet grinding in a tube mill in closed circuit with a classifier and a hydroseparator. The principle of operation is the same as that used in the closed-circuit grinding of metallurgical ores, cement slurry, and whiting, but the product is ground to a much finer degree.

Since discoloration of the pigment must be avoided at all cost, the classifying equipment must be constructed of wood or such special, non-tarnishing alloys as aluminum-zinc. Instead of using a bowl classifier, it is the practice to use a standard classifier of wood construction in direct-closed circuit with the tube mill and a hydroseparator (an undersized thickener) also constructed of wood for reclassifying the classifier overflow. This arrangement is equivalent, in so far as results are concerned, to a tube-mill bowl-classifier layout and may more easily be provided in acid-resisting construction.

The tube mill, lined with silex blocks and loaded with flint pebbles, receives the quenched lithopone and discharges to the classifier. The classifier overflow is pumped to the hydroseparator for reclassification, and the classifier rake product is returned to the mill for further grinding in the conventional manner.

The hydroseparator, a shallow thickener insufficiently large to settle all of the lithopone, overflows the finest material, and discharges a pulp containing intermediate-sized lithopone which is pumped back to the tube mill for regrinding with the classifier returns. The fine, hydroseparator overflow is dewatered in a thickener, filtered on vacuum filters or presses, dried, and packed for shipment. The pulp and solutions are generally maintained at a temperature of about 140° to 150°F. in order to facilitate the making of the fine separations required.

The following data were obtained at one of the plants grinding lithopone in this manner:

Table 15. Making 350-mesh Lithopone

	Dilution water to solids	% + 350 M
Feed to tube mill	1.5:1	7.0
Tube-mill discharge	3.5:1	4.5
Classifier overflow	10 :1	2.6
Hydroseparator overflow	20 :1	0.25

Abrasives Classification. In the preparation of finely divided abrasives, such as Carborundum, Aloxite, Alundum, etc., wet grinding and classification are generally employed. Experience has shown that hydraulic classification into a variety of different sizes is greatly simplified if the feed to the series of hydraulic cones has first been classified so as to remove, first, the very coarse grains and, second, the very fine—almost colloidal—grains. Apparently the presence of these end sizes interferes with the proper regulation of the hydraulic cones.

To meet this condition, a unique closed-circuit grinding layout has been adopted by four of the large abrasive companies. The crushed abrasive is ground in a ball mill, operating in closed circuit with a bowl classifier. All oversize material is eliminated at this point and returned to the ball mill for further grinding. The overflow from this first bowl is pumped to a second bowl classifier which overflows to waste those superfine grains which are injurious to the operation of the hydraulic cones. The rake product of this second classifier is the finished product and contains a range of sizes of grain which may later be easily separated into the desired commercial grades.

These abrasive installations are unique in that the second bowl classifier makes a separation around the equivalent of 600 to 700 mesh. So delicate is the adjustment, that sometimes a dispersing agent, sodium silicate for example, is used, especially if the plant's water supply contains a trace of acid or other flocculating agent.

The following operating data are typical of results secured at a fine abrasive plant producing as finished product grains ranging in size from 0.10 to 0.005 mm.:

Table 16. Moisture Determination and Microscopic Analyses of Products in Classification Circuit

Product sampled	% solids	Dry lb per hr.	Approximate distribution of grain sizes in mm.*				
			0.06 and coarse	0.06–0.05	0.05–0.025	0.025–0.005	Finer than 0.005
No. 1 bowl classifier:			%	%	%	%	%
Feed	41.5	870	25	35	20	10	10
Overflow	6.9	317	3	47	30	15	5
Rake product	70.0	553	80	20			
No. 2 bowl classifier:							
Overflow	1.16	144	..	..	Trace	Trace	Balance
Rake product†	70.0	173	15	50	20	10	

* Made by microscopic examination.

† Finished product.

SEDIMENTATION

BY ANTHONY ANABLE

CONTINUOUS THICKENING

Definition. The term "sedimentation" or "thickening" generally implies gravitational settling of solid particles that are suspended in a liquid. It may be divided into two general classes, sedimentation of sandy material and sedimentation of slimes. Usually the term sedimentation or thickening implies the removal of the bulk of the liquor or water from the slime by settling.

Mechanics of Clarification (Deane, Settling Problems, *Trans. Am. Electrochem. Soc.*, p. 659, 1920). In all clarification problems, dilution ratio, or the weight ratio of liquids to solids, has a most important bearing. If, for example, a thin pulp, say a greatly diluted metallurgical slime, is poured into a glass cylinder and is allowed to settle, the following is observed: First, we see that a classification takes place, in which the coarsest particles settle to the bottom at a comparatively rapid rate, while the finest particles, settling at a slower rate, remain on top, with gradations in size ranging between these limits. All the particles have free movement and, excepting those of colloidal size, settle at a constant velocity, which is expressed mathematically by the formula of Stokes.

Stokes's formula for spherical grains is

$$u = \frac{gD^2(\rho_s - \rho)}{18\eta}$$

where u = terminal velocity; D = diameter of the sphere; g = acceleration due to gravity or 981 cm. per sec.; ρ_s = density of sphere; ρ = density of the fluid; η = coefficient of viscosity of the fluid; the quantities all being expressed in c.g.s. units.

For water at 20°C.

$$\rho = 1, \text{ and } \eta = 0.010$$

Expressing u in millimeters per second and D in millimeters, the Stokes formula becomes:

$$\frac{u}{10} = \frac{981}{18}\frac{(\rho_s - 1)}{0.010}\left(\frac{D}{10}\right)^2$$

Simplifying,

$$u = 545(\rho_s - 1)D^2$$

Later we see that a gradual clarification takes place, relatively slow in the last stages if very fine particles are present, and that there is an absence of any line of demarcation between the settling solids and the supernatant liquid.

Now let us gradually increase the density of the pulp by adding amounts of solids and note what happens after each addition after the resulting pulp has been well mixed and allowed to settle. We observe, first, that a dilution is reached in which the fastest settling particles form into a zone and settle from then on collectively and at a retarded rate; second, that this zone commences to form at progressively earlier periods until eventually a point is reached where the initial subsidence of solids is in mass, no independent particle movement being discernible, and proceeds with a sharp line of separation between it and the supernatant liquid; third, from this point, where the subsidence takes place more or less at a constant rate, a point in concentration is reached where there is a marked decrease of the rate of settling. This is called the **point of compression** and marks the dividing line between the zone of **clarification** and the zone of **thickening.** Accordingly, pulps may be classified as follows:

Table 1. Character of Subsidence of Different Types of Pulps

	Type of pulp	Character of subsidence	Description	Examples
Clarification free settling zone	Dilute	Independent particle subsidence	Particles or flocs settle independently. No definite line of subsidence. Settling unhindered and dependent upon size of particle or floc	Turbid water, sewage, and trade wastes
	Intermediate	Phase subsidence	Upper zone of independent particle subsidence. Lower zone of collective subsidence. Line of demarcation not sharp	Chemical and metallurgical pulps
Point of compression	Concentrated	Collective subsidence	Definite line of subsidence. Settling rate decreases with increasing concentrations of solids. Settling rate retarded by particle or floc interference	Chemical and metallurgical pulps
Thickening compression zone	Compact	Compact subsidence	Flocs and particles in intimate contact. Subsidence due to compression	All pulps by sedimentation pass into this zone

The following is an abstract from Coe and Clevenger's paper, Methods for Determining the Capacities of Slime-thickening Tanks (*Trans. Am. Inst. Mining Met. Engrs.*, pp. 356–384, 1916):

If a thin pulp, of a dilution of, say 10 to 1, is placed in a 1000-cc. cylinder, after thorough mixture, at least momentarily, it forms a homogeneous mass as shown in Fig. 1*E*. After a short time, however, it assumes a flocculent structure which, after settling a brief period of time, forms four distinct zones, *A*, *B*, *C*, and *D* as in Fig. 1*F*.

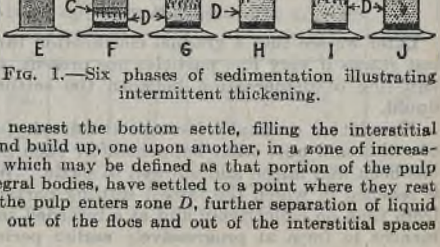


FIG. 1.—Six phases of sedimentation illustrating intermittent thickening.

The first particles that reach the bottom of the cylinder are the coarser granular sands which may be present in the pulp. Immediately following this and somewhat contemporaneously with the settling of the sand, the slime flocs nearest the bottom settle, filling the interstitial spaces between the sand particles and build up, one upon another, in a zone of increasing depth. This we term zone *D*, which may be defined as that portion of the pulp wherein the flocs, considered as integral bodies, have settled to a point where they rest directly one upon another. After the pulp enters zone *D*, further separation of liquid must come through liquid pressed out of the flocs and out of the interstitial spaces between the flocs.

Immediately above zone *D* is a transition zone *C*. The pulp in zone *C* decreases in percentage solids from the bottom, where the flocs enter zone *D*, to the top, where the consistency of the flocculated pulp is the same as that of the original pulp. In speaking of flocculated pulp, it is intended to eliminate from consideration the coarser portion of the contained sand which falls directly through the overlying zones into zone *D*.

Above *C* is zone *B*, of constant consistency of flocculated pulp and of the same consistency as the flocculated pulp in the feed pulp. Zone *A*, overlying zone *B*, is clear water or solution. In the case of a very rapidly settling slime, zone *A* in the earlier stages may be turbid, due to finely divided matter remaining in suspension. Later this very fine material settles and the liquid becomes clear, although there are cases, especially when the liquid contains very little electrolyte, where it remains turbid for a long time.

Figure 1*E* shows a cylinder freshly filled with pulp of a consistency of about 10 parts water to 1 part ore. In this illustration zone *B* occupies the total depth. *F*, *G*, and *H* of Fig. 1 show progressive stages of settling in which zones *A* and *D* are growing deeper, zone *B* is decreasing in depth, and zone *C* remains constant, a feature of this particular pulp. Figure 1*I* shows the condition when all of the pulp has entered zone *D* and compression of the slime flocs is going on. Figure 1*J* shows the final stage of settling, beyond which the pulp will not thicken further.

With intermittent operation, any one of the stages described may represent the condition in the thickener depending upon the length of time that the pulp has been allowed to settle. In the operation of continuous thickeners, the feed of the thin pulp at the center of the tank, the overflow of clear liquid at the periphery of the tank, and the discharge of the thickened pulp at the bottom are generally continuous. In a continuous thickener, the four zones previously described in discussing intermittent settling are generally present as shown in Fig. 2. At the top there is a zone of clear water, *A*. Beneath this is a zone *B*, consisting of flocculated pulp of uniform consistency. Directly beneath this is a transition zone *C*, and at the bottom a zone *D* of pulp which is undergoing compression. In making tests, the settling rates of thin pulps are determined by readings taken at the juncture of zones *A* and *B*, *i.e.* where the pulp surface joins the liquid.

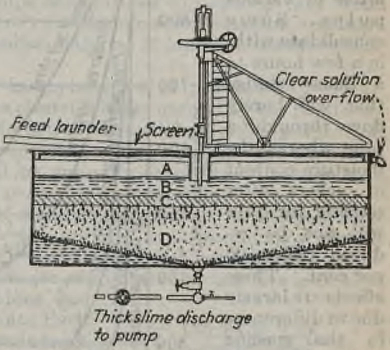


FIG. 2.—Four zones of settling pulp illustrating continuous thickening.

Clarification Capacity. The relationship, between the settling rates of particles at their various dilutions, in terms of thickener area required, may be expressed by the following formula.

$$A = \frac{1.333(F - D)}{R \times \text{sp. gr.}}$$

where A = square feet per ton of dry solids per 24 hr.; R = settling rate in feet per hour of a feed with F dilution; sp. gr. = specific gravity of liquid; F = weight ratio of liquid to solids for the rate R; D = weight ratio of liquid to solids in discharge.

By applying this formula to pulps of different densities, ranging in dilution from feed to discharge density, the zone requiring the greatest unit area is found, and it is this zone which determines the area that must be provided for the pulp being tested.

Thickening Capacity. The volume provided in a tank in the thickening zone depends directly upon the period of detention required for the sludge to reach the desired density and may be determined by the following formula.

$$V = \frac{4T(G - \text{sp. gr.})}{3G(S - \text{sp. gr.})}$$

where V = volume in cubic feet required for thickening per ton of solids per 24 hr.; S = average specific gravity of thickened pulp during compression period; sp.gr. = specific gravity of clear solution; G = specific gravity of solids in pulp; T = period of detention in hours.

Mechanics of Thickening. There is a remarkable difference in the thickening properties of various pulps. Some consolidate within a few hours to a dense sludge that will barely flow through a pipe, whereas the moisture content of others after settling for days will not be reduced below 95 per cent. These effects are largely due to differences in the specific gravity of the solids and the physical differences in the character and structure of the flocs.

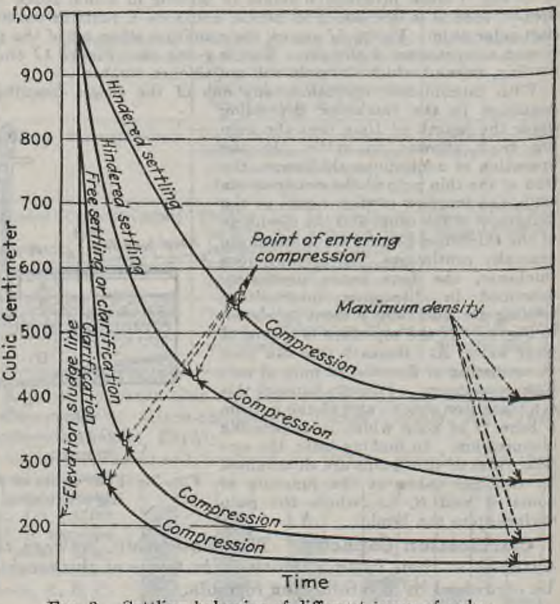


Fig. 3.—Settling behavior of different types of pulps.

Figure 3 illustrates the phases of the mechanics of clarification and thickening of different types of pulp.

Destabilization of Colloids. In all clarification-thickening problems, the aim is to obtain the lowest operating area (in terms of square feet per ton of solids treated) permissible with satisfactory operation, including a safe allowance for variations in feed conditions, such as dilution, temperature, degree of flocculation, particle size, etc. From the formula for area it is seen that the rate of settlement or clarification is one of the controlling factors. The area of a thickener varies inversely with the settling or clarification rate, other conditions remaining constant.

Any agency which will effect an increase in settling rate will bring about a corresponding reduction in tank area required. As the rate of subsidence is the settling rate of the slowest settling particles, the control of the settling rates of the finest particles is extremely important. As the particles become finer, the effect of gravity is gradually reduced and is eventually overbalanced by the forces of surface energy and Brownian movement, and a colloidal state is reached where the dispersed particles remain in permanent suspension in the liquid. From a state of coarse suspension, through finer suspensions and colloidal solutions to true solutions, there exists a perfect continuity in change of particle size and the change from one state to another is not sharply defined.

In general metallurgical practice **sands** are considered to be particles coarser than 200 mesh, 0.074 mm.; and **slimes** any material finer than 200 mesh. According to Zsigmondy's classification of solid particles, the following ranges of sizes are given:

Suspensions. Particles coarser than 0.0001 mm. in mean diameter.

Colloidal solutions. Particles under 0.0001 mm. and over 0.000001 mm. in mean diameter.

True solutions. Particles under 0.000001 mm. in mean diameter.

The metallurgical and chemical pulps encountered in practice vary from fine to coarser suspensions, with relatively small amounts of colloidal material. Probably the largest content of colloidal material is found in certain clays and in sewage and trade wastes.

The most widely accepted theory in explanation of the stability of the colloid is based on the assumption of an electric charge carried by the particles, a surface phenomenon derived by preferential adsorption of either positive or negative ions from dissociation of compounds. Particles, charged with like signs, all positive or all negative, will be mutually repelled and remain dispersed through the liquid medium in permanent suspension.

In order to destroy this condition of stability and induce clarification, it is necessary to neutralize the charge on the particle by introducing a charge of an opposite sign, by the addition of either an electrolyte or another colloid. Accordingly, negatively charged colloids are precipitated by positive ions, and positively charged colloids by negative ions, and the neutralized particles agglomerate into flocs, producing the condition known as **flocculation.** The use of lime in cyaniding is a good illustration of such an effect of flocculation.

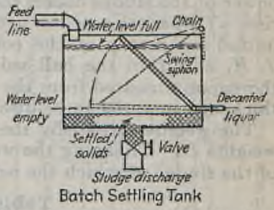


FIG. 4.—Intermittent settling tank.

Equipment Used. Intermittent Settling Tanks. This is the simplest and oldest device for thickening.

The intermittent settling tank is of any convenient shape or size, rectangular tanks, however, being more common than cylindrical. A discharge valve or gate is placed in the bottom for the removal of the thickened material. The clarified solution is withdrawn either by a swing siphon as shown in Fig. 4 or through draw-off connections, located at suitable intervals along the side. This is a home-made device on which few or no operating data are available.

Operation. The tank is filled with the pulp to be thickened, which is then allowed to stand undisturbed for the period of time which experience has shown necessary for settlement of the solids and their compacting as a heavy, dense sludge in the lower part of the tank (Fig. 4).

The supernatant layer of clarified solution is thereupon removed, by either gradually lowering the swing siphon or opening the draw-off connections one by one, starting with the uppermost one and working downward. When the decanted liquid begins to show appreciable amounts of sludge, or the sludge level is exposed, the decantation is stopped as the sludge line has been reached and further separation by settlement cannot be obtained.

The sludge-discharge valve or gate is now opened and the compacted mass flows out, aided possibly by a stream of water from a hose or by manual shoveling or sweeping. The valves are then closed, the tank refilled, and, after a suitable settling period, the cycle of operations repeated. Intermittent settling tanks are usually operated in groups of a half dozen or so, one or more being filled, or emptied while settlement is taking place in others.

Non-mechanical Continuous Thickeners. The settling cone is a non-mechanical settler. Continuous operation can be secured, since it is equipped with facilities for the continuous overflow of clarified solution and the continuous discharge of thickened sludge.

The cone, as its name implies, consists of a conical tank, the angle at the apex of which is 45 to 60 deg. The conical tank is mounted with its apex pointing directly downward, being provided with a manually or automatically controlled sludge-discharge valve at the bottom and a centrally located loading well and peripheral overflow trough at the top.

Equipment Used. a. The Allen Cone. This type is shown in Fig. 5. The feed enters through the central loading well. *A*, the clarified solution, overflows into the externally located peripheral trough *C*, and the solids settle in the base of the cone *K*. The baffle *B* in the loading well prevents undue agitation.

The solids settling in the cone *K* increase the density of its contents until the buoyant power of the sludge causes the actuator *F* to rise. This motion of the actuator is transmitted by means of the connecting parts *G*, *H*, and *I* to the ball valve *J*, which is thereupon unseated from the orifice in the apex of the cone thus allowing the thickened material to be discharged.

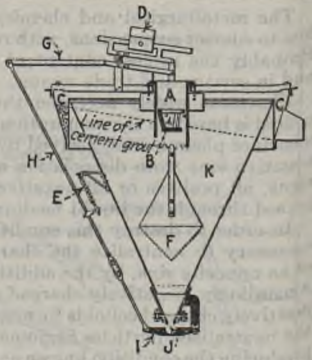


Fig. 5.—The Allen settling cone.

The weight carried by the actuator *F* is controlled by the position of the weights *D*. Changing the weight or its position changes in turn the density of the sludge at which the actuator *F* becomes buoyant.

Table 2. Allen Cone Sizes

Diameter	Depth, Inlet to Spigot
3′6″	5′2″
4′6″	6′2″
6′0″	7′8″
8′0″	9′11″

b. The Callow Cone. The apex angle of the Callow cone is 60 deg. As shown in Fig. 6, the feed enters through a semisubmerged well at the top center of the tank. The peripheral trough for the collection of clarified solution is formed by a metal strip, attached to the inside of the tank near the

Table 3. Callow-cone Sizes

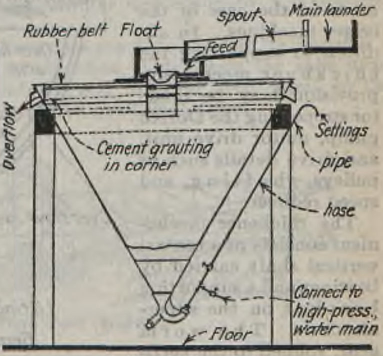
Outside diam. (diam. tank top)	Inside diam. (diam. overflow weir)	Depth (tank top to bottom spigot)	Overflow capacities g.p.m.
2′ 9″	2′ 0″	3′ 4″	
3′ 3″	2′ 6″	3′ 9″	
3′ 9″	3′ 0″	4′ 2″	
4′ 9″	3′ 0″ and 4′ 0″	5′ 0½″	6–8
5′ 9″	4′ 7″ and 5′ 0″	6′ 0″	10–12
6′ 9″	5′ 5″ and 6′ 0″	6′ 10¼″	14–18
8′ 9″	7′ 5″ and 8′ 0″	8′ 7″	25–30

top, and fitted with an adjustable strip of belting to insure uniform overflow across the entire periphery.

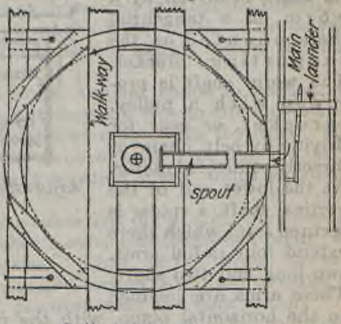
The thickened material is discharged from the bottom of the tank by a bushing, a plug valve, or by an adjustable gooseneck siphon. The purpose of the gooseneck siphon is to control the discharge density. The greater the elevation of the siphon discharge above the apex of the cone, the greater is the density of the discharge and vice versa.

Mechanical Continuous Thickeners. The Dorr thickener, of which there are several different types, consists essentially of a shallow, cylindrical settling tank, equipped with a central feed well, peripheral overflow-collection trough, pump-regulated sludge-discharge outlet at the bottom and a slowly revolving, centrally located shaft, equipped with radial arms and plow blades for moving the settled sludge gently to the central sludge outlet. Distinctive features are the use of shallow cylindrical tanks, mechanical methods for the collection of the settled solids, and volumetric regulation of discharge density by means of a diaphragm pump with variable displacement.

The Single-compartment Dorr Thickener. This type is shown in Fig. 7. The tank is cylindrical and flat bottomed, constructed of steel, concrete, or wood, and if steel, may be rubber lined, or, if wood, lead lined for acid-resisting duty. An overflow-collection trough, annular in plan view and rectangular in section, is provided around the inside



Callow Settling Cone



Callow Slime Cone, Plan

Fig. 6.—The Callow settling cone.

top of the tank and is equipped with a leveling strip to ensure uniform overflow of clarified solution across its entire length. A discharge casting, constructed of cast iron or hard lead (for acid-resisting duty), is secured in the center of the tank bottom. A discharge line extends from this cone to the suction side of a Dorrco diaphragm pump.

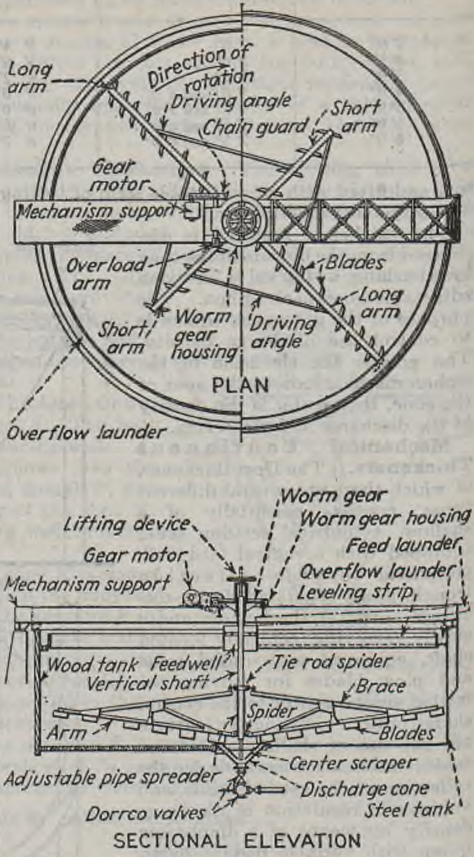


FIG. 7.—The Dorr thickener.

A structural-steel superstructure spans the top of the tank, supported by the tank sides in relatively small units and by steel or concrete columns in the case of the larger machines. In addition to supporting the thickener mechanism, provision may be made for supporting the Dorrco pump, motor drive unit, and drive details such as pulleys, shafting, and speed reducer.

The thickener mechanism consists of a central vertical shaft carried by bearings and a supporting bracket on the superstructure. The worm gear, splined to the vertical shaft, is driven by a worm on a tangential shaft mounted on the supporting bracket. The worm shaft is provided with a pulley, sprocket, or gear for driving by belt, chain, or directly connected motor. At the lower end of the vertical shaft, a spider is secured from which there extend four radial arms, two long and two short. These arms are inclined to the horizontal plane, with the result that a slightly conical bottom of settled solids is built up under the plow blades which are attached to the under sides of these arms. Where the settled solids are too valuable to be used in forming the conical bottom, a cement or dirt fill may be used. These plow blades are mounted at an angle to a radial line and are so arranged that the entire area of the bottom is swept by them each revolution and solids deposited thereon moved gently toward the central discharge outlet. A manually operated lifting device, mounted on top of the superstructure and engaging a cap, turned on the upper end of the thickener shaft, makes it possible to lift the mechanism vertically a foot or so to relieve the load when starting up after a shutdown or operating interruption. The thrust of the worm shaft is borne by a spring-loaded thrust bearing, the displacement of which actuates a visible, overload-indicating pointer which calls to the attention of the operator the presence and degree of overload. Should the overload increase to the danger point threatening a mechanical breakage, an electrical contact is made which shuts down the drive motor and rings a warning bell.

The Dorr Torq Thickener. This type of thickener, shown in Fig. 8, is also a single-compartment thickener but differs from the unit described in the foregoing by reason of its special, antistalling arm construction. The tank is cylindrical and flat-bottomed, constructed of steel or concrete and infrequently of wood. Feed enters centrally, overflow is collected peripherally, and sludge is removed from a central annular depression by a diaphragm pump.

The unique new feature from which this machine derives its name is a torque-actuated, automatic lifting construction that causes the rake arms to raise when an overload is encountered and to lower them again to the normal operating position after the overload has been passed.

FIG. 8.—The Dorr Torq thickener.

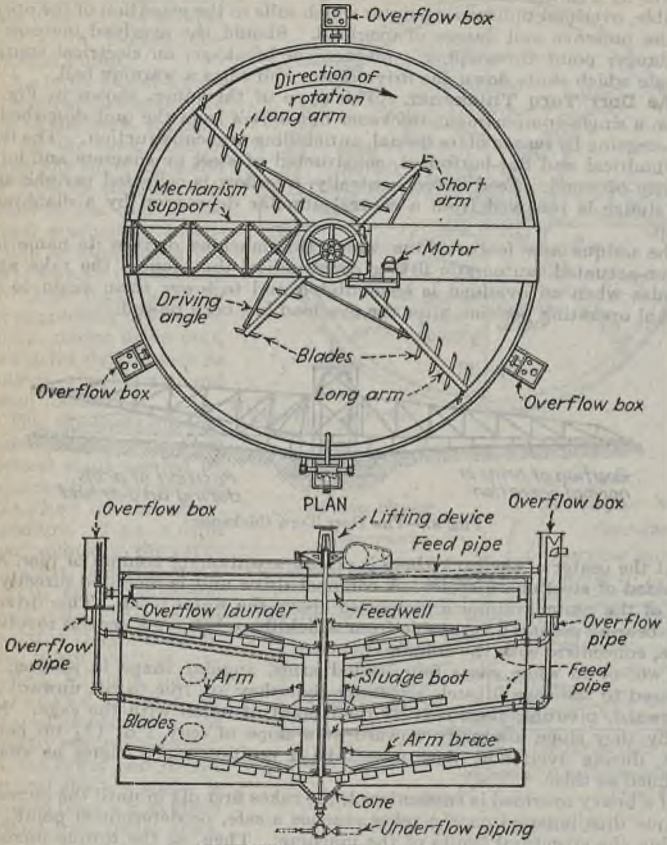
At the center of the tank there is placed a stationary column or pier, constructed of steel or concrete. A compact drive unit is mounted directly on top of the center column a foot or so above the water level. This drives a ball-bearing mounted turntable from which there is hung a central revolving cage, concentric with the column.

Two, or in some cases four, radial arms, angular shape in section, are secured to the cage in such a manner that they are free to tilt upward and rearward, pivoting diagonally at their point of union with the cage. Normally they slope gradually upward at a slope of only 1 or 1¾ in. per ft. But, during overloads, they may take a position many times as steeply inclined as this.

If a heavy overload is encountered, the rakes first dig in until the increased torque thus imposed on the rakes reaches a safe, predetermined point, well within the structural limits of the machine. Then, as the torque increases above this predetermined point, this increased torque is utilized to cause the rake arms to swing gradually upward, pivoting near the tank center. The greater the torsional load, the higher swing the arms until the rakes completely clear the obstruction or reduce it to a lower torsional equivalent.

As the overload is reduced by the continuous raking action, the torque decreases and the rake arms drop lower and lower. Finally, when the torque has decreased to a value less than that predetermined in the design of the machine, the rakes resume their normal operating position.

The Dorr Tray Thickener. As shown in Fig. 9, the tank of the Dorr tray thickener is subdivided into a multiplicity of shallow, superimposed settling compartments by a number of slightly conical steel trays which are self-supporting and attached to the sides of the tank by rim angles at their peripheries. Each compartment has an individual connection for feed pulp and for clarified overflow solution. The solids settled in each compartment are discharged by gravity into the lower portion of the compartment directly below through a centrally located, down-cast seal ring which is concentric with an upcast ring attached to the mechanism of the compartment below.



SECTIONAL ELEVATION

Fig. 9.—The Dorr tray thickener.

The bed of sludge effectively prevents the intermingling of sludge and solution.

A feed box, located slightly above the top of the tank, is provided with as many V-notch meters as there are settling compartments, thus assuring that each compartment receives its portion of the total feed. The overflow pipes from all compartments terminate in an overflow-collection box near the top of the tank. Adjustable rings on the ends of these pipes permit close regulation of the volume of solution clarified in each compartment and assist in the maintenance of the correct depth of sludge bed in each. All sludge eventually reaches the lowermost compartment from which it is continuously removed at the proper rate by a Dorrco diaphragm pump.

The tray-type thickener, similar in principle and in major structural details to the single-compartment thickener, provides the maximum capacity per unit of floor space. Since capacity is proportional to settling area and since each compartment operates substantially as an individual thickener, each compartment added increases the capacity of the original single-compartment tank approximately 100 per cent. Several other types of Dorr tray thickeners are furnished which differ from the above only in the construction of the tray and the feed, discharge, and overflow features, which latter are affected to a certain degree by the character of the pulp handled.

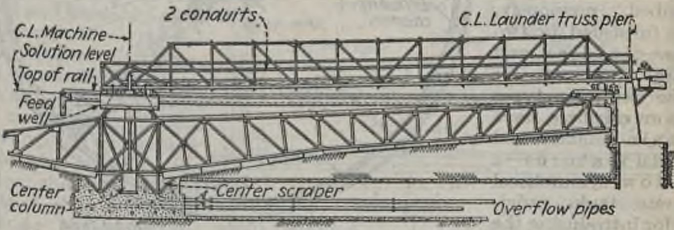


FIG. 10.—The Dorr traction thickener.

The Dorr Traction Thickener. As shown in Fig. 10, this is essentially a single-compartment thickener, the distinctive feature of which is the application of the driving power to the end of a radial truss by means of a motor-driven carriage, running on the top of the tank sides. Feed enters through a loading well at the center, overflow is collected in a circumferential trough at the periphery, and sludge is continuously removed from the center by a Dorrco diaphragm pump.

The driving truss extends from a stationary vertical column at the center of the tank, on which its central bearing is mounted, to the periphery of the tank where its driving unit is attached. It also extends in the opposite direction from the center of the tank to a distance approximately one-third of the tank radius. Plow blades, secured to the lower chord of the truss, sweep settled solids into an annular depression, concentric with the central column, from which the sludge-discharge line extends.

Connected to a source of electrical power, the central column supports slip rings which are in contact with brushes on the revolving truss, electrically connected to the driving motor. A stationary, bridge-type truss, extending from the tank periphery to the center, supports the feed trough and electrical conduits and serves as a walkway for operators.

An under-speed alarm rings a bell when an overload occurs, and, if the operator fails to correct this condition at once, an automatic device shuts off the feed of the unit by opening a by-pass in the feed trough.

The traction thickener is very rugged structurally and especially adapted to large tonnages and handling of severe raking loads. It may be furnished with corrosion-resisting construction.

The Dorr Clarifier. The Dorr clarifier is especially adapted to the handling of light, finely divided solids such as trade wastes, water purification sludges, and domestic sewage. It is a modification of single-compartment thickeners, described previously. It is furnished in two types—circular and square.

The two main types of Dorr clarifiers have certain common and essential features—a shallow, symmetrical concrete tank; provision for introducing the feed, overflowing the clarified liquor and discharging the thickened sludge; and a motor-driven revolving mechanism for sweeping the settled solids to a central discharge hopper in the bottom of the tank. Positive removal of sludge is effected by a diaphragm or plunger pump. Skimming devices may be furnished, if desired, for continuously removing scum and other light material that tends to float on the surface.

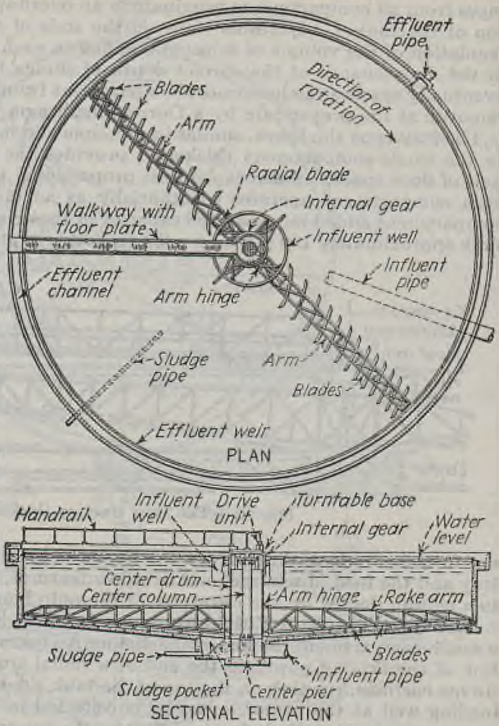


FIG. 11.—Dorr Sifeed clarifier.

The Dorr Sifeed clarifier is installed in circular tanks. Its name, "Sifeed," is a contraction for the words "siphon feed," which is one of its distinguishing features.

Feed enters centrally below the water level through suitable connections, terminating in a slotted, cylindrical diffuser. The central column, supporting the revolving mechanism, and the central drum form a conduit for the influent.

The feed leaves the central drum in a radial direction through slots. The head of water above the slots and the circular baffle have the effect of tapering the velocity of flow and giving quiescent feed conditions.

Distribution continues throughout the tank on the same radial diverging lines. The rate of diffusion is gradually decelerated as the circle of propagation increases, so that the velocity reaches the absolute minimum as the flow approaches the side of the tank.

A continuous annular trough with a continuous weir on its inboard side extends around the complete periphery of the tank. This gives maximum trough and weir length for any tank of equivalent capacity and assures minimum velocity of flow at the point of take-off. A circular scum baffle may be provided just inside the weir.

The clarifier mechanism consists of two radial trussed arms, driven by a motor on the stationary central column, and equipped with plow blades that just clear the bottom and sweep settled solids to the central discharge hopper in the bottom. The rake arms are attached to a central drum, concentric with the central column.

Sludge is removed continuously by a diaphragm pump. Where scum and floating solids tend to accumulate on the surface of the tank, positive mechanical means are provided for its removal.

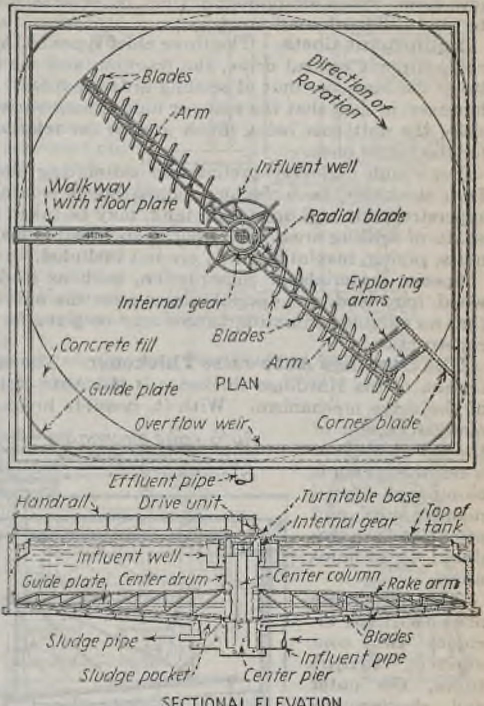


FIG. 12.—Dorr Squarex clarifier.

The Dorr Squarex clarifier is installed in square sedimentation tanks. It follows closely the arrangement of the Dorr Sifeed clarifier with the exception that the tank is square, not round, and one of the rigid arms of the clarifier mechanism is equipped with a special corner blade that reaches out into the four corners of the tank and moves the settled sludge in to the point where it may be picked up by the regular mechanism. The action of the corner blade is positive and controlled automatically. Every square foot of the tank bottom is swept at each revolution of the mechanism.

Feed enters centrally through suitable connections, is distributed radially by a submerged diffuser, and is collected peripherally across a continuous weir extending around the four sides of the tank. Two radial arms with plow blades attached are secured to a central revolving drum and are driven by a gear motor mounted on top of the center pier. These revolving rake arms sweep the area of a circle inscribed within the square bottom of the tank.

A diaphragm or plunger pump is used for sludge removal and for control purposes. Scum-skimming devices of several different types are supplied to meet different conditions.

Equipment Costs. The three chief types of Dorr thickeners—the single-compartment central drive, the traction, and the tray—are priced substantially the same per unit of settling area provided. The cost of construction, however, is such that the cost per unit of area varies materially with different sizes, the unit cost being much greater for relatively small thickeners than for the larger ones.

For rough and very preliminary estimating purposes only, the cost of a Dorr thickener, f.o.b. factory, consisting of iron and steel mechanism, steel superstructure, and open steel tank, may be taken at from $2.50 to $6.50 per sq. ft. of settling area. Accessories such as pumps, tank covers, motor-drive units, piping, insulation, etc., are not included.

Special materials of construction, such as lead or rubber-covered steel, wood, hard lead, and special alloys, alter the unit cost to such a wide extent that no reliable estimating figures may be given for thickeners of such special construction.

The Hardinge Auto-raise Thickener. The outstanding and interesting feature of the Hardinge thickener is the Auto-Raise device included as part of the drive mechanism. With it, possible breakage due to overloads and manual raising of the mechanism when they occur is eliminated, and maintenance and attention are reduced to a minimum.

The Auto-Raise mechanism includes two concentric torque tubes, the outer and shorter one being entirely above the thickener liquid level. A yoke at the top of the inner torque tube has extended rollers which normally rest at the bottom of two diagonally opposite sloping slots in the outer torque tube. When the scraper encounters an obstruction, the abnormal resistance created causes the aforementioned rollers to move along and up the sloping slots with a telescoping effect of the two torque tubes and a shortening of their total length.

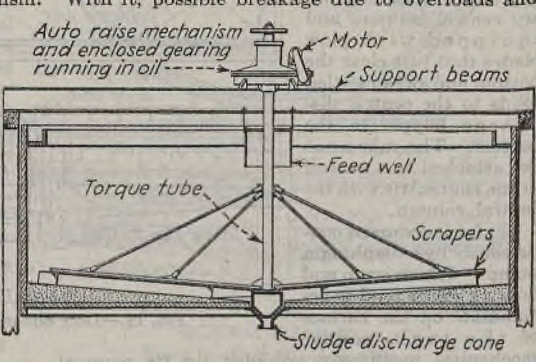


Fig. 13.—Hardinge Auto-Raise thickener.

When the overload or resistance is decreased, the scrapers automatically lower, by the effect of their own weight, to their normal operating position. If the overload increases, the scrapers raise near their maximum distance, sound an alarm, and cut off the driving motor.

The general arrangement of the Auto-Raise thickener is shown in Fig. 13. A support structure of I-beam or low-truss construction spans the top of the tank and supports the rotating mechanism.

The driving mechanism is compact and includes fully enclosed oil-lubricated gears and overload protection devices. The rotating mechanism is supported on a ring-type ball bearing which is designed to prevent any sway of the mechanism and which operates in an oil bath. The thickener is supplied in sizes from 6- to 100-ft. diameter, with full double spiral scrapers, with segmental scrapers and with scrapers of wood, stainless steel, or rubber-covered steel construction. Power requirements are low, a 1-h.p. motor being ample for a 40-ft. diameter machine.

Hardinge Diaphragm Pump. Hardinge diaphragm pumps are used for the removal and control of sludge or pulp as underflow from Hardinge thickeners. The general arrangement of the pump is shown in Fig. 14.

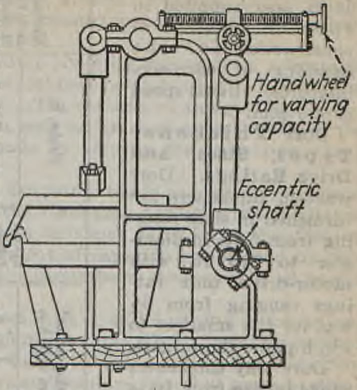


Fig. 14.—Hardinge Simplex diaphragm pump.

The outstanding feature of the Hardinge diaphragm pump is the easy control attachment by which the stroke and capacity can be varied without stopping the pump. This easy control feature is a valuable one where it is desirable to closely regulate the moisture content of sludge coming from the thickeners, particularly in countercurrent decantation washing plants where a small decrease in the moisture content of the sludge discharge from each thickener adds substantially to the plant efficiency.

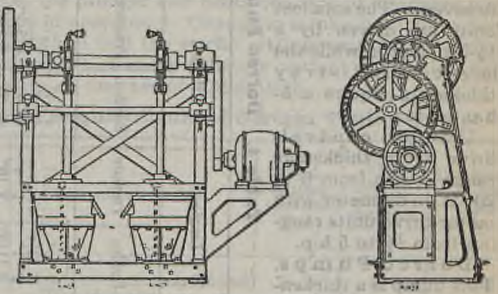


Fig. 15.—The Dorrco suction pump.

The constant-speed eccentric supplies the primary motion, and the variation in the movement of the diaphragm piston is obtained by moving the connecting rod attached to the eccentric along a lever arm.

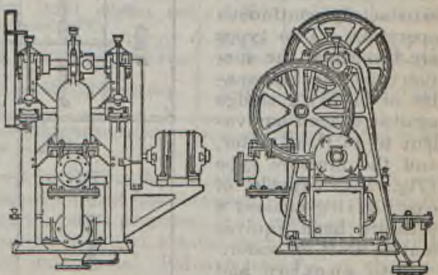


Fig. 16.—The Dorrco pressure pump.

The pump is supplied in 3- and 4-in. sizes, in simplex, duplex, and triplex arrangements, and arranged for belt or direct motor drive. It is also supplied in special materials of construction for the handling of corrosive mixtures. Usual speed is 50 r.p.m.

Dorr Thickener Types, Sizes, and Drive Ratings. Dorr traction thickeners are furnished in sizes ranging from 6 ft. in diameter to 375 ft. with motor-drive unit ratings ranging from ½ h.p. for the smallest to 7½ h.p. for the largest.

Dorr tray thickeners range in size from 10 to 75 ft. in diameter with any number of trays up to seven. The smallest units are driven by a ½-h.p. motor, while the largest multi-tray thickeners require a 5-h.p. motor.

Unit-type central-drive Dorr thickeners range in size from 6 to 200 ft. in diameter with motor-drive units ranging from ½ to 5 h.p.

Dorrco Pumps. This pump is a thickener accessory which is virtually essential for satisfactory continuous operation. Two types are furnished, the suction type (Fig. 15) capable of lifting sludge against a head equivalent to 14 ft. of water, and the pressure type (Fig. 16), capable of operating against a pressure head equivalent to 45 ft. of water.

Both suction and pressure pumps are of the diaphragm type, the

Table 4. Dorrco Pumps

Size and type‡	Width		Length		Height		Rating motor* drive unit, h.p.		Speed, r.p.m.	Stroke, in.		Displacement† cu. ft. water per min.
	Suction	Pressure	Suction	Pressure	Suction	Pressure	Suction	Pressure		Max.	Max. recommended	
1 Simplex	2′ 2½″	.....	1′ 6″	....	4′ 4″	....	1	....	50	1	¾	0.77
1 Duplex	2′ 2½″	.....	3′ 2½″	....	4′ 4″	....	2	....	50	1	¾	1.54
2 Simplex	2′ 6″	2′ 3″	2′ 1″	2′ 4″	4′ 9″	4′ 5¼″	1½	2	50	2	1½	2.03
2 Duplex	2′ 6″	2′ 3″	3′ 11½″	3′ 10″	4′ 9″	4′ 5¼″	2	5	50	2	1½	4.06
2 Triplex	2′ 6″	2′ 3″	5′ 2″	5′ 4″	4′ 10″	4′ 5¼″	3	5	50	2	1½	6.09
3 Simplex	3′ 0″	2′ 9″	2′ 2½″	2′ 8″	6′ 3″	5′ 3¼″	2	3	50	2½	2	3.60
3 Duplex	3′ 0″	2′ 9″	4′ 4″	4′ 4″	6′ 3″	5′ 3¼″	3	5	50	2½	2	7.20
3 Triplex	3′ 0″	2′ 9″	5′ 5″	6′ 0″	6′ 5″	5′ 3¾″	5	7½	50	2½	2	10.80
4 Simplex	3′ 0″	2′ 9″	2′ 5″	2′ 8″	6′ 5″	5′ 3¾″	2	3	50	3	2½	5.72
4 Duplex	3′ 0″	2′ 9″	4′ 7″	4′ 4″	6′ 5″	5′ 8¾″	3	5	50	3	2½	11.54
4 Triplex	3′ 0″	2′ 9″	5′ 10″	6′ 0″	6′ 5″	5′ 8¾″	5	7½	50	3	2½	17.16
4 Quadruplex	3′ 0″	2′ 9″	8′ 9″	8′ 0″	6′ 8″	5′ 8¾″	7½	10	50	3	2½	22.88
4 Quintuplex	3′ 0″	2′ 9″	10′ 5″	10′ 0″	6′ 8″	5′ 8¾″	7½	10	50	3	2½	28.60

* Actual power consumption about 50 per cent of motor rating.

† Displacement on water greater than on sludge on account of the frictional resistance with latter. Discount displacements 20 per cent for 50 per cent of moisture sludge and 30 per cent for 40 per cent moisture.

‡ Size refers to inside diameter of pipes leading to and from each pump bowl.

diaphragm, of rubber-cord construction, being clamped rigidly around its periphery to the pump bowl by means of a metal retaining ring. Intake and discharge valves are of the ball type, constructed of rubber with a lead slug in the center to give the correct weight.

A connecting rod, driven from the pump shaft by means of an adjustable eccentric, actuates the diaphragm, the central portion of which oscillates while the periphery remains stationary. The volume of sludge displaced at each stroke may be varied at will from zero to the maximum for which the pump is designed, adjustment being made by varying the eccentricity by a handwheel on the eccentric.

Once adjusted to average conditions, the pump tends to maintain constant sludge density, since the more dilute the discharge becomes the smaller is the amount of solids removed so that the tendency is to bring the dilution back to normal. The final, fine adjustment of capacity is obtained by permitting a small amount of air to enter the pump bowl through a small needle valve connected thereto.

Dorrco pumps are supplied in four sizes, 1, 2, 3, and 4 in. and in five types, simplex, one pump body; duplex, two pump bodies; triplex; quadruplex and quintuplex. Pumps are arranged with tight and loose pulleys for belt drive or equipped with individual drive motors and double-reduction gears, or silent chains and sprockets for giving the required reduction in speed.

The new Dorrco V-type pump differs from the standard type described above in that the stroke of the plunger and hence the rate of discharge may be varied while the pump is in operation. Change in stroke from $\frac{5}{8}$ to 3 in. is effected by a small handwheel and the stroke setting recorded on a dial. This feature permits close regulation of the moisture content of the discharge which obviously is a function of the rate of withdrawal.

Table 5. Settling-area Requirements of Certain Typical Pulps*

Type of pulp	Chemical composition	Usual dilution thickener feed	Usual moisture content discharge, %	Unit area, sq. ft. per ton per 24 hr.
Cyanide-process slime (gold-bearing ores).	1% sodium cyanide solution and −200-mesh quartz	2– 5:1	45–60	5– 13
Lead-flotation concentrates......	Alkaline water and −65-mesh PbS	3– 4:1	20–40	7– 18
Lime-soda-process lime mud.....	15–20% NaOH solution and precipitated $CaCO_3$	8–10:1	50–70	16– 30
Water-floated whiting...........	Water and −300-mesh $CaCO_3$	20–25:1	50–70	45– 75
Bauxite residue, after H_2SO_4 digestion.	30° Bé. $Al_2(SO_4)_3$ and fine silica	10–20:1	20–40	75–150
Water-floated clay..............	Water and 300–325-mesh clay	30–60:1	12–60	50–225

* The figures above are general averages for illustrative purposes only, as each material must be checked by tests before selection of size of machine required.

The unit settling area required for thickening pulp varies greatly, not only between pulps of different composition but also between pulps of seemingly identical composition. In general, pulps prepared from metal-bearing ores by wet grinding to a moderate mesh, 65 to 100, exhibit the most rapid settling rates and require the smallest unit areas. Pulps consisting of chemical precipitates suspended in a solution exhibit medium settling rates and medium unit areas. Pulps prepared from non-metallic minerals, ground to a fine mesh, 250 to 325, exhibit generally the slowest settling rates and require the largest unit areas.

The variation in settling rate and unit-area requirement is even more striking in the case of seemingly identical pulps. In cyanide pulps, unit areas vary from as low as 2 sq. ft. per ton per 24 hr. for extremely granular solids to as high as 15 for clay-like, slimy solids.

Lime mud pulps ($CaCO_3$), precipitated in the lime-soda process of caustic soda manufacture, vary widely with respect to the unit areas required for thickening. Variations from 2 to 40 sq. ft. per ton per 24 hr. are common. The settling characteristics are determined not so much by the precipitate itself as by the physical conditions during causticizing, including time, temperature, speed of agitator, and strength of solution. In clay pulps, the unit areas vary from 5 to 225 sq. ft. dependent upon the type of clay, its physical character, the dilution of the pulp, and, finally and most importantly, the natural flocculating or deflocculating characteristics exhibited.

Selection of Type of Continuous Mechanical Thickener

The selection of the type of thickener for handling a given pulp is generally based on the following considerations:

1. Floor Space Occupied. The tray type gives the greatest capacity, settling area, and volume, per unit of floor space occupied.

2. Conservation of Heat. The tray type, easily covered and insulated, gives the least temperature drop between feed and overflow.

3. Large Raking Capacity and Structural Ruggedness. The traction type has the greatest raking capacity per unit of area and is especially adapted to handling difficult raking problems, since power is applied at the end of a long arm and deeper and larger plow blades are used than on other types.

4. Handling Corrosive Materials. The single-compartment (central-drive) type is best suited to handling acid and corrosive solutions, since the portion of the mechanism below the solution level may be constructed of any one of several materials, the efficiency of which has long been established for acid-resisting duty: *e.g.*, wood, lead-covered steel, hard lead, rubber-covered steel, or such alloys as Duriron, Pioneer metal, bronze, stainless steel, etc. The traction type may be furnished with certain corrosion-resisting materials.

5. Periodic Overloads. The torque type, with automatic self-raising and lowering arms, adjusts itself to the severity of the raking load.

Thickening Costs

1. Erection. The following average figures are suitable for preliminary estimates:

Erection of thickener mechanisms	$70 per ton
Erection of thickener superstructure	$30 per ton
Erection of thickener tanks (steel)	$30 per ton
Erection of thickener tanks (wood)	$80 per ton

Foundations and concrete tanks:

Excavation	$1 per cu. yd.
Concrete in place	$30 per cu. yd.
Beams and joists (wood) in place	$50 per 1000 ft. b.m.
Columns and beams (steel) in place	$120 per ton

2. Labor. This is a very small item as attention is generally confined to pump adjustment, starting and stopping (not over once or twice per shift), and lubrication, requiring about 10 min. per day per thickener.

At the Phelps-Dodge Corporation, Morenci branch, one laborer devotes 3 hr. per 24 hr. to the operation of one 200-ft. thickener, handling 2000 tons of tailings per day. At the Inspiration Copper Company, one man per shift at a wage of $4.40 operates eight 60-ft., three 80-ft., three 100-ft., and one 200-ft. thickeners.

At the average chemical plant operating four to six thickeners, one man and a helper operate the thickeners, as well as other equipment such as agitators, filters, etc., and, in addition, carry out routine analyses for chemical control.

3. Power. See figures given on p. 1634 under Dorr Thickener Types, Sizes, and Drive Ratings. These figures refer to horse-power ratings of motor-drive units and should be discounted 50 per cent to give approximate power consumption during continuous operation.

4. Repairs and Supplies. Owing to slow speed of rotation and location of all wearing parts above solution level, repairs and supplies are virtually negligible. Whatever breakages do occur are generally due to faulty or negligent operation, resulting in severe overloads.

5. Maintenance on 12 Thickeners at the Inspiration Copper Company.

	Labor	Material	Total
1924	$25.49	$15.16	$ 40.65
1925	$81.55	$77.96	$159.51
1926 (6 mo.)	......	$ 8.40	$ 8.40
Total (2½ years)	......	......	$208.56

Table 6. Typical Thickening Costs from Practice

Plants	Daily tonnage	Thickeners installed	Cost per thickener per ton
Tonapah Extension Mining Co.	350	Four 30 × 10 ft.	$0.02
South American Development Co.	250	Six 39 × 12 ft.	$0.0115
Tom Reed Gold Mines	250	Ten 30 × 10 ft. Five 40 × 12 ft.	$0.0086
McIntyre Porcupine Gold Mines, Ltd.	1600	Four 50 × 10 ft. Ten 30 × 10 ft.	$0.0050
Inspiration Copper Co. (copper tailings)	9000	Seven 60 ft. One 80 ft. Three 100 ft. One 200 ft.	$0.0014
Inspiration Copper Co. (flotation concentrates)	400	Two 80 ft. One 60 ft.	$0.0374

Filter Thickeners

The filter thickener, as the name implies, operates on a combination of the thickener and the filter principles. It consists essentially of a cylindrical or rectangular tank, equipped with a connection for the introduction of feed; a connection, generally in the tank bottom, for the discharge of sludge; and one or more submerged filter elements.

The solution is drawn through a filtration medium by vacuum or by gravity. The filtration medium, in some cases, consists of a multiplicity of fabric-filter elements, immersed in the pulp contained in the tank, and in other cases of a layer of sand or other granular substance laid upon a false bottom.

The submerged cake, forming upon the filtration medium, is periodically removed either by the application of low-pressure air or water on the reverse side of the medium or else by a mechanical scraping device, traveling over the medium.

The solid material in the pulp is in all cases discharged as a thick sludge, rather than as a filter cake. This sludge collects in the bottom of the tank and is discharged either by a spigot or by some type of pump.

The Genter Filter Thickener. As shown in the plan and elevation views (Figs. 17 and 18), the filter area of the Genter thickener is subdivided into a number of cylindrical filter frames, these frames being arranged in a circular tank and around a centrally located, automatic valve and filtrate-collecting receiver. During the operation of the unit, the filter elements are kept constantly submerged in the mixture being thickened.

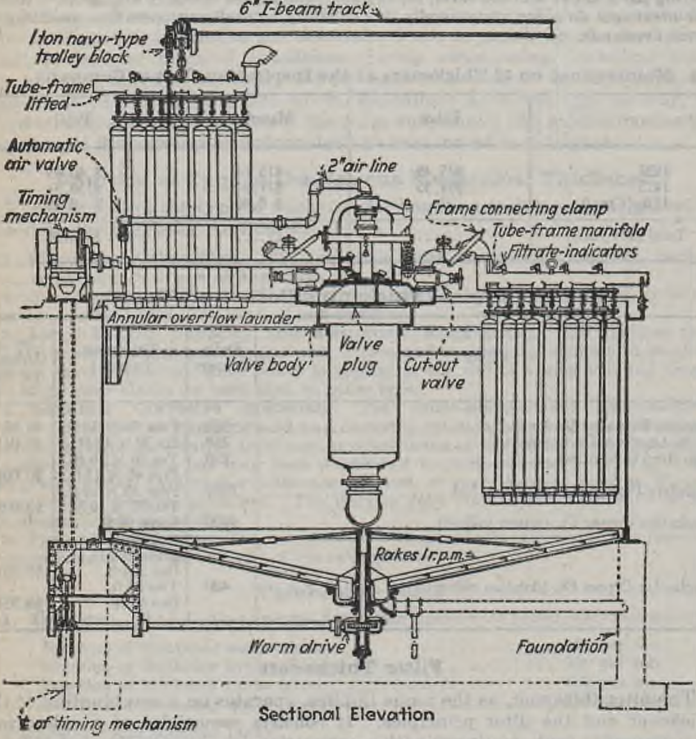


Fig. 17.—Genter filter thickener.

Each tubular element is made of corrugated iron or wood and is approximately 6 in. in diameter and 6 ft. long. Each element is covered by a suitable filter medium in the form of a tube containing about 9 sq. ft. of filter area. The top of the tubular element is closed, having the filtrate outlet at the bottom.

From 4 to 16 filter tubes are grouped in pairs in one frame, having an upper outlet manifold connecting to a port in the central valve housing, which in turn is connected with the filtrate-collecting receiver situated beneath this valve. Faulty elements may easily be replaced by releasing the spring holding that particular element in place.

A slowly revolving set of rakes operating near the bottom of the tank moves the thickened product to the central point of discharge. A special-type valve is placed in the discharge line, providing for control of sludge density. The one mechanism for actuating both valve plug and rakes is placed on the operating floor at the edge of the thickener tank. The central valve plug, made with a 45-deg. taper to prevent binding and to provide for uniform wear, is supported in the housing in an inverted position and is balanced against any desired suction head. The top of the plug is geared to a horizontal actuating shaft that connects to the timing mechanism located at the side of the tank. This timing mechanism is motor operated and is so arranged that the parallel chains not only drive the rake mechanism at a constant speed but also through a notched gear produce an intermittent movement of the horizontal shaft and the valve plug. The rotation of the valve plug within its housing, in this manner, produces repeated cycles of filtration followed by short countercurrent liquid-flow periods for cake removal, the latter action occurring within one frame of filter elements at a time.

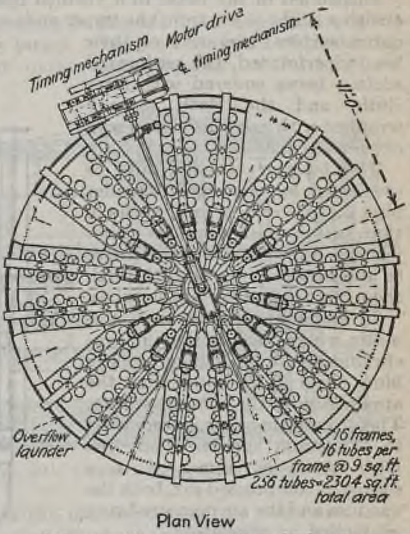


Fig. 18.—The Genter filter thickener.

The frames are lifted by means of a differential hoist suspended from a light overhead trolley. The tubes only are removed and are easy to handle as they weigh but 15 lb. each.

Operation. The central filtrate receiver is connected to a wet vacuum pump, or any equivalent suction means, which can be placed in any convenient location. This pump removes the air from the central receiver, thus inducing a suction on the interior of all elements. The flow of the filtrate is downward in the elements to the lower ends of the pipes of the supporting frames, then upward in the large pipes, and then through the upper horizontal manifold of each frame, through the valve-body parts and the valve plug to the central collecting receiver.

Filtration in individual frames and their group of filter elements or tubes is successively interrupted by rotating the valve plug in a step-by-step movement within the stationary central valve body, and a short, sudden countercurrent shock of filtrate automatically takes place, thus removing the cake of wet solids adhering to the exterior of the tubes in question.

While the filtration period on each frame is normally about 5 min. in duration, this can be varied from less than 1 to 10 min., or even more, according to the nature of the materials being thickened. The countercurrent shock of cleaning filtrate lasts but 2 or 3 sec.

The Oliver-Borden Filter Thickener. This thickener (Fig. 19) consists of a two-compartment steel tank 1; of a rectangular horizontal cross section, each compartment having a V-shaped bottom 2; fitted with a multi-bladed impeller mixer.

Suspended in the tank, in a vertical position, are a number of steel tubes 4 with a slight taper from the upper ends to the lower. The tubes have their entire surface, exclusive of their heads, perforated, the perforated surface being covered with filter cloth, and the cloth spirally wrapped with galvanized iron wire of uniform spacing.

The interior of each tube is connected by means of a header pipe 5, to a valve mechanism 6 that automatically applies either vacuum or positive air pressure to the header pipes and, through them, to the interior of the tubes.

In conjunction with the automatic valve, and acting in synchronism with it, is an automatic blow timer 7 for controlling the application of the compressed air. The automatic valve is connected to a vacuum system, while the blow timer is connected to a source of compressed air, both the vacuum and the air pressure being controlled by regulators.

Solution samplers are provided so that clarity of filtrate from each group of tubes can be readily determined.

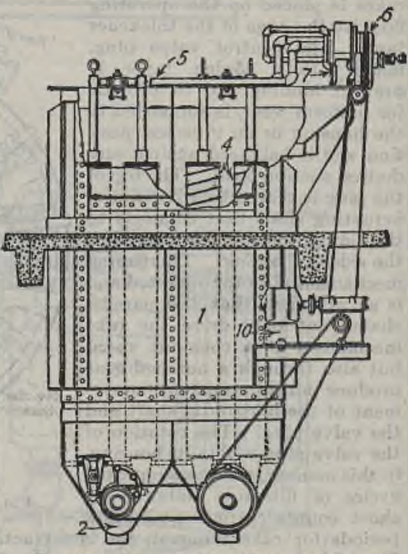


Fig. 19.—Oliver-Borden filter thickener.

The V-shaped bottoms of the thickener tank, carrying the impeller mixers, are each fitted with a gooseneck pipe outlet leading from the center of the bottoms and controlled by means of a throttle valve.

The automatic valve 6, the blow timer 7, and the impeller mixer are all driven through a roller-chain drive, by a motor 10, connected to a speed reducer.

To facilitate the placing of the tubes of the tank and their removal therefrom when necessary, an overhead track and trolley with light chain block is used.

The Sweetland Filter Thickener. This thickener is a variant of the Oliver-Borden thickener and also consists of a group of tubular cloth-covered filter elements, arranged vertically in a vat or tank which contains the liquid suspension to be thickened.

Provision is made for regular alternation of suction and back pressure on the tube by which means filtration is induced and cake deposited on the tube is discharged. Removal of the thick cake sludge is effected from the bottom of the tank by suitable means.

The tubes are fluted wood cylinders while the cloth tube covering is fastened by cord or wire winding only at top and bottom of tube.

Header connections of suitable design provide for application of suction and removal of filtrate from the tubes, as well as reversal of pressure which serves to discharge the cake of solids accumulated on the surface of tubes. The removal of thickened solid from the thickener tank resembles the method used in the Oliver-Borden thickener.

Operation of the Sweetland thickener is effected by a reversible electric motor connected to a geared rotary pump. During the period of thickening with formation of cake, the rotary pump causes suction on the tube and removes the filtrate produced, while at regular intervals the motor reverses quickly and causes back pressure on the tube to effect discharge of cake from the tube surface. A second reversal of the motor renews the cycle of thickening and flow of filtrate.

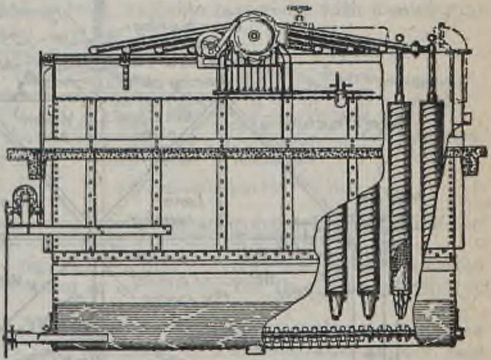


FIG. 20.—Oliver-Borden filter thickener.

Hardinge Sand-filter Clarifier. The Hardinge sand filter is particularly applicable to final clarification operation where a crystal-clear liquid product is desired.

In many cases it is employed for the clarification of the overflows from settling tanks on problems where perfect clarification cannot be economically accomplished by sedimentation. It offers advantages over the ordinary sand filter in that a wash-back arrangement with the consequent wasting of the washing liquids is eliminated.

Description. The general arrangement of a Hardinge sand filter is shown in Fig. 21. The sand-filter bed in the bottom of a round wood, steel, or concrete tank is supported on a wooden false bottom or a gravel-drainage bottom. A steel truss across the top of the tank carries the sand-cleaning mechanism which consists of a central, vertical shaft suspended and operated from the truss and having curved scrapers attached to its lower end. The shaft is threaded at the top, and its weight and that of the scrapers are carried by the shoulder of the threaded ratchet. A worm driving gear is keyed to the shaft below the ratchet. As the scraper revolves, it moves the material which has been caught on the surface of the sand bed to the central sludge-discharge well. The scraper can be lowered an infinitesimal amount, so that a thin layer of the sand bed is scraped to the center with the mud. The sludge is usually drawn off through a spigot on the end of the sludge-discharge line.

The capacities of Hardinge sand filters vary on different materials; $\frac{1}{30}$ gal. per sq. ft. filter surface per minute is obtained on 50°Be. caustic liquor; $\frac{1}{3}$ gal. on brine; and 1.0 gal. on gasoline. Suspended matter content of the liquid before filtering varies from 0.1 to 0.01 per cent.

Operating Principles. Generally speaking, the four factors controlling the filter rate are, in order of their importance, as follows: quality of suspended materials; quantity of suspended material; viscosity of liquid; and hydrostatic head or vacuum used.

The governing feature of any operation is the quality of the suspended matter. If the suspended solids are semicolloidal, the surface of the sand may become coated so rapidly by a fine impermeable layer that the rate will decrease very quickly, and the scrapers may have to be operated continuously to keep the surface of the sand bed clear. If, on the other hand, the suspended solids are crystalline, a fairly high continuous filtration rate can be maintained for many hours, and the scraper can be operated and the filter bed surface cleaned every 8 or 24 hr.

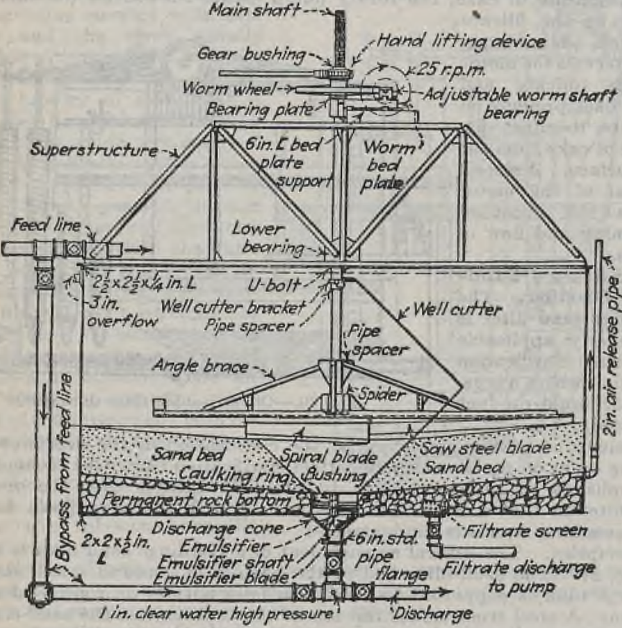


FIG. 21.—The Hardinge sand-filter clarifier.

The physical nature of the liquid is of minor importance, although the rate at which it will pass through the filter medium naturally varies inversely with its viscosity. The velocity through the filter medium is controlled and retarded sufficiently to prevent dragging the suspended particles into or through the voids in the filter medium. In other words, the filter rate is so controlled that the suspended particles will be caught at the opening of the voids. In some operations, vacuum has been supplied underneath the filter bed by the use of a centrifugal pump which serves to remove the filtrate as well as to create the vacuum.

CONTINUOUS COUNTERCURRENT DECANTATION

Definition. Continuous countercurrent decantation (abbr., C.C.D.) is the term applied to a continuous system of washing finely divided solids, such as ground ore, chemical precipitates, residues from leaching operations, etc., to free them from liquids containing dissolved substances. In practice, it consists of the operation of a series of continuous thickeners so that the solids to be washed pass through them in series, being diluted after each settling by a weaker liquid overflowing from subsequent thickeners in the system and flowing in the opposite direction.

Purpose. It is the purpose of C.C.D. to attain a high washing efficiency (separation of soluble materials from insoluble materials) with a minimum number of decantations and with the use of minimum amount of wash liquid. In practice, the desirable portion of the pulp fed to the C.C.D. system may be the solids, the solution, or infrequently both.

The washing efficiency is expressed as the percentage of the soluble salts in the feed pulp removed as an overflow product from the first thickener in the series.

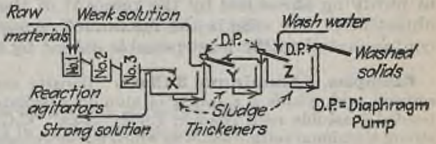


FIG. 22.—Simple C.C.D. flow sheet.

Theory. Figure 22 here represents a diagrammatic flow sheet of a simple C.C.D. plant and may be helpful in considering the theory upon which it depends.

The chemical reaction taking place in the three continuous agitators results in the formation of a pulp consisting of a concentrated solution and insoluble solids. It is desired to recover both solution and solids in as pure form as possible and with the least possible reduction in the concentration of the solution.

This pulp is sent to the first of the three continuous thickeners, *X*, *Y*, and *Z*, arranged in series as shown. The clear, concentrated solution overflows to further treatment, while solids settle on the bottom in the form of a thick sludge and are removed by a diaphragm pump.

The sludge pumped from thickener *X* is repulped with weak wash solution overflowing thickener *Z*, and the pulp so formed is rethickened in thickener *Y*. The overflow from thickener *Y*, weaker than the original solution in thickener *X* but more concentrated than that in thickener *Z*, is used in place of fresh water in the agitation step, thus conserving the soluble salts contained in it. The sludge pumped from the thickener *Y* is repulped with fresh water, and the pulp so formed resettled in thickener *Z*. The weak solution overflowing from thickener *Z* flows to the trough feeding thickener *Y*, being there used for repulping the sludge from thickener *X*. The sludge discharged from thickener *Z* is finished washed solids. The overflow from thickener *X* is the finished concentrated solution.

In C. C. D. operation the solids move in a direction countercurrent to the wash solution and are progressively impoverished in soluble-salt content by coming in contact with progressively weaker wash solutions and finally with fresh water. Similarly, the wash water flowing in a direction opposite to that of the solids, becomes progressively enriched in soluble salts by coming in contact with solids containing greater and greater amounts of soluble salts.

The efficiency of a C.C.D. system is dependent, first, upon removing the settled solids from each thickener with the minimum amount of solution, *i.e.*, at greatest or final density; and second, upon obtaining thorough repulping or remixing of the wash solution and sludge between thickeners and prior to resettling. When these two conditions are satisfied in plant operation, the washing efficiency will be that predicated from C.C.D. calculations, discussed later.

Use of C.C.D. in Chemical Processing. This method is applicable to all problems in chemical-engineering practice wherein a pulp, consisting of finely-divided, "settleable" solids and a solution, is, first, to be separated into its two constituents; and, second, the solids are to be washed for the purpose of either recovering the solution entrained by them or for the purpose of purifying the solids by the removal of the contaminating solution. The objective in any case is the maximum recovery of solution and solids, each contaminated to the least possible extent by the other.

Examples. Aluminum Sulfate. Bauxite ore, containing Al_2O_3, digested with sulfuric acid, yields a solution of aluminum sulfate [$Al_2(SO_4)_3$], in which there is suspended insoluble rock residue, chiefly silica. C.C.D. treatment of this pulp yields a strong aluminum sulfate solution which is concentrated and crystallized to form commercial "alum," and a washed silica residue which may be discarded virtually free from the valuable aluminum salt.

Caustic Soda. Soda ash (Na_2CO_3) solution, causticized with lime, yields a solution of caustic soda in which there is suspended precipitated calcium carbonate ($CaCO_3$). C.C.D. treatment of this pulp yields a strong caustic soda solution and a washed calcium carbonate.

Phosphoric Acid. Phosphate rock, containing P_2O_5, digested with sulfuric acid, yields a solution of phosphoric acid in which there is suspended finely divided calcium sulfate ($CaSO_4.2H_2O$) precipitate. Both the acid and the synthetic gypsum being of value, continuous countercurrent decantation gives two products: first, a strong phosphoric acid and, second, a washed gypsum sludge suitable for the manufacture of building materials.

C.C.D. Calculations. The washing efficiency, purity of washed solids, and concentrations of solutions may be determined by simple calculations, if reliable data are available on the character of the pulp to be handled, the amount of wash water available, and the number of thickeners to be used. The diagrammatic flow sheet (Fig. 23) and the accompanying calculations will serve to illustrate the method of procedure.

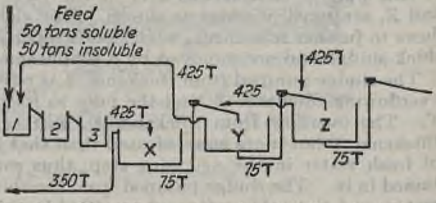


FIG. 23.—Flow sheet set up for C.C.D. calculations.

Given. (1) 100 tons per day of 50 per cent soluble material to be leached with 425 tons of water; (2) insoluble residue settles to 60 per cent moisture.

To Find. (1) Solution concentration in all thickeners; (2) washing efficiency with three thickeners (C.C.D. system); (3) percentage soluble in washed residue.

Calculation for Flow-sheet Tonnages.

Solids with sludge from each thickener = 50 tons (given)
Water with sludge from each thickener = $50 \times {}^{60}\!/_{40}$ = 75 tons

Overflow 3d thickener = 425 + 75 − 75 = 425 tons H_2O

Overflow 2d thickener = 425 + 75 − 75 = 425 tons H_2O

Feed 1st thickener = 425 tons

Overflow 1st thickener = 425 − 75 = 350 tons H_2O

Calculations for Solution Concentrations.

a. Let X, Y, and Z represent pounds of soluble salts per ton of water in the respective thickeners.

b. Then equating pounds of dissolved salts into and out of each thickener, set up the following simultaneous equations:

$$\begin{array}{ll} \text{Thickener 1} & 350X + 75X = 425Y + 100{,}000 \text{ lb.} \\ \text{Thickener 2} & 425Y + 75Y = 425Z + 75X \\ \text{Thickener 3} & 425Z + 75Z = 425 \times 0 + 75Y \end{array}$$

c. Solving by substitution:

$$\begin{aligned} X &= 282.80 \text{ lb. soluble salts per ton} \\ Y &= 47.83 \text{ lb. soluble salts per ton} \\ Z &= 7.17 \text{ lb. soluble salts per ton} \end{aligned}$$

d. Solution strength:

$$\begin{aligned} \text{Thickener 1} &= 282.80 \text{ lb. per 2000 lb. water} = 12.4 \text{ per cent} \\ \text{Thickener 2} &= 74.83 \text{ lb. per 2000 lb. water} = 3.6 \text{ per cent} \\ \text{Thickener 3} &= 7.17 \text{ lb. per 2000 lb. water} = 0.36 \text{ per cent} \end{aligned}$$

e. Washing efficiency:

$$\begin{aligned} \text{Washing efficiency} &= \frac{\text{salt recovered}}{\text{salt in feed}} \times 100 \\ &= \frac{350 \times 282.80}{100{,}000} \times 100 = 98.98 \text{ per cent} \end{aligned}$$

f. Percentage of soluble material in washed sludge:

$$\begin{aligned} \text{Water with sludge} &= 75 \text{ tons} \\ \text{Soluble salts with sludge} &= 75 \times 7.17 = 537.75 \text{ lb.} \\ \text{Solids with sludge} &= 50 \times 2000 = 100{,}000 \text{ lb.} \\ \text{Soluble salt content} &= \frac{537.75}{100{,}000 + 537.75} \times 100 = 0.535 \text{ per cent} \end{aligned}$$

Various Types of C.C.D. Flow Sheets. While the flow sheet described above is the one most generally used in C.C.D. operations, various types of related flow sheets have been developed for special cases.

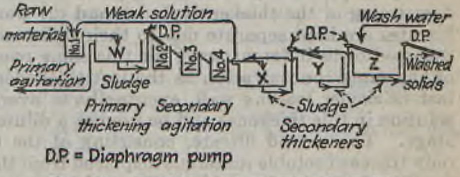


FIG. 24.—Double-stage reaction flow sheet.

1. The Double-reaction Flow Sheet (Fig. 24). With this arrangement, reactions such as leaching, digestion, or causticizing take place in two stages with an intermediate stage of thickening to effect a change of solution. It facilitates the rapid removal of finished solution and subjects the incompletely treated solids to additional agitation under carefully controlled conditions.

It is very widely used in the cyanidation of gold-bearing ores, the first stage of leaching taking place in the wet-grinding mills which grind the ore in cyanide solution. When used in double-acid digestion, the acid added in the first stage is less than enough to complete the extraction and in the second stage more than enough to complete the extraction of the remaining soluble materials, thus assuring (1) a basic final solution and (2) a high recovery of insoluble material. In the double causticization of sodium carbonate solution, an excess of lime is used in the first stage and an excess of sodium carbonate in the second. This results in (1) a rapid and relatively complete reaction in the first stage due to the presence of excess lime and (2) a relatively complete utilization of this excess lime through the subsequent reagitation of the sludge with more than sufficient sodium carbonate to satisfy the reaction requirements.

2. The Intermediate-agitation Flow Sheet (Fig. 25). Certain residues from reactions exhibit adsorptive powers to such an extent that the mixing of sludges and wash solutions, as generally carried out in repulping troughs, does not result in the usual displacement of soluble materials. In such cases, small agitators are placed between all thickeners so that longer and more violent repulping may release the soluble materials adsorbed by the solid constituent of the sludge being washed.

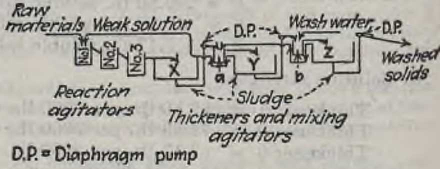


Fig. 25.—Intermediate-agitation flow sheet.

3. Continuous-countercurrent-decantation Flow Sheet with Filter at End (Fig. 26). When the solids are of value and are to be delivered in as moisture-free condition as possible, a continuous vacuum filter may be placed at the end of the series of thickeners. If desired, the filter may be arranged for cake washing in which case all or a portion of the water used in the C.C.D. series may be applied through the filter.

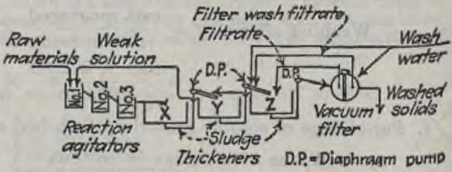


Fig. 26.—Flow sheet with filter at end.

In such a case the filtration cycle is divided into two parts: (1) straight dewatering of the thickened sludge and (2) washing of the filter cake. The filtrates are kept separate due to their difference in soluble-salt content and disposed of in different ways for this same reason. The first filtrate, being of the same concentration as the last thickener overflow, is returned to the last thickener loading well, eventually to overflow with the balance of the solution in this thickener and be used as a dilute wash solution at a preceding stage. The second filtrate, consisting of the filter wash water, containing only traces of soluble materials displaced from the already well-washed sludge, is utilized as a weak wash solution and is applied to the sludge from the next to the last thickener before the last stage of thickening.

Economic Advantages of C.C.D. The use of a series of continuous agitators and thickeners instead of a number of intermittent agitation and settling tanks has made possible certain distinct operating economies not possible otherwise. The advantages may be subdivided into two classifications, tangible and intangible.

1. Tangible Advantages:

a. Less operating labor.
b. Less steam for heating.
c. Higher washing efficiency, *i.e.*, extraction and recovery.
d. Higher temperature of finished liquor.
e. Higher concentration finished liquor.

Example. The following actual example is taken from an "alum" plant where C.C.D. replaced batch digestion of rock and washing of residue with savings of the following order:

Table 7

	C.C.D plant	Batch plant	Annual saving
Labor	1 man per shift	3 men per shift	$ 7,776
Steam in reaction	....	....	5,760
Over-all recovery, available Al_2O_3	97.5%	90%	13,406
Temperature of finished liquor and saving of evaporator steam	90°C.	30°C.	6,660
Concentration of finished liquor and saving evaporator steam	35°Bé.	30°Bé.	9,000
Total	....	....	$42,602

Data used in above comparison were as follows:

(1) Capacity, 100 tons "alum" (17 per cent Al_2O_3) per day.
(2) Labor, 48 cts. per hr.
(3) Bauxite, 50 per cent available Al_2O_3 at $14.25 per ton.
(4) Steam at 37 cts. per 1000 lb.

Intangible Advantages:

a. Less tanks and less floor area occupied.
b. Reduction of human factor in efficient operation.
c. Simple chemical control through routine analyses of first thickener overflow and last thickener discharge.
d. Reduction of "unaccounted losses" since finished product can leave system only at one point.

Typical Operating Figures from C.C.D. Practice. The data presented below are averaged from good plant operation and accordingly are representative of results secured in practice.

Table 8

	Aluminum sulfate	Caustic soda	Phosphoric acid
New materials			
a. Solid	Bauxite ore	$Ca(OH)_2$	Phosphate rock
b. Liquid	50°Bé. H_2SO_4	Na_2CO_3	50°Bé. H_2SO_4
Extraction in agitators	98.5%	91.5% causticity	97%
Strength of finished solution	35°Bé., hot	15.8°Bé., hot	30°Bé. (22% P_2O_5)
Temperature of finished solution	90°C.	74°C.	
Number of agitators	4	3	3
Number of thickeners	3	3	6
Washing efficiency	99%	99.3%	99%
Over-all recovery	97.5% of available Al_2O_3	....	96% of available P_2O_5

Complete C.C.D. System in a Single Unit. The Dorr washing type of tray thickener (Fig. 27) differs from tray thickeners previously described in that in this case the several, superimposed settling compartments operate in series, just as do the thickeners in a C.C.D. system, and not in parallel, as do the compartments of the other types of tray thickeners. The feed pulp is introduced into the first or uppermost compartment, and strong solution overflows from this compartment, while wash water is introduced into the last (lowest) compartment.

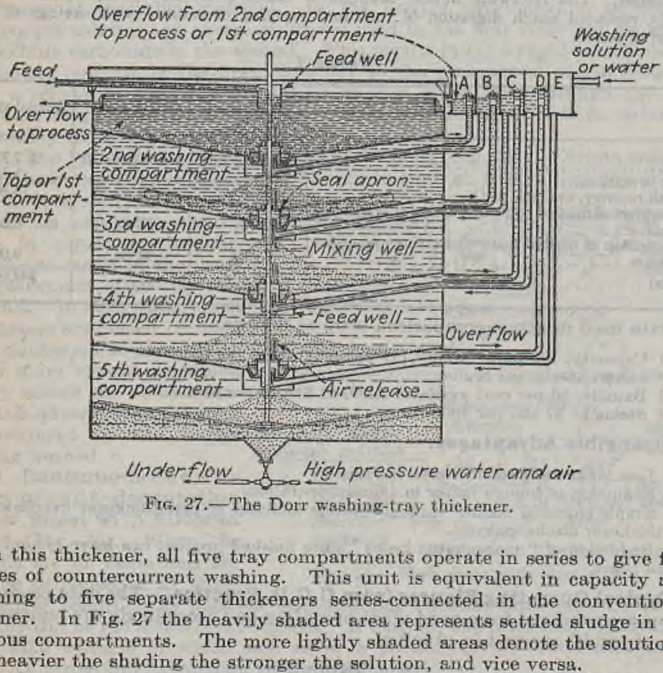


FIG. 27.—The Dorr washing-tray thickener.

In this thickener, all five tray compartments operate in series to give five stages of countercurrent washing. This unit is equivalent in capacity and washing to five separate thickeners series-connected in the conventional manner. In Fig. 27 the heavily shaded area represents settled sludge in the various compartments. The more lightly shaded areas denote the solutions, the heavier the shading the stronger the solution, and vice versa.

Feed enters the top compartment via a central, semisubmerged feed well. Strong solution overflows into a peripheral launder. Solids settle to the bottom, are raked to the center, and flow into the top of the second compartment, directly below, via an inverted cup and seal.

Integral with the sludge seal is a down-cast boot which projects into the second compartment and serves as a mixing well. Wash solution enters here to repulp and dilute the settled sludge just prior to its being thickened again in the second compartment.

The wash solution used in the second compartment is the overflow from the third compartment and is controlled in section *B* of the overflow box. The overflow from the second compartment is controlled in section *A* of the overflow box and is generally returned to the agitator or other processing steps and sometimes to the first compartment. Settled sludge passes from the second to the third compartment as before, being repulped and diluted on the way with a weak overflow solution from the fourth compartment.

This operation is repeated twice more, first in the fourth compartment and then in the fifth or bottom compartment. In each case, as before, the sludge is repulped with a weak wash solution overflowing from a later stage of decantation. In each case, after thickening, the sludge flows through the seal to the next stage of washing, and the overflow is collected and utilized for washing purposes in an earlier stage of decantation.

Solids pass downward, compartment by compartment, against a counterflow of wash solution and finally of fresh water. Thus, in accordance with the basic C.C.D. principle, the solids transfer their dissolved values to the wash solution and finally are removed by a diaphragm pump from the bottom compartment virtually free from values. Similarly, the wash water, as it flows through the system, becomes increasingly enriched in dissolved values until it overflows from compartment 2 and returns to process.

The washing type of thickener is especially adapted to relatively small C.C.D. operations where floor space is limited for a multithickener washing series. It is well adapted to the handling of hot solutions since it is easily insulated against temperature drop. It is regularly supplied in diameters up to 60 ft. and depths up to 40 ft., this depth corresponding to a machine with five compartments.

Economic Advantages of Washing-tray Thickener Compared with a Multithickener C.C.D. Series. A comparison between a five-thickener C.C.D. plant and its equivalent in the form of a single, five-compartment washing-tray thickener shows the following points of advantage in favor of the latter:

	C.C.D. plant	Tray thickener	Saving, per cent
Floor area	8,439 sq. ft.	2,615 sq. ft.	69
Exposed radiating surface	21,128 sq. ft.	12,367 sq. ft.	41
Building volume	270,929 cu. ft.	122,126 cu. ft.	55
Diaphragm pumps	5	2	40
Foundations	5	1	80
Power consumption	10	3	70

Typical Washing-tray Thickener Operating Data. At Cactus Mines Co., Mojave, Calif., there are two washing plants operating on identical feeds and discharging to identical further processing. The feeds are gold flotation tailings cyanided in a continuous agitation series. One washing system, receiving 40 per cent of the flow, consists of four single-compartment thickeners—24 ft. in diameter by 10 ft. deep, arranged in the conventional C.C.D. manner. The other system, receiving 60 per cent of the flow, is a four-compartment washing-tray thickener, 35 ft. in diameter by 27 ft. deep. Flows to each are proportional to the settling areas provided, which in turn determine capacity.

The two tables that follow are from Johnson, *Am. Inst. Mining Eng.*, *Tech. Pub.* 1082, 1939. They show that the solution strengths are approximately the same in the corresponding steps of the two systems and that the over-all recovery of dissolved gold, which is really the washing efficiency, is only ½ per cent less in the tray unit than in the individual thickener unit. The installed cost of the tray unit per ton of capacity was 25 per cent less than for the individual thickener unit—a more than sufficient saving to offset the slightly lower recovery secured.

Table 9. Comparative Solution and Residue Values

Assays, oz. Au per ton

	Thickener unit	Tray unit
Overflow solution 1	0.0344	0.0340
Overflow solution 2	.0147	.0146
Overflow solution 3	.0084	.0077
Overflow solution 4	.0033	.0036
Underflow residue 4	.0196	.0200
Underflow solution 4	.0037	.0046

Table 10. Comparative Efficiencies of the Two Units

C.C.D. Thickeners and Repulpers

Washed tails No. 4 thickener	0.0196 oz. =	\$0.686 per ton ore
Underflow solution No. 4 thickener	0.0037 oz. =	0.1295
Soluble loss per ton of ore	1.26 × 0.0037 =	0.163
Total gold dissolved per ton:		
Agitator No. 1 heads	0.16 oz.	
Agitator No. 3 residue	0.023 oz.	
Dissolved in agitators	0.137 oz. =	\$4.795
Gold dissolved in C.C.D. (0.023–0.0196) × \$35	=	0.119
Total dissolved		\$4.914
Soluble loss		0.163
Total dissolved gold recovered per ton		\$4.751
Percentage recovered	$\frac{\$4.751}{4.914} \times 100 =$	96.6 per cent

Washing Tray Thickener

Washed tails No. 4 compartment	0.020 oz. =	\$0.700
Underflow solution No. 4 compartment	0.0046 oz. =	0.161
Soluble loss per ton of ore	1.19 × 0.0046 =	0.1916
Total gold dissolved per ton:		
Agitators		\$4.795
Dissolved in washing thickener (0.023–0.02) × \$35	=	0.105
Total dissolved		\$4.900
Soluble loss		0.1916
Total dissolved gold recovered per ton		\$4.7084
Percentage recovered	$\frac{\$4.7084}{4.9000} \times 100 =$	96.1 per cent

Accessories. Successful operation of a C.C.D. plant depends to a very large extent on the continuous feeding of measured volumes of raw materials to the agitators and wash water to the last thickener in the series and to the intimate mechanical mixing of solutions and sludges in the repulping troughs between thickeners. Certain accessory devices have been developed to satisfy these essential requirements.

1. Feeders for Rock or Other Solid Substances (See p. 2286). Feeders must be accurate to at least 1 per cent, adjustable to deliver a moderate range of tonnages, able to start easily under load, and easily operated and adjusted by common labor.

The belt-type weigh meter (Fig. 28) has been found very satisfactory in C.C.D. practice. The material is drawn from the feed hopper in the form of a continuous ribbon by a short endless belt located at the bottom of the hopper. The drive pulley is connected to a constant speed motor. The belt passes over rollers balanced by an adjustable weighing element, which in turn actuates the sliding gate at the hopper opening through which the loaded belt issues. Once the weights on the beam of the weighing element have been set at a certain point, the machine continues to deliver the tonnage or poundage desired. The more closely the material is sized the greater is the accuracy and vice versa. In the case of −100-mesh, dry, pneumatically sized bauxite, the machine is said to be accurate to within 0.25 per cent.

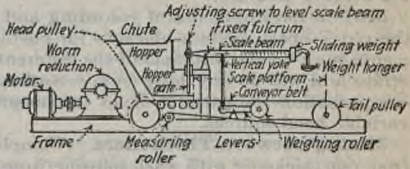


FIG. 28.—Belt-type weigh meter. (*Courtesy of Schaffer Poidometer Co.*)

Other feeders of reliability and fair accuracy are the belt feeder, apron (or pan) feeder, rotary-drum feeder, revolving-disk feeder, reciprocating-plunger feeder, etc.

2. Solution Feeders (See p. 2290). There are various feeders on the market for this purpose, namely, those of the adjustable displacement plunger-pump type, revolving bucket-wheel type, and weir (or V-notch) type, many of which may be supplied in materials resistant to corrosive solutions.

The arrangement shown in Fig. 27 is simple in construction and operation and is sufficiently accurate for feeding acid or other chemicals to a C.C.D. plant.

Referring to Fig. 29, acid, stored in a lead-lined tank, is continuously kept in circulation by an acid-resisting plunger pump which delivers the acid to a small lead feed tank with a gravity-return line to maintain a constant acid level therein. A lead siphon, adjusted by a handwheel, feeds the required amount of acid to the agitators, the rate of feed for different positions of the siphon being determined experimentally and the results transferred to a calibration scale for the use of the operator.

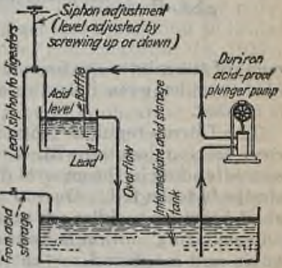


FIG. 29.—Simple acid feeder.

Some engineers prefer either adjustable speed or adjustable stroke-plunger pumps or orifice-controlled flow with variable head.

Wash-water Feeders (See p. 2286). An arrangement somewhat similar to that used for feeding acids or solutions may be utilized for regulating the amount of wash water added in the last thickener of the C.C.D. series. In this case, the intermediate storage tank is supplied with water by a float-controlled valve so that virtually no attention is required.

V-notch meters are sometimes used, these being mounted in boxes attached to, and connected with, the small feed tank. The overflow-return pipe in the feed tank is vertical and passes through the tank bottom. Removable adjusting rings of various widths may be placed on the top of the vertical overflow pipe to change the water level and accordingly the amount of water passing the V-notch per unit of time.

Heat exchange coils may be submerged in the feed tank where it is desirable to maintain the contents of the agitators and thickeners at an elevated temperature. Condensate or low-pressure exhaust steam is generally used as the heating medium.

Modern developments of recording and controlling meters have rendered the old V-notch system obsolete for chemical work. Most of these instruments work on the principle of the differential pressure on the two sides of an orifice and will do anything from recording and controlling of flow including compensation for variable head to the proportionate mixing of flows from variable head sources.

Mixing between Thickeners. Complete mixing of the thickened sludge from one thickener with wash solution from another is essential before resettlement of the mixture so formed. Incomplete mixing takes place in the inclined feed troughs, even if baffles are used, and accordingly mechanical agitation is needed.

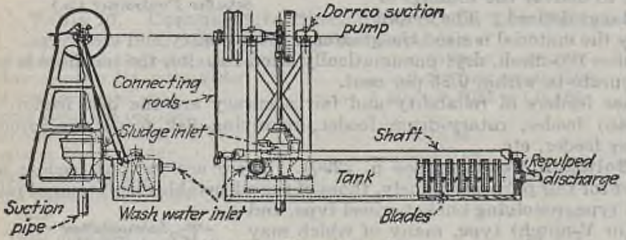


FIG. 30.—Dorrco repulper.

The Dorrco repulper (Fig. 30) consists of a trough of square or rectangular cross section, located with one end close to a sludge pump in the C.C.D. series and extending in the general direction of the thickener into which the repulped sludge is to be fed. On top of this trough, and running parallel with it, is a shaft to which paddles are attached. Through a crank at one end of the shaft an oscillating motion is imparted to the paddles. The paddles are set at an angle and staggered so that in addition to providing thorough mixing they convey the pulp toward the discharge end of the repulper.

The repulper is driven direct from a Dorrco pump by means of a connecting rod or may be equipped with an individual motor-drive unit. The trough may be set at a slope of $\frac{1}{4}$ in. per ft. instead of the $\frac{3}{4}$ in. per ft. required with the non-agitated trough thus saving a great deal of head room in a multi-thickener C.C.D. plant. It is supplied in standard and acid-resisting construction in lengths up to 50 ft.

FILTRATION

BY DONALD F. IRVIN

Definition. Filtration is defined here as the separation of solids from a liquid and is effected by passing the liquid through a porous medium. The solids are retained upon the surface of the medium in the form of a cake.

Purpose of Filtration. The purpose of filtration in industrial work is to separate the liquid from solids suspended in it, either one or both being valuable.

For instance, in causticizing plants, "lime mud" (chiefly calcium carbonate) is filtered from a solution of sodium hydroxide. In this case, the object of filtration is to remove caustic solution from the lime mud as completely as possible, obtaining high percentage separation of caustic and at the same time producing lime-mud cake nearly free from soda so that the cake may be calcined and converted into quicklime which is again used in the further production of caustic soda.

The petroleum refiner filters wax from paraffin-base oil which he is processing, both products being valuable.

The metallurgist filters cyanide-slime pulp, obtaining valuable gold- and silver-bearing solutions and discards the solids as unprofitable for further treatment; or he filters concentrates obtained by flotation and discards the water.

In many of these processes, filtration is preceded by thickening the pulp, enabling high capacity to be obtained from the filter and separating an initial portion of clear solution. This is dealt with under Sedimentation (p. 1619).

Usually when the liquid contains a very small proportion of solids in suspension, clarification by settlement should precede filtration. There are exceptions to this statement, but it is of quite general application.

When the solids are present in small proportion, the process is usually spoken of as **clarification.** In this article it is not intended to deal with the clarification of potable water or of municipal water supplies or boiler-feed water. In most cases, however, there is a comparatively large volume of solids to be removed from the liquid. Between the two extremes we have all proportions which might be found in a single industry.

The broad application of industrial filtration is realized by a survey of industries in which filtration plays an important part: pulp and paper, metallurgy, oil refining, chemical manufacturing, beet- and cane-sugar milling, sugar refining, sewage disposal, cement manufacture, etc.

THEORY OF FILTRATION*

By Hugh W. Bellas

Filtration has been developed as a practical art rather than as a science and the theory of filtration has received little attention in industry. The mathematical aspect of this unit operation, as developed by Lewis, Ruth, Cannon, and others, is presented here in the belief that a rational explanation in support of practical results has always been desired.

---

* This brief treatment of the theory of filtration is included to provide a basis for the study of theory in relation to practice.

Filtration theory, while seldom used in the actual design of a filter for a given operation, is valuable in interpreting laboratory tests, in seeking the optimum conditions for filtration, and in predicting effects of changes in operating conditions. The use of filtration theory is limited by the fact that the filtering characteristics must always be determined on the actual slurry in question, data obtained on one slurry being inapplicable to another.

Filtration usually results in the formation of a layer (or cake) of solid particles on the surface of the porous body, frequently a textile fabric, that forms the filtering medium. Once this layer has formed, its surface acts as the filter medium, solids being deposited and adding to the thickness of the cake while the clear liquor passes through. The cake is therefore composed of a bulky mass of particles of irregular shape, among which run small capillaries. The flow of liquor through the capillaries is always streamline and may therefore be represented by Poiseuille's equation, which may be adapted in the following form:

$$\frac{dV}{A\,d\theta} = \frac{P}{\mu\alpha[(W/A) + r]} \tag{1}$$

[Carman, *Trans. Inst. Chem. Engrs.* (*London*), 16, 174 (1938); also Walker, Lewis, McAdams, and Gilliland, "Principles of Chemical Engineering," McGraw-Hill, 1937], expressing the differential or instantaneous rate of filtration per unit area as the ratio of a driving force, pressure, to the product of viscosity by the sum of cake resistance and filter medium resistance.

The rate of filtration can usually be expressed in terms of volume of filtrate collected V, area of filtering surface A, and time θ. The pressure P is the total drop through the filter medium and the cake upon it. The viscosity μ is that of the filtrate. (Any convenient units may be used, inconsistencies being absorbed in the cake and cloth resistances.)

W is the weight of dry-cake solids, which may be replaced by one of several equivalent terms, since

$$W = wV = \left(\frac{\rho c}{1 - mc}\right)V$$

where w is the weight of dry-cake solids per unit volume of filtrate, ρ is the density of the filtrate, c is the weight fraction of cake solids in the solute-free slurry, and m is the weight ratio of washed wet cake to washed dry cake.

The symbol α represents the average specific cake resistance, which is a constant for the slurry in its immediate condition. In the usual range of operating conditions it is related to the pressure by the expression

$$\alpha = \alpha' P^s$$

where α' is a constant determined largely by the size of the particles forming the cake; s is the cake compressibility, varying from 0 for rigid, incompressible cakes such as fine sand and kieselguhr, to 1.0 for very highly compressible cakes. For most industrial slurries, s lies between 0.1 and 0.8. The symbol r represents the resistance of unit area of filter cloth, as well as pressure drop in lines, etc.

Equation (1) can be integrated as follows for constant-pressure filtration, giving the relationship between the total time and filtrate measurements:

$$\frac{\theta}{(V/A)} = \frac{\mu\alpha}{2P}\left(\frac{W}{A}\right) + \frac{\mu r}{P} \tag{2}$$

$$\frac{\theta}{(V/A)} = \frac{\mu\alpha w}{2P}\left(\frac{V}{A}\right) + \frac{\mu r}{P} \tag{2a}$$

For a given constant-pressure filtration, these may be simplified to

$$\frac{\theta}{(V/A)} = K_p\left(\frac{W}{A}\right) + C = K_p'\left(\frac{V}{A}\right) + C \tag{2b}$$

where K_p, K_p', and C are constants for the conditions employed.

Equation (1) may be integrated for constant rate of filtrate flow (or cake deposition) to give the following equation, in which filter-medium resistance is treated as a constant pressure to be deducted from the rising total pressure [Ruth, *Ind. Eng. Chem.*, **27**, 717 (1935)]:

$$\frac{\theta}{(V/A)} = \frac{1}{(\text{rate per unit area})} = \frac{\mu\alpha}{(P - P_1)}\left(\frac{W}{A}\right) \tag{3}$$

which may also be written:

$$\frac{\theta}{(V/A)} = \frac{1}{(\text{rate per unit area})} = \frac{\mu\alpha w}{(P - P_1)}\left(\frac{V}{A}\right) \tag{3a}$$

In these equations P_1 is the pressure drop through the filter medium.

$$P_1 = \mu r\left(\frac{V}{A\theta}\right)$$

For a given constant rate run, the equations may be simplified to

$$\frac{V}{A\theta} = \text{rate per unit area} = \frac{P}{K_r} + C' \tag{3b}$$

where K_r and C' are constants for the given conditions.

In the filtration of small amounts of fine particles from liquids by means of bulky filter media (absorbent cotton, felt, etc.), it has been found that the above equations based upon the resistance of a cake of solids do not hold, as no cake is formed. For these cases, where filtration takes place in the capillaries of a thick medium, Hermans and Bredee [*J. Soc. Chem. Ind.*, **55T,** 1–4 (1936)] have developed equations which they have found applicable to the constant-pressure filtration of viscose, sugar solutions, etc.

Practical Significance of the Filtration Equations. The differential form, Eq. (1), of the filtration equation yields interesting information on the mutual effects of the operating variables.

When the cake is composed of hard, granular particles that make it rigid and incompressible, an increase in pressure results in no deformation of the particles or their interstices, whereby $s = 0$, and, neglecting filter-medium resistance, Eq. (1) becomes

$$\frac{dV}{d\theta} = \frac{AP}{\mu\alpha'(W/A)}$$

For incompressible cakes, therefore, the flow rate is directly proportional to the area and pressure and inversely to the viscosity, to the total amount of cake (or filtrate), and to α'.

When the cake consists of extremely soft, easily deformed particles, such as ferric and other metal hydroxides, s approaches 1.0, whereby Eq. (1), again neglecting the filter medium, reduces to

$$\frac{dV}{d\theta} = \frac{A}{\mu\alpha'(W/A)}$$

For very compressible cakes, therefore, the rate is independent of pressure.

The *effect of pressure* shown above is modified in most industrial filtrations, where the cake compressibility usually lies between 0.1 and 0.8. Furthermore the resistance of the filter medium reduces the effects of the respective variables. It has been found true, however, that in the filtration of granular or crystalline solids an increase in pressure causes a nearly proportionate increase in flow rate. Flocculent or slimy precipitates have their filtration rates increased but slightly by an increase in pressure. Some materials have a critical pressure above which a further increase results in an actual decrease in flow rate.

In the filtration of certain non-homogeneous sludges, such as those of slimy solids to which filter aids have been added, it has been found that a constant flow rate during filtration is more satisfactory than a constant pressure, which latter results in poor initial clarity of the filtrate and a rapid build-up of cake resistance. As a matter of fact, filtration of any but the most incompressible sludges is more satisfactory when a low pressure is used at the beginning of the run. This is especially important in filtering slurries of low solid content.

Since most pressure filters are fed by centrifugal pumps, their operation is seldom either constant pressure or constant rate but, in accordance with the characteristic of the pump, essentially constant rate during its early stages and constant pressure during much of the later part of the cycle. Pumps having steep head-discharge characteristics do not operate at either constant rate or constant pressure during any part of the cycle, but always under intermediate conditions of increasing pressure and decreasing flow rate.

Cake thickness is an important factor in determining the capacity and design of a filter, and upon it the cycle of operation depends. Filtration theory shows that, cloth resistance neglected, the average flow rate during a filtration is inversely proportional to the amount of cake deposited.

If the cake has a high resistance relative to that of the filter medium, therefore, the highest capacity of a given filter is reached with zero cake thickness. Consideration of the fact that a thin cake does not usually discharge easily, however, together with the important factor of time required to clean the filter, leads to the selection of an appreciable cake thickness. Filter capacity is often measured in terms of dry solids handled per unit of filtering area.

If the cake has a low resistance compared with that of the filter medium, the economic cake thickness will be increased.

In washing filter cakes it is usually found that there is a definite cake thickness at which a given ratio of wash water to cake solids will produce a minimum soluble salts content of cake. Conversely, the ratio of wash water to cake solids which is found necessary to produce a given soluble content of the cake is a minimum at this cake thickness. In many cases, however, the effect of cake thickness on washing efficiency is not marked. Minimum volume of wash water is desirable since excessive volumes may derange plant procedure.

The *effect of temperature* upon the filtration rate of most incompressible cakes is evident through its effect on viscosity. A temperature rise lowers the viscosity of the filtrate and causes the flow rate to change in inverse proportion to the viscosity.

Many compressible sludges are affected in other ways by temperature change, although the general effect is toward an increase in flow rate with temperature.

The *effect of particle size* on cake and cloth resistances is marked. Even small changes in particle size affect the coefficient α' in the equation for cake resistance, $\alpha = \alpha' P^s$, and larger changes affect the compressibility s. Decreased particle size results in lower filtration rates and higher moisture content of the cake but sometimes in better washing efficiency. It is important, therefore, that close control be kept of the particle size in the feed to the filter. Agglomeration of particles by coagulation is often an important aid in filtering difficultly filterable materials.

It has occasionally been found possible to increase the filtering rate of a slurry by adding larger non-compressible particles to it. Where there is a very wide range in the size of particles in the slurry, however, care must be taken to avoid excessive settling in the filter.

The *effect of the type of filter medium* is often not fully recognized. In selecting the medium for a given filtration, a balance must be struck between as open a weave as possible in order to reduce plugging and as tight a weave as is necessary to prevent excessive "bleeding" of fine particles. After a small thickness of cake has formed on the medium, bleeding often stops, fine particles being caught in the cake.

Of the weaves of filter cloths described under a following section, the number duck weaves have the greatest ability to retain fine solids, followed in decreasing ability by chains (broken twills), twills, and hose ducks. The tendency to plug, however, is in the reverse order. Thick, stiff cloths tend to plug more readily than thin, pliable ones. The effect of cloth plugging on filtration rate is so appreciable that it will ultimately be the cause of replacement of the cloth. It also results in a need for using a safety factor in predicting filter capacities.

The *effect of solid content* of the slurry on the rate of filtration is shown in Eqs. (2*a*) and (3*a*), where it is expressed as w, the weight of cake-forming solids per unit volume of filtrate. These equations show that, filter-medium resistance neglected, the rate of filtrate flow is inversely proportional to the ratio of solids to filtrate but that the rate of cake deposition is directly proportional to this ratio. If a slurry is thickened before filtration, time required for its filtration on a given filter area will be reduced in direct proportion to the decrease in ratio of liquid to solids in the slurry.

Application of Filtration Theory to the Interpretation of Data

The filtration equations are useful in predicting the effect of a change in any variable if the constants are determined from data taken on the slurry in question. For example, vacuum test data can be extrapolated to show the approximate filtering rates that could be obtained if the slurry were filtered under pressure. Another problem often of interest is the effect of cake thickness or time cycle on over-all filtration rate.

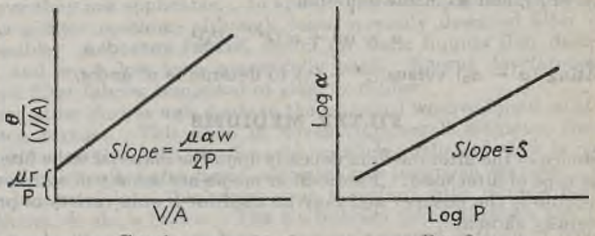


Fig. 1. Fig. 2.

Figs. 1 and 2.—Typical plots of filtration data.

If a *constant pressure test* is run on a slurry, care being taken that not only the pressure but also the temperature and the solid content remain constant throughout the run and that time readings begin at the exact start of filtration, one can observe values of filtrate volume or weight and time. With the use of the known filtering area, values of $\theta/(V/A)$ can be calculated for various values of (V/A) which, when plotted with $\theta/(V/A)$ as the ordinate and (V/A) as the abscissa (Fig. 1), result in a straight line having the slope $\mu\alpha w/2P$, and an intercept on the vertical axis of $\mu r/P$. Since μ, w, and P are known, α and r can be calculated from

$$\alpha = \frac{2P}{\mu w} \times (\text{slope})$$

and

$$r = \frac{P}{\mu} \times (\text{vertical intercept})$$

The effect of a change in any variable except P (which affects α) may now be estimated.

To determine the effect of a change in pressure, it is necessary to run a test under one or more other pressures, and to calculate α and r at those pressures in the same way. By plotting α and r against P on loglog paper (or by plotting $\log \alpha$ and $\log r$ *vs.* $\log P$), straight lines result from which it is possible to determine α and r at any reasonable pressure (Fig. 2). In many cases it has been found that r does not vary appreciably with pressure, in which case an average value can be used at all pressures. This is often due to the fact that a low filtering pressure is used at the start of filtration.

When a low pressure is used for any appreciable time at the start of filtration, the beginning of time and filtrate readings should be delayed until the constant pressure is reached, in which case r is the resistance of the filter medium plus that of the cake deposited at low pressure. When the weight of the dry cake is measured *vs.* time, as is usually done in vacuum leaf tests, two or three tests are sufficient to permit plotting the straight-line function of θ/V *vs.* W. The ability to interpolate or extrapolate on this line eliminates the need for a large number of tests.

In *constant rate filtration* it is suggested that the method of Bonilla [*Trans. Am. Inst. Chem. Engrs.*, **34**, 243–250 (1938)] be used, involving the determination of P_1, and α_0 in the equation

$$\alpha = \alpha_0 + \alpha'(P - P_1)^s$$

and plotting $(\alpha - \alpha_0)$ versus $(P - P_1)$ to determine α' and s.

FILTER MEDIUMS

The choice of the filter medium depends upon the material to be filtered and upon the type of filter used. Fabric-filter media are usually of cotton or wool, and as cotton is the cheaper and may be used for a wide variety of products, it is generally chosen.

Wool cloths are more suitable when acid solutions are to be filtered. The weight and grade used are determined by the clarity desired and the local conditions.

The fabrics most often used are cotton, in duck, twill, and chain weaves. Duck is a *plain cloth* of square weave having "warp" and "weft" or "filling" threads equal in thickness and texture, and in weaving they are passed over and under each other alternately, thus producing a weave of square appearance. "Hose duck" is somewhat open in texture and when held up to the light may show tiny square orifices due to the threads not being tightly woven. *Regular ducks, number ducks,* or *roll ducks* are more closely woven by keeping greater tension on both the warp and fill threads and driving them together tightly during weaving. A single-fill duck has a single thread for each warp or filling. A double-fill duck has two threads twisted together to form each warp or filling and is designated as two ply; and so on. The object of increasing the ply is to obtain a closer weave and, if the threads are tightly driven together, the fabric is correspondingly closer in texture. Many cloths have numbers arbitrarily assigned to them by the manufacturers. These numbers do not give the weight of the fabric per square yard but enable this information to be obtained from a table of weights supplied by the manufacturer. It varies directly with the *count* or number of warp and fill threads per inch.

Twill fabrics invariably exhibit diagonal ribbing due to the way in which they are woven. In the simplest twill, the fill or weft passes over two warps and then under two; and so on. The next fill does the same but is one warp "out of step" so that it separates each pair of warp threads that were formerly together. This gives a diagonal rib at 45 deg. if the number of warp and fill threads per inch are equal. Differences in threads or in weave alter the angle of rib. **Twill and chain weaves** are described like ducks according to the ply in their threads, so that a 3 by 3 has three ply in both warp and fill threads. The count is also given, for instance a 36 by 26 fabric has 36 warp threads and 26 fill threads per inch. Its weight is separately expressed as 18 oz., or 18 oz. per sq. yd. For instance, a No. 11 twill weighing 17 oz. per sq. yd. 4 by 4 threads and 45 by 30 count. When ordering cloth for filter mediums, a sample should be submitted to avoid mistakes due to confusion of the designating numbers.

Metallic filtering mediums are used extensively on continuous vacuum filters for handling alkalies, crystalline material, wood pulps, and paper pulps. They are also used in pressure filters in conjunction with a filter aid for clarifying sugar liquors, lubricating oils, gasoline, etc. Metallic filter mediums are woven of iron, copper, or brass wire, or of special corrosion-resisting alloys such as monel. Mechanical strength and resistance to corrosion prolong the life of woven-wire filter mediums to such an extent that higher first cost is amply justified and for this reason such mediums should be considered whenever they are applicable. In special cases finely punched metal plate is used as a filter medium, although other specially designed filter mediums are possible. **Asbestos fabric,** useful for some liquids that destroy both cotton and wool, has been successfully used. Recent developments have provided filter fabrics composed of glass or rubber.

Woven glass cloth is now made in the principal weaves found most suitable in cotton textiles. This cloth is woven from small diameter glass fibers; its use is for services wherein the acid-resisting quality of glass is required.

Rubber fabrics are also made, although not in woven textiles. The fluid rubber is passed in a thin sheet over a surface with minute perforations through which air is blown. The air bubbles rising through the gradually solidifying rubber sheet produce a regularly spaced pattern of fine perforations, which may be as numerous as 6400 per sq. in.

The development of a rubber fabric having microscopic apertures is in progress; it has high rates of porosity in terms of water.

Two new synthetic yarns being applied to filter cloths are Vinyon and Nylon. The former has excellent resistance to cold acids and alkalies, while Nylon is required for higher temperatures.

Metacloth is the trade name of a filter cloth that has been treated with copper sulfate solution to make it mold-proof and to increase its resistance to caustic solutions.

Dupont Nitro-filter is cotton fabric nitrated in such manner that the tensile strength of the fabric is from 70 to 80 per cent of the untreated cloth. It is used for the filtration of sulfuric, nitric, or hydrochloric acids or mixtures of these, of varying strengths and temperatures, for instance 40 per cent sulfuric acid at 90°C. Being a nitrocellulose it will not withstand alkalies, and because of its flammable nature must be kept wet and is stored under water. Special cotton-filter cloths prepared by impregnating with solutions of secret formulas are available for the filtration of caustic solutions.

Factors Governing Selection. The filter medium selected must have good mechanical strength and resistance to the solution filtered, so that it has an economic life.

The clarity of filtrate desired and the size of particle to be filtered together determine the class of weave selected and the grade or weight of cloth.

Where the first portion of filtrate may be returned to the circuit as a "cloudy" liquor, a cloth of open texture is permissible and larger filter capacity is obtainable than when a close-textured cloth is used to obtain the entire filtrate at high clarity.

Pigments, non-corrosive chemical precipitates, metallurgical products, and weak alkalies may be filtered on cotton-duck or twill weave, the latter giving the higher clarity. Woolen fabrics are used for dilute acid liquors.

Woven-wire cloth is used chiefly for acid and alkaline liquors. Monel and other alloys are very useful because of their resistance to corrosion. Open-weave metallic cloth is especially useful for paper pulps and crystalline and granular materials because rapid drainage and high capacity are obtained per unit area.

FILTRATION LEAF TESTS

It is unusual to be able to forecast what may be accomplished in the filtration of an untested product, and even the results obtained upon known products vary greatly with the conditions of filtration. Therefore, unless exact data have already been established, preliminary tests should be made to determine the filter requirements for a given filtration problem. Such tests are easy to make and require very simple, small-scale test equipment. Whether vacuum or pressure filtration is to be used is generally known beforehand. Occasionally tests are made for comparison.

Vacuum Tests. The leaf shown in Fig. 3 is connected to a filtrate receiver equipped with a vacuum gage. The receiver is connected to an aspirator. Different filter mediums may be used on this leaf for comparative tests.

In making leaf tests, the operation of a continuous vacuum filter should be kept in mind. The cycle is divided into three periods, **cake formation** (or "pickup"), **drying**, and **discharge**. Sometimes pickup is followed by a period of displacement washing, and the cake may also be subjected to compression during drying. These things should be considered, and a plan of the cycle or cycles to be tested should be formed.

If the object of filtration is simply the removal of solids from the liquor, the cycle may be: one-third pickup, one-third drying, and one-third discharge and reentry time. While under vacuum, the test leaf is submerged for the pickup period in the material to be tested. The leaf is then removed and held with the drain pipe down for the drying time allotted. Observations should be made during the test such as vacuum readings during pickup and drying; time at which cracks in the cake appear; temperature of the material; percentage of cake-forming solids present; acidity or alkalinity.

Usually a few preliminary tests will indicate the time range. Careful tests may then be made and, in these, variations in temperature, dilution, conditioning agents, etc., should be tried, and capacities and clarity of filtrate noted.

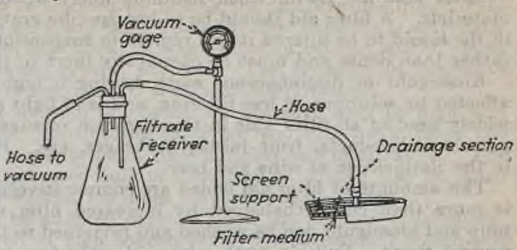


Fig. 3.—Small-scale vacuum-filtration testing unit. *(Oliver United Filters, Inc.)*

Pressure Tests. For plate-and-frame press work, tests are best made with a laboratory-size model. This will give representative "cake packing," etc. The apparatus shown in Fig. 4 is used for tests to obtain data for operation with a shell-type pressure filter.

Operation of the commercial unit should be kept in mind and the cycle arranged accordingly. After determining the cake-building or filling time, displacement washing and drying the cake with compressed air should be tried. For wet discharge it is advisable to open the cell and experiment upon washing the cake away with a jet of water. For dry discharge the effect of a gentle air blast in the test leaf should be tried.

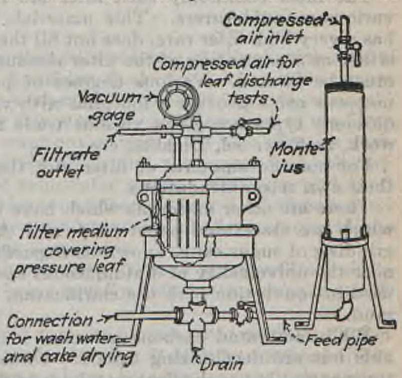


Fig. 4.—Small-scale pressure-filtration testing unit. *(Oliver United Filters, Inc.)*

In both vacuum and pressure tests the daily filter capacity is determined by the dry weight of cake per unit area of test leaf multiplied by cycles per 24 hr. and multiplied by the filter area. Capacity in solids is usually expressed in pounds per square foot per day and filtrate in gallons per square foot per minute or per day.

The material tested should be representative, and samples should be tested immediately after they are taken. In some instances, samples stored for several days have given results very different from those obtained when tested immediately because of changes that occur upon standing. All tests should be made under conditions that represent large-scale operations so far as possible.

Results obtained by leaf tests for capacity are irregular with extremely free-filtering materials, such as crystals in mother liquor. In such cases it is better to employ small-scale equipment.

Before undertaking test work it is advisable to consult a manufacturer of filtration equipment, giving as many data on the materials as possible, together with the objects of filtration.

FILTER AIDS*

Filter aids are useful when handling finely divided solids and colloidal materials. A filter aid should be of low specific gravity so that when mixed in the liquid to be filtered it will remain in suspension. It should be porous rather than dense and must be chemically inert to the liquid being filtered.

Kieselguhr or diatomaceous earth having a high silica content is little affected by solutions, is free filtering, and is of light gravity. It is the most widely used of all filter aids in the filtration of sugar juices, vegetable oils, petroleum products, fruit juices, beverages, etc. Paper pulp is also used in the clarification of wine and beer.

The amounts of filter aid added are comparatively small, and the expense is more than counterbalanced by increased filter efficiency. Both paper pulp and kieselguhr can be washed and revivified so that they may be reused several times. Fuller's earth, charcoal, asbestos, sawdust, magnesia, salt, and gypsum are used as filter aids in special cases.

Decolorizing carbons and earths, such as Darco, Carbrox, Suchar, Norit, Filtrol, Palex, and activated clays act both as decolorizers and as filter aids for oils, fats, etc. In many cases, a coating of the filter aid is applied to the filter medium to act as a clarifying agent and to prevent blinding of the filter medium.

The most commonly used filter aid is diatomaceous earth prepared by various manufacturers. This material, being skeletal remains of diatoms, has a very high filter rate, does not fill the pores of filter mediums, and is used either as a precoating of the filter medium itself, or as a pulp mixture which must be filtered. Various degrees of purification used on the raw diatomaceous earth provide a material with varying filtration properties. These different types are given various trade names identifying them for certain work, as Filter-cel, Dicalite, etc.

For specific amounts of filter aid, the producers of various grades give their own recommendations.

There are other materials which have been employed as filter aids, among which are macerated paper pulp, and the finely shredded residue from the grinding of sugar cane, known as *Bagacillo*. Neither has attained anywhere near the universality of diatomaceous earth as a filter aid. Bagacillo is only used in connection with the clarification and filtering of cane juice and cane mud.

While activated carbons are used in filtration they are not really filter aids but are decolorizing agents. Doubtless they do promote filtration to some extent but such effect may be considered incidental to their real function as decolorizing agents.

Preconditioning. Another method of preventing excessively fine particles from decreasing the filtration rate abnormally is to cause the fine particles to coalesce or to form agglomerations of larger size. This pretreatment or conditioning of an otherwise unsuitable feed provides good filtration rates and makes industrial filtration more profitable. Sewage sludge is an example of this method.

Coagulation of sewage sludge prior to filtration is effected by the use of such reagents as alum, ferric chloride, or other chemicals.

Effect of Temperature and Viscosity. By increase of temperature, water decreases in viscosity and gives rates of flow proportional to the following data:

---

* See p. 1269.

Temperatures	0°C.	20°C.	40°C.	60°C.
Rate of flow	1.0	1.8	2.7	3.7

Thus, the rate of flow is doubled by raising the temperature from 20° to 60°C. In practice there are many cases where the filter capacity is materially increased by heating the filter feed, as when filtering cement slurry, clays, some flotation concentrates, syrups, oils, etc. The economy of heating a given filter feed may be determined by tests and computations.

When a concentrated solution is filtered, its viscosity may be high and give a low filtration rate. By diluting the filter feed with fresh water, the viscosity of the solution is reduced and gives a higher rate of filtration. The total filtration time for increased bulk of filter feed as diluted may be much less than for the smaller bulk of the more viscous strong solution.

Whether dilution can be adopted depends upon the purpose of filtration. If the solution is required at high concentration for subsequent treatment, this may preclude dilution, but if it is to be discharged afterwards or if it may be reconcentrated by evaporation, then dilution may be allowable.

TYPES OF FILTERS

Filters may be conveniently grouped under four heads:

1. Gravity filters.
2. Pressure filters.
3. Intermittent vacuum filters.
4. Continuous vacuum filters.

Gravity Filters. A gravity filter generally consists of a tank with a false floor covered by a filter medium. Leaching tanks used in cyanide plants have a cloth-covered filter bottom and may be termed "gravity filters." By far the largest number of gravity filters employ a bed of sand as the filter medium and are used for clarifying solutions or water. To be effective such a filter medium must be comparatively thick and the amount of solution large compared with the amount of removable solids. Strainers and sand and charcoal filter beds, used for water purification, are good illustrations of gravity filters. Gravity filters are often useful for small-batch operations in chemical industries where corrosion is excessive.

The Nutsche Filter. This is the simplest form of gravity filter and is not made by any one manufacturer, being usually built by the plant in which it is to be used.

In most cases it consists of a simple support for the filter medium chosen and a vessel in which this filter medium is placed. It may be operated either by gravity drainage through the filter medium, by vacuum in which suction is applied at the bottom of the filter medium, or, in some cases, by pressure, in which case the filter is placed in a pressure case and the material pumped into it. These filters are found in small units adapted to batch operation and are of the simplest possible design.

Pressure Filters. There are two types of pressure filters in general use, **plate-and-frame** presses and **enclosed pressure** filters.

In plate-and-frame presses, a filter cloth is held between a cast-iron plate and a frame, assembled to form a cell unit. A number of these cells assembled in series form the filter.

Enclosed pressure filters have a number of filter leaves suspended inside a shell into which the material to be filtered is charged under pressure. The leaves may be parallel or perpendicular to the horizontal axis of the filter shell. In most instances the leaves are stationary, but in some cases they can be rotated during the filtration cycle. In one instance the filter medium is installed as an inner lining of the shell and rotates with the shell.

The principle of operation of all pressure filters is the same. A filtering medium is stretched over a frame provided with channels for the collection and drainage of solution, and the material to be filtered is forced under pressure into the space between the filter medium and outer housing or frame.

Plate-and-frame presses must be taken apart and cleaned by hand at the end of each cycle. Pressure filters having leaves enclosed in a shell are opened and closed by mechanical means and manual labor is reduced. In both types, a sluicing mechanism may be employed for discharging the cake, resulting in a material saving in time and decreased maintenance cost.

A disadvantage common to all pressure filters is their intermittent operation. The advantages are that high pressures may be used to dry the cake. Higher costs for labor and renewals of filter medium are characteristic of pressure filters.

Life of Wood Used in Plate-and-frame Presses under Acid-filtrate Conditions.* The Independent Filter Press Company, Inc., of Brooklyn, New York, advises that the life of various types of lumber under acid filtration differs considerably, and it is not possible to give a definite figure thereon.

They do state that after trying various kinds of lumber, long-leaf yellow pine was chosen to be the most durable. Even so, they state that the degree of acidity is the most important factor. Two cases are cited wherein long-leaf yellow-pine plates lasted, in one instance, under normal acid filtration, for 2 to 3 years; on the other hand, in a particular case where the conditions were trying, a set of long-leaf yellow-pine plates and frames lasted about 2 months. No record is available by this company as to the relative life of long-leaf yellow pine compared to maple or cypress in this work.

Intermittent Vacuum Filters. The intermittent type generally consists of a series of frames or leaves over which the filter medium is stretched, the leaves being provided with channels for the drainage of liquid.

Several leaves are connected to a common header, which is in turn connected to a vacuum line by flexible hose. By completely submerging the leaves in a tank of material to be filtered and applying vacuum, the cake is formed. As soon as sufficient cake has been formed, washing and discharging of the cake are performed by hoisting the leaves out of the tank, and, while still under vacuum, transporting them to a tank containing wash water or to the point where discharge is to be made. Discharge is accomplished by release of vacuum and inflating the leaves by compressed air.

The intermittent vacuum filter gained a wide use for some years in cyanide-plant operation, in separation of gold- and silver-bearing solutions, but has been supplanted generally by continuous vacuum filters.

Continuous Vacuum Filters. Continuous vacuum filters are of two types: the **rotary drum** and the **rotary disk**. The drum type is a cylinder whose periphery forms the filtering surface; this surface may be either external or internal and is divided into separate compartments. Each compartment is separately connected to an automatic control valve which regulates the period under vacuum for forming, and compressed air for discharging, the filter cake. The internal drum filter receives its feed inside the drum, but the external drum filter is mounted in a feed tank.

The top feed filter is an external drum filter which has no tank. Feed is supplied near the top of the drum and a hopper below the drum receives the cake when discharged.

* See page 2164.

The rotary-disk type has its filtering medium shaped in segments which are assembled to make up a disk. Each segment is connected, through a central axis or shaft, to an automatic valve, similar to that used on the rotary-drum filter. A number of disks may be assembled on the common shaft.

MAKES OF FILTERS

Plate-and-frame Presses. A plate-and-frame press consists of a series of solid vertical plates and hollow frames cast with side lugs, so that they may be mounted on two parallel horizontal bars and clamped together. Figure 5 illustrates a **Shriver press.** Each plate and frame is accurately machined to give a tight joint when clamped together with a cloth between them, and the faces of the plate are ribbed and channeled or have a pyramid surface to provide for drainage of the filtrate (Fig. 6). A filter cloth is laid over each plate to cover both faces, and a frame is set on each side of it so that by alternating plates and frames and clamping them together a series of filter cells is formed. Each cell consists of the empty frame bounded on both sides by filter cloth behind which are the plates.

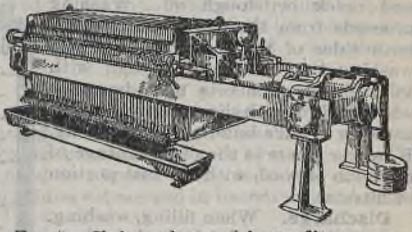
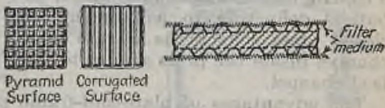


Fig. 5.—Shriver plate-and-frame filter press.

One end of the series is closed by the head of the press; the other end by the final plate against which a capstan screw is tightened, thereby clamping the series together and enabling the whole to be operated under pressure. A ratchet gear and pinion or hydraulic closing device may be used to obtain greater force. The feed channel is formed by a hole in each plate and frame, these holes registering together. In each frame there is an opening from this channel that admits feed into the frame, and at the bottom of each plate there is an outlet for the filtrate. If the filtrate from a plate becomes turbid, that particular plate is removed from service by closing the outlet cock. The filter cloths are examined after each run and any defective ones are replaced.



Pyramid Surface

Corrugated Surface

Fig. 6.—Filter plate.

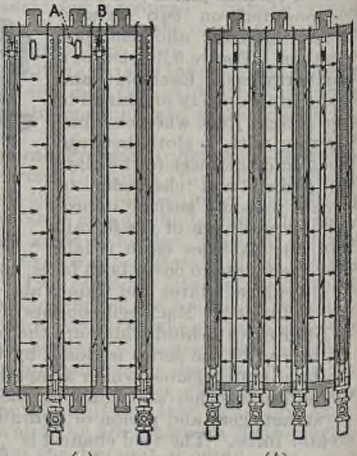
When necessary to wash the filter cake two methods are used depending upon the type of plates. In one of these, the wash water is forced through the feed channel and follows the filtrate. In the other method, wash is supplied from a channel that passes through the plates and frames similarly to the feed channel. From this wash channel there is an inlet in one corner of each alternate plate. The former construction is used for the method of operation called center filling and the latter for solid filling.

Solid Filling. When the solids quickly form a thick cake at low pressure (25 to 50 lb. per sq. in.), feed is continued until the frames are completely filled, as indicated by the filtrate ceasing to flow from the drain cock at the bottom. Filling may be followed by an air blow to expel solution before wash is applied, and as the cell is completely filled with solids no displacement of cake can occur though there may be cracking during this air blow. Washing is then performed by admitting wash solution or water under pressure behind the filter cloth of each alternate plate. The wash is forced through the cake and the cloth on the opposite side and issues through the drain cock shown in Fig. 7*a*. Air blow may again be given to displace the wash and to obtain as dry a cake as possible. Dryness of cake is an advantage of plate-and-frame presses.

Center Filling. In this method the feed is stopped while there is still a space down the center of each frame thus dividing the cake into two halves (Fig. 7*b*). Wash solution is forced in behind the feed without allowing the pressure to drop so that there is no interval during which the cake can dry and crack or slough off. Washing proceeds from the center out toward both sides of the frame so that less washing time is required than with solid filling. This is an advantage when dealing with slimy materials that are slow in cake building and washing. However, there is the disadvantage of diffusion of wash with the last portion of filtrate.

Discharge. When filling, washing, and air blow are finished, the cake is discharged. The labor involved in this discharging is the chief drawback of presses. The press screw is released, the plates and frames separated, and the frames are emptied into a tray or into a conveyor below the press.

Open and Closed Filtrate Discharge. An open press is one having a control cock to each plate outlet, all outlets delivering to a launder. A closed press collects its filtrate in a channel inside the press similar to the feed channel.



(*a*) (*b*)

FIG. 7.—Cross sections of plate-and-frame filter presses, showing operation of solid (*a*) and center (*b*) filling.

The advantages of plate-and-frame presses are simple construction, low cost, simplicity of operation, and dryness of cake. They may be used for high pressures, and for acid filtrates the plates and frames may be made of wood.

The disadvantages are high labor charges when handling large tonnages and high cloth consumption due to damage at the joints when opening and closing the press. Washing is imperfect and the percentage of idle time in the cycle is high because of the time required for opening and closing.

Failure at the edges of a cloth where it serves as a gasket between plate and frame cannot be remedied without dismantling the press and installing a new cloth; so for the balance of the cycle, the feed leaks at the point of defect resulting in losses and a dirty filtering plant.

The plate-and-frame press was the pioneer in filtration and still finds use in a limited field but, in general, may be said to have become obsolete.

Merrill Press. The Merrill press is essentially a plate-and-frame press with an automatic sluicing device that enables the cake to be discharged without opening the press.

Along the median line at the bottom of the press, and passing through the plates and frames, is a continuous channel, within which is a sluicing pipe with nozzles, one projecting into each compartment. This pipe is slowly rotated back and forth through an arc of approximately 180 deg., by means of a rack and pinion, driven from a line shaft or small individual motor. Simultaneously water under pressure issues from the nozzles and plays upon the slime cake in the chambers, washing it down into the annular space around the sluicing pipe from which it leaves the press through a number of discharge cocks. This press finds application in cases where discharge in the wet stage is permissible. It was originally developed to handle a large daily tonnage of slime in which the treatment with cyanide took place (at least partially) inside the press. Its success marked an advance in slime filtration by plate-and-frame presses since manual labor was reduced and cloth life lengthened. However, pressure filters have now been generally replaced by continuous vacuum filters for this work.

Merrill Precipitation Press. The collection of gold and silver precipitated from cyanide solution by zinc dust was formerly effected by using a plate-and-frame filter press of triangular cross section. This duty is now performed by an ordinary press of rectangular cross section or by a series of cylindrical cloth bags. The bags are fixed on a header connected to the discharge of a centrifugal pump and are at all times submerged in barren solution which passes outward through the cloth bags. To clean up the bags, the solution is drained from the tank, the bags are blown with compressed air, and the inner bag liners are then removed for fluxing and melting.

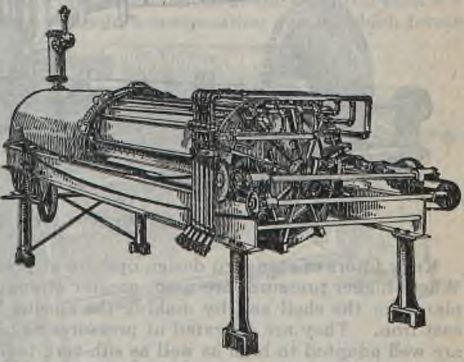


FIG. 8.—Kelly filter.

Kelly Filter. The Kelly filter consists primarily of a steel cylindrical tank enclosing a number of rectangular filter leaves. The axis of the tank is set approximately horizontal (Fig. 8). The leaves are supported by a carriage attached to the movable head of the cylinder. Filtrate is discharged through the head, which is locked to the shell by a set of radial bolts and a special mechanism. The solids form a cake on both sides of the leaves. To discharge the cakes the head is unlocked and moved back on its supporting rails, taking with it the attached carriage and leaves until all is free from the shell. The cake is then discharged by its own weight assisted by a slight back blast of air.

The leaves are rectangular and hang vertically in the shell. They are all of the same length but of varying widths. The widest one is on the vertical center line of the shell and the narrower ones on either side, their widths decreasing with the distance from the center so that proper clearance between the top and bottom of each leaf and the shell is maintained.

Each leaf consists of a heavy wire screen, bound on the four edges with a rolled steel shape similar in cross section to a slotted, flattened pipe. The screen forms the drainage element and the steel shape provides rigidity, protects the filter cloth from the edges of the screen, and serves as a channel for the filtrate. The upper corners of the leaves next to the head are connected to the head by nipples and unions forming outlets for the filtrate. The filtrate passes through these fittings and then through passages in the head to a trough or filtrate header outside.

The leaves are enclosed in bags of filter cloth. These bags are made with the front end open; they are slipped over the leaves and the open ends are sewed by hand. Metallic filter cloths may be used.

The largest Kelly filters are of the twin type, *i.e.*, there are two shells mounted on the same beams with the head ends facing one another and using a space common to both for running out the carriages. This arrangement (Fig. 9) gives a greater filter area per unit of floor space and therefore reduces the cost of the filtration process.

The two halves of the unit are opened and closed automatically, means being provided for releasing the chain opening device (which is common to both shells) from one half while opening and closing the other half. The maximum travel for the carriage of either half is fixed by a spring bumper.

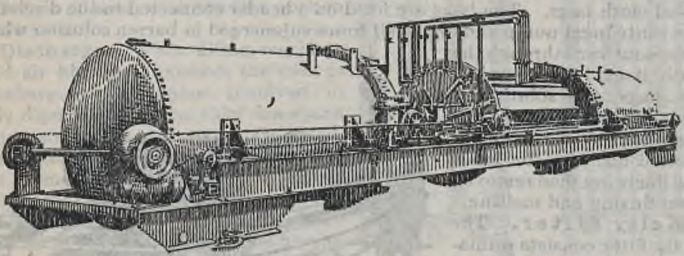


Fig. 9.—Kelly filter (twin type).

Kelly filters of standard design operate at pressures up to 60 lb. per sq. in. Where higher pressures are used, greater strength is secured by heavier steel plates for the shell and by making the closing head of cast steel instead of cast iron. They are operated at pressures as high as 250 lb. per sq. in. and are well adapted to high as well as sub-zero temperatures, due to the ease of insulation.

Operation. The material to be filtered is charged into the shell under pressure. The filtrate is forced through the cloth and through outlets from each frame into a launder, while the cake is being formed on both sides of the leaves.

Filtration is stopped when the desired cake thickness is obtained. The excess feed is drained off, and, as the level drops in the shell, the float of an automatic air regulator drops and opens an air valve. This admits air at a pressure of 3 or 4 lb. per sq. in., which holds the cake upon the leaves and forces the excess liquor more rapidly from the shell.

After draining, the wash liquid is forced into the shell following the same path as the filtrate. Excess wash is drained off, and if a dry-cake discharge is desired the drain valve is closed and an air blast is used.

For discharging, the air is turned off, the release valve is opened, the head is unlocked, the carriage rolled out, and a slight back blast of air causes the cake to discharge.

For wet discharge, the excess feed is drained out, the filter is opened, the carriage run out, and the cake sluiced off.

The Kelly filter competed with the intermittent vacuum filters in early cyanide-slime operations and later was adopted in beet-sugar manufacture. It is now widely used in petroleum refining, for handling lubricating oils, for pressure-still sludges, and for dewaxing operations, for which duties its design makes it especially suitable.

Fig. 10.—Sweetland filter (closed).

Sweetland Filter. The Sweetland filter (Fig. 10) consists of a series of circular filter disks suspended inside a cylindrical cast-iron shell transversely to its axis. The shell is divided along the horizontal center line into two halves, hinged together along the back. The upper half is rigidly fastened to supports, so that the lower half may swing open thus exposing the interior of the filter for cleaning. The lower half (Fig. 11) is counterweighted to facilitate opening and closing, and a special locking mechanism makes it possible to open or close the filter within a fraction of a minute. The edges of the two halves of the cast-iron shell are accurately machined and grooved to hold a composition gasket which forms a tight joint when the filter is closed.

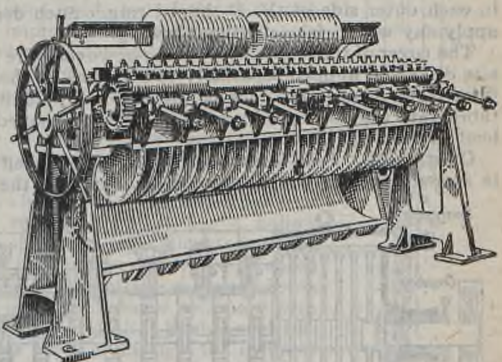


Fig. 11.—Sweetland filter (open).

A "boss" is cast along the top half of the filter body, and holes are drilled through to receive the filter-leaf outlet nipples. Each hole is counterbored on the inside to receive a filter-leaf rubber washer and, on the outside, to receive the cap nut and the lead washer.

Inside the upper half, leaf spacers are placed along the front and back sides of the filter to keep the leaves in alignment.

Special uses have been found for the Sweetland filter in many industrial chemical plants as well as in gasoline refining and in some phases of cane-sugar and beet-sugar refining.

Automatic sluicing mechanism is provided consisting of a manifold pipe passing through the entire length of the filter shell just above the leaves. Nozzles are fitted at equal intervals along this pipe so that one nozzle is directly behind each leaf.

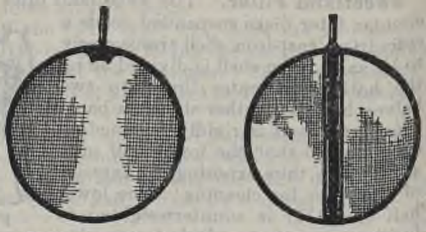
By an external rack-and-pinion drive the nozzles are oscillated through an arc of 110 deg. and are moved longitudinally while oscillating; thus, when water is supplied under pressure, one jet cleans both sides of a filter leaf.

Filter leaves are illustrated in Fig. 12. Each leaf is a circular piece of heavy screen bound at the edge with a U-shaped peripheral ring. This gives stiffness to the leaf and provides a smooth edge making it impossible for the screen to damage the filter-cloth bag covering the leaf. An outlet sleeve welded to the top of the leaf receives the nipple which holds the leaf in place in the filter and also carries away the filtrate.

For bottom drainage the outlet sleeve is welded into a flattened tube which extends all the way to the bottom of the leaf. When air blast is used to dry the cake, the air must force ahead of it all liquid left in the drainage screen, as the only outlet is at the bottom of the flattened tube.

Hence the liquid or wash water is displaced and dry-cake discharge secured.

An improved type of leaf construction has lately been devised, which gives a better means of cloth attachment. This is provided by a special peripheral member (in place of the conventional U-shape mentioned above), which has a dovetailed groove in each outer side of the U-shaped ring. Such design makes it possible to apply any woven fabric without sewing.



Top Drainage Leaf   Bottom Drainage Leaf

FIG. 12.—Sweetland filter leaves.

The cover of the leaf (either cotton or woven-wire fabric) is cut into proper-size disks and applied separately to each face of the filter leaf by placing the filter cloth against the face of the leaf and caulking the outer edge of the fabric into the dovetailed groove in rim of U-shaped, using cotton sash cord, lead wire, or the like.

Operation. Feed is supplied to the closed shell under pressure. Liquid is forced through the filter medium covering the leaves and issues from the individual outlets through sight glasses, the solids building upon both sides of the leaves in the form of a cake. When the rate of flow drops below the economic limit, feed is shut off and the excess is expelled by air under pressure sufficient to hold the filter cake in place. When the shell is cleared, washing is commenced by admitting water under pressure. After washing, discharge is effected by opening the drain valves and operating the sluicing mechanism. Water from the sluicing jets cleans the leaves and sluices out

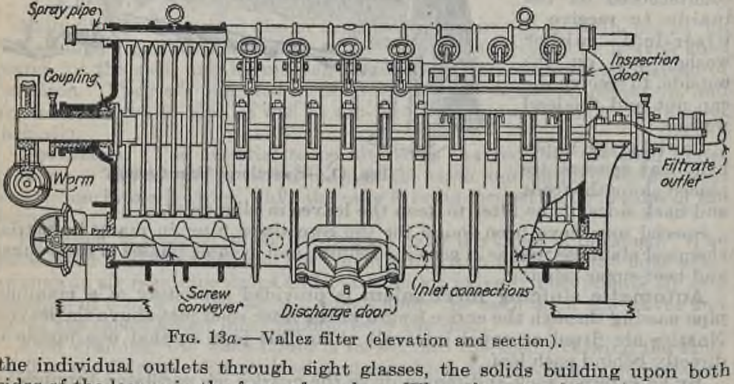


FIG. 13a.—Vallez filter (elevation and section).

the solids through the drains. If dry cake is required, washing is followed by an air blast, after which the lower half of the shell is swung open and a low air pressure turned into the leaves to discharge the cake which falls into a hopper or conveyor below.

Vallez Filter. In the Vallez filter (Fig. 13*a*), the leaves rotate inside a cylindrical pressure shell. The leaves are annular screen disks covered with filter medium and assembled in parallel at regular intervals upon a horizontal hollow shaft which rests in bearings at either end of the shell. This shaft serves as a filtrate channel and is rotated by worm-gear drive. The shell is divided into halves at the horizontal center line and the upper half is provided with inspection doors opposite the disks.

To replace a disk, it is necessary to unbolt the upper half of the shell and remove it with an overhead crane, and, as the disks are not sectored, the shaft is next raised and the disks removed until the defective one can be taken off in its turn.

Feed is forced in from a manifold pipe having openings into the bottom of the shell. For discharge, a sluicing pipe is located at the extreme inside top of the shell with holes drilled so that jets impinge at an angle on each side of every leaf. At the extreme bottom of the shell is a trough equipped with a revolving scroll to remove the sluiced solids through an opening in the bottom of the shell.

The Vallez filter was designed for certain phases of sugar-refining work, while a variant of the Vallez type has been used in oil-refinery operations.

Burt Filter. The Burt filter (Fig. 14) is a steel cylinder which is rotated like a cement kiln. It has a hollow trunnion at the feed end, and a tire and rollers toward the other end. It is revolved by a pinion drive, and its speed is varied according to the nature of the material being filtered. The cylinder is lined on its inner periphery with drainage panels covered by filter cloth. Each panel has one or more outlet nipples passing through the cylinder shell, and external stationary launders receive the filtrate dropping from each circle of nipples. The rear end is closed by a cast-iron door equipped with a quick-opening outlet for discharge.

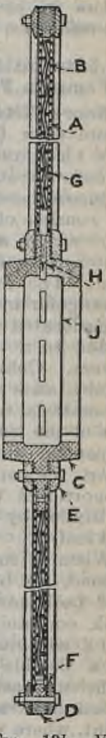


FIG. 13*b*.—Vallez filter (section of leaf).

Operation. The material to be filtered is fed through the hollow trunnion at the head end, as the filter revolves. When the required charge has been introduced, the feed inlet is closed, air is admitted under pressure, and this pressure is maintained in order to force filtrate through the filter while forming and holding the cake in place. During rotation the filter cake forms while the filter medium is submerged by the feed. The cake is homogeneous because of rotation. Any cracks which form are sealed when the cake reenters the feed. When the flow of filtrate ceases, air pressure is released and wash water is admitted. Rotation continues filling the cake with the water. Compressed air is then admitted which forces the wash through the cake. By using a muddy wash, all cracks or pit holes are sealed and a uniform cake washing secured. For discharge, the air pressure is cut off, water is admitted, the discharge ports are opened, and rotation is continued.

The Burt filter had a limited use in cyanidation some years ago, since which time it has been used to some extent in handling zinc sulfate liquors in electrolytic zinc refineries.

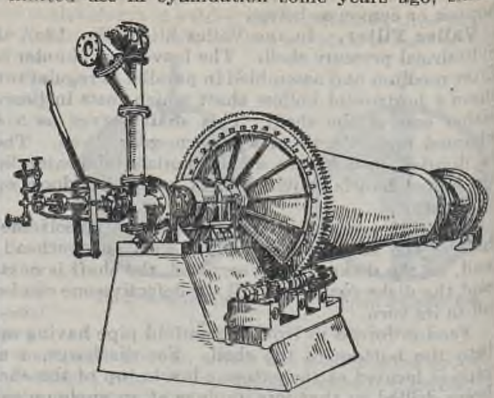


Fig. 14.—Burt filter.

Intermittent Vacuum Filters

Moore Filter. The Moore filter (Fig. 15) was the first to use vacuum filtration on a commercial scale. The leaf consists of a frame over which a bag is stretched to form the filter. The frame is rectangular and is made of perforated pipe connected to a vacuum system. Collapse of the bag is prevented and drainage is provided for, by wooden slats sewed vertically into the bag. When the leaf is submerged in the material to be filtered and vacuum is applied, the cake forms on the outside of the leaf, while the filtrate is drawn through it. A number of leaves suspended side by side constitute a *basket*, the capacity of which is directly proportional to the number of leaves multiplied by the area of the two sides of a leaf.

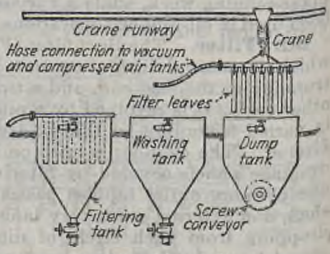


Fig. 15.—Moore filter.

When a sufficiently thick cake has formed, the basket is raised out of the feed tank and transported to another tank containing wash water, vacuum being continued meanwhile by means of a flexible hose connection. After sufficient washing, the basket is again raised and transported to the discharge point, where the vacuum is shut off and air pressure is applied. The pressure distends the bags and discharges the cake. The moisture that remained on the inside walls of the filter cloth is blown back and wets the outer side. It is the lubricating action of this water, coupled with the weight of cake and the flexing of the cloth, that causes the cake to slide off. As the leaves hang vertically from the frame, they are in the best position for a complete discharge to be made.

Handling the filter baskets with an overhead traveling crane reduces labor, one crane serving a number of baskets installed in the same group. The number and size of the leaves in each filter may be altered to give a convenient output per cycle.

The advantages of the Moore filter are simplicity of operation, ready inspection of leaves after cake discharge, and ease with which a new leaf may be installed in place of a defective one.

A disadvantage is that, during transfer from tank to tank, vibration may cause the cake to drop off. If the cake drops from a leaf, the washing is inefficient due to a short-circuiting of the wash liquid through the exposed portion of cloth. This requires strict control of the amount of fine sand contained in the feed which acts as a filtration aid. Also, if possible, cake cracking must be avoided.

Butters Filter. The Butters filter differs from the Moore filter in that the leaves are installed in the tank and remain stationary during the filtration cycle.

Feed is admitted until the leaves are submerged, after which vacuum is applied. When the required load of cake has formed, the remaining feed is pumped into a stock tank, the discharge valve is closed, and wash solution or wash water is run in until the leaves are again submerged. Vacuum is maintained in the meantime to prevent the cake from falling off. After washing, the solution or water is pumped out. The cake is discharged by opening the discharge valve of the tank, shutting off the vacuum, and applying air pressure, when the cake falls off and is sluiced away.

Small-scale operations require only one filter and one tank, whereas the Moore arrangement requires two or three tanks. For large-scale work, several tanks and baskets are needed in each method.

At times the wash solution has been pumped on to the surface of the charge in the filter while drawing off excess charge. While this caused some diffusion, the saving in time and the continued submersion of leaves were thought to justify the practice.

Both the Moore and the Butter filters have been widely used in filtering cyanide slimes, but their use has now been generally supplanted by continuous vacuum filters and they no longer have the major importance in modern filtering practice which they once had.

Continuous Vacuum Filters

Oliver Filter. The Oliver filter (Figs. 16*a*, 16*b*, 16*c*) consists essentially of a cylindrical drum supported in an open-top tank or vat and in such a manner as to allow rotation of the drum therein around its own axis which is in a horizontal plane. The position of the drum in the tank is such that its lower portion is confined within the tank walls, while the upper portion is exposed above.

The ends of the drum are either open spiders or closed heads which carry the two main trunnions by means of which the drum is supported. The drum shell is composed of a number of shallow compartments over which is secured a covering of filter cloth. The cloth is supported by a drainage grid and is held in place by a spiral winding of wire uniformly spaced.

Since the pulps handled on the Oliver filter differ widely in the percentage of liquid content and hence in the filtering rate also, the nature of the drainage grid is determined by the use of the filter.

For the simpler types of minerals and chemical products, the screen grid is $\frac{7}{8}$ in. deep, handling moderate amounts of filtrate. If built for various free-filtering material (sulfite pulp as an example), the filtrate passages must be ample and drainage grids are therefore deeper, *i.e.*, $1\frac{1}{2}$ to $1\frac{3}{4}$ in.

The opposite condition is found in the very shallow grids in the cane-mud filters. These filters handle smaller flows and also separate the initial cloudy filtrate; hence, sharp separation with the least dilution occurs with the shallow grid (less than $\frac{1}{8}$ in. deep).

The screen support for the filter medium is a specially milled cedar grille for most neutral or acid filtrates, but, with caustic filtrate, perforated steel sheet or cast iron is used. The latter is also used for salt solutions. Concentrated acids require cast-lead grids.

The interior of each compartment communicates through a separate conduit (28), to a valve mechanism (31) which, during operation, automatically applies either suction or positive air pressure to the several conduits in rotation and through them in turn to the interior of the compartments. The automatic valve (31) is connected to a vacuum system and to a source of compressed air.

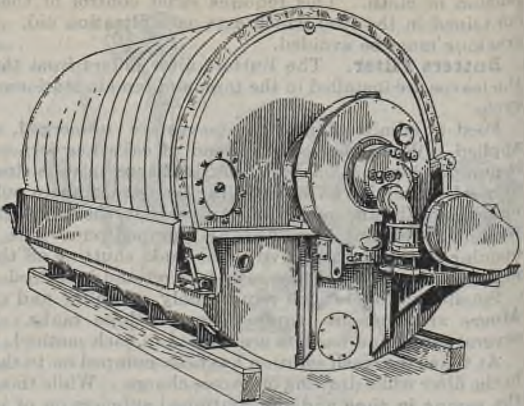


FIG. 16a.—Oliver filter, discharge side.

As the automatic valve is an indispensable control, it has been specially developed in the Oliver filter. The earliest designs of automatic valves for Oliver filters made provision for separation of wash solution from filtrate. This was made by placing a bridge or stop in the valve interposing a barrier between the filtrate outlet in the lower half of the valve and the wash solution coming from the upper half. In many cases no separation of initial filtrate from wash solution is required. For such cases and for simple dewatering operations no bridge in the valve is required, and only one outlet from the valve is used.

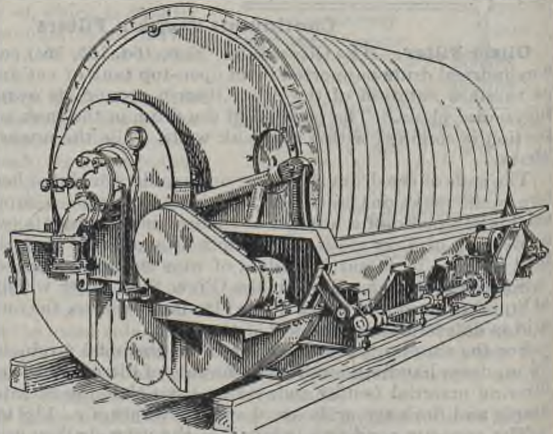


FIG. 16b.—Oliver filter, agitator side.

Two outlets are usually enough, but in some cases a "cloudy filtrate" port or outlet is provided when using "open," or wire-mesh, filter covers, which diverts a cloudy filtrate produced in some uses of the filter just as the cake begins to form. This small amount of filtrate is returned to the filter feed and thus the remaining filtrate is kept quite clear.

The usual bridges or stops in the valve are set before starting operation of the filter. In some cases they are arranged to be set by a handwheel on the outside of the filter valve, the handwheel allowing exact adjustment without removal of the valve for setting the bridges. This is a patented feature.

Other valve bridges, also externally controllable, permit modifying the amount of vacuum without altering the peripheral location of the bridges.

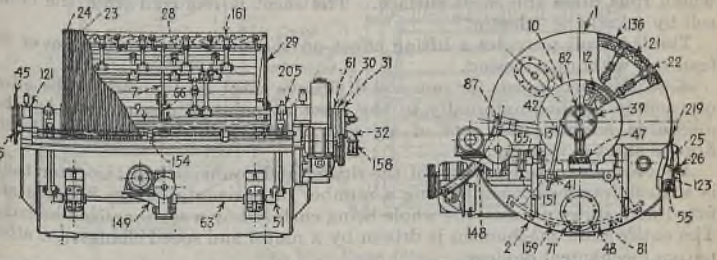


FIG. 16c.—Oliver filter with concentric-type agitator with independent drives for filter drum and agitator.

PARTS LIST

1. Filter drum
2. Filter tank
7. Drum arms
9. Drum shaft
10. Housed bearing
12. Worm-drive gear
13. Worm shaft
21. Wood staves
22. Division strips
23. Filter medium
24. Wire winding
25. Scraper blade
26. Scraper adjuster
28. Drum piping
29. Closed drum head
30. Wear plate
31. Automatic valve
32. Vacuum connection
39. Felt washer
41. Valve-adjusting pivot
42. Valve-adjusting rod
45. Wiring sprocket
47. Worm
48. Agitator rakes
51. Agitator crank
55. Scraper bearing
61. Pipe plate
63. Crank shaft
66. Center spider
71. Handhole cover
81. Agitator arc
82. Automatic valve flange
87. Connecting-rod pin
121. Rear bearing
123. Scraper plate
125. Feed-screw sprocket
136. Filter cover support
148. Type A drum drive
149. Type A agitator drive
151. Shaft coupling link
154. Wiring center dolly box
155. Shaft coupling
158. Diaphragm vacuum connections
159. Saddle clips
161. Drum nipples
205. Oscillating spider
219. Scraper tip

For high rates of filtrate flow the vacuum passages or conduits leading to the valve from the surface of the filter drum are made very capacious to avoid frictional losses. In extreme cases the passages convert the entire interior of the filter drum into manifold wedge-shaped compartments, which take the place of the usual tubular conduits.

Under the drum, barely clearing the bottom of the tank is suspended a framework (81) supporting horizontal rakes (48) which, during operation, slowly oscillate thus agitating the feed.

The filter cake is usually discharged from the drum surface by a scraper blade which is set in a vertical position, and in the present design the blade does not touch the wire winding of the drum. On the edge of the scraper is affixed a detachable rubber tip. The scraper itself is mounted upon the edge of the filter tank. The low-discharge point of the Oliver filter cake permits cake discharge by pressure reversal only, in most cases, the scraper serving as a diversion plate only.

There are important exceptions to this among which are the various fibrous materials formed in paper making. For these fibrous sheets, so called, vacuum alone does not reduce the moisture content sufficiently low in some cases, nor does the scraper always give the best discharge.

By applying pressure upon the wet sheet of fiber with heavy steel or cast-iron rolls, much additional moisture is expressed, and the same sheet is suitably removed from the drum of the Oliver filter by couch rolls or lead rolls in contact with the sheet; the effect is by adhesion to the couch roll which rolls upon the sheet surface. The sheet is removed from the couch roll by a knife or "doctor."

The lead roll provides a lifting effect on the sheet which passes over the lead roll without adhesion.

A widely used means for removal of fibrous sheet is obtained by the effect of vacuum applied externally to the sheet at point of discharge. Another discharge method uses jets of air and/or water introduced beneath the fibrous sheet externally.

Above the exposed portion of the drum, and connected to the filter tank, is a steel framework supporting a number of horizontal water headers and fitted with spray nozzles, the whole being enclosed by a sectionalized housing. The entire filter mechanism is driven by a motor and speed changer, or alternative mechanical devices.

The simplest type of drive for the Oliver filter is by a belt and pulley.

Various mechanical drives are possible for the Oliver filter. The one most favored now is a combination of motor directly connected to the worm-gear speed reducer and thence by V-belts and sheaves to the worm shaft that drives the worm gear on the filter drum. The motor and speed reducer are mounted on the filter tank. Roller-chain and sprocket drives are less favored, though many installations still use them. Speed variations can be made in various ways such as changing V-belt sheaves or sprockets. Others may use multispeed motors, or cone-pulley devices like the Reeves or the Link-Belt drives. Where d.c. current is available, rheostat speed control is possible.

The original duty of the Oliver filter was the filtration, at air temperatures, of mineral slime from water or from a moderately alkaline cyanide solution. This required a simple design of steel, cast iron, wood, and cotton material.

Soon its expanding field of use required a design modified to handle higher temperatures with acid or alkaline solutions. The results were such that the Oliver filter now uses, besides the original type of steel, wood, cast iron, and cotton, filters of all wood, others of all steel, stainless steel, or monel metal.

Certain duties require an all cast-lead filter, and there are also all cast-iron filters. Lead-lined steel filter tanks are built and in some cases a sheathing of brass or copper is used over the wood and steel portion. Any practical material of construction may be used in building the Oliver filter, and filter covers are provided in a wide variety of textiles, woven-wire cloth, as well as finely perforated metal sheets.

Operation. During operation, the drum rotates slowly while the tank is supplied with the material to be filtered and the level is maintained to ensure a constant depth of submergence of the lower portion of the filter drum. In some types this depth may be set between limits ranging from zero to almost complete submersion of the drum. Once chosen, the valve is set for the given conditions

Through the action of the automatic valve, vacuum is applied to those compartments of the drum passing through the sludge. The vacuum created within the compartments causes a flow of filtrate through the filter medium, conduits, and automatic valve, and a layer of cake solids is deposited upon the filter medium covering the submerged portion of the drum.

As the drum revolves, the vacuum in the compartments is maintained, and the layer of cake solids emerges and passes through the arc included by the upper or exposed portion of the drum. It is subjected to washing by water from the spray nozzles; the wash water permeates the cake and displaces the liquid contained.

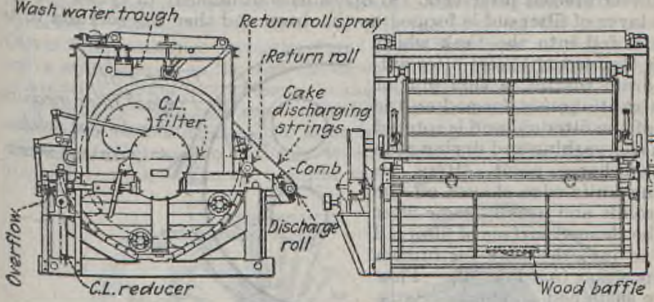


FIG. 17.—Feinc filter.

During this washing operation, the replaced liquid, together with some of the wash water, flows through the filter medium and conduits and is discharged from the automatic valve in the same manner as the liquid from the cake-forming operation. Wash liquor may be discharged separately from the original filtrate at the automatic valve.

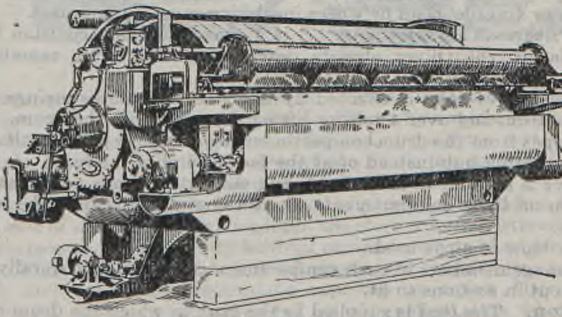


FIG. 18.—The Oliver precoat filter.

Following the washing period, as each successive compartment reaches the scraper, the vacuum is cut off by the action of the automatic valve, and air at low pressure is applied. This operation, with or without the action of the scraper, effects the discharge of the cake, after which the cleaned filter surface again rotates into the tank and the cycle outlined above is repeated.

Oliver Continuous Precoat Filter. This filter is a major modification of the standard Oliver continuous drum-type filter. Its unique principle of filtration and method of cake discharge have opened up many opportunities for the continuous filtration or clarification of products that have hitherto been difficult to handle. Precoat filtration is particularly adapted to handling solutions with pasty, gummy, or colloidal substances, or solutions with small amounts of solids held in suspension.

The Oliver precoat filter (Fig. 18) operates continuously in cycles. First, a heavy layer of filter aid is formed on the drum and then the solution to be clarified is fed into the tank which has been drained of excess filter aid. As the drum rotates, a thin film of solids is continuously formed on the surface of the filter aid and is rotated through the washing and drying zone to the discharge point. Here, an advancing knife-edge shaves off the film of solids and usually some filter aid. The cleaned surface of filter aid rotates on into the tank for further cake deposition (see Fig. 19). Flow rates are sustained and high. Satisfactory clarification usually takes place in one step. Precoating takes less than an hour; filtering or clarifying continues for periods ranging from 16 hr. to a week, depending upon how much precoat is removed with the cake.

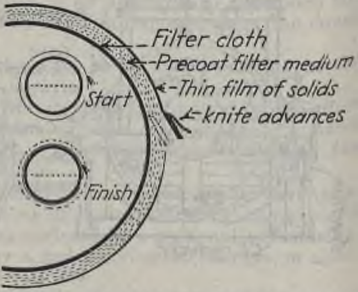


FIG. 19.—Operating method of Oliver precoat filter.

The Oliver precoat filter is made for both continuous-vacuum or continuous-pressure operation. Enclosed units are made for handling products giving off noxious or volatile gases or where insulation must be provided.

Feinc Filter. This filter (Fig. 17) is a continuous, rotary-drum vacuum filter similar to the Oliver already described but differing essentially as follows:

The discharge of cake is effected by a system of endless strings passing around the drum and over a roll for lifting the cake from the drum.

Outlet pipes from the drum compartments terminate in ports on the periphery of the rotating hub instead of at the face or end of the hub.

The valve is annular in shape enclosing the hub and has ports that break the vacuum on those compartments where the strings lift the cake from the filter drum.

No back blow of air is used.

The drainage members in each compartment are formed of spirally wound wire mats cut in sections to fit.

Operation. The feed is supplied to the tank in which the drum revolves and vacuum is applied, causing cake to form and to embed the endless strings on the submerged sections.

As the drum revolves, the cake emerges and travels around with the drum to the point of discharge, which is normally on the descending side. Here the vacuum is broken by the automatic valve and the strings leave, lifting the cake from the drum. They pass over a discharge roll, as shown in Fig. 17, the flexure causing the cake to fall from the strings. As the strings return to the drum they pass through a comb to keep them in alignment and to remove any adhering cake.

Dorrco Filter (Fig. 20). This is a vacuum filter of the rotary-drum type with the filter medium placed on the inner surface of the drum as a series of panels parallel to the drum axis. The drum serves also as the container for the pulp and no supplementary tank is used. Except for the annular retaining ring which creates the bath, the drum is open at one end for convenient inspection.

The drum is supported by a tire and riding rolls at each end. It is driven by a motor and speed reduction connected to the riding-roll shaft by chain and sprocket. The automatic filter valve is the same as the one used on the Oliver filter, although formerly it was of annular type. Bridges used in this valve are the same as those in the Oliver filter, but additional bridges permit alternating suction and pressure in the cake-discharge section.

A troughed belt conveyor runs through the machine at one side of the center line collecting the cake as it is discharged and delivering it through the end of the machine at the center-line elevation. This conveyor is driven through the opening in the annular valve, and the drive arrangement is such that the speed of the filter drum can be adjusted independently of the speed of the conveyor. In some cases cake is removed by a spiral scroll in a trough.

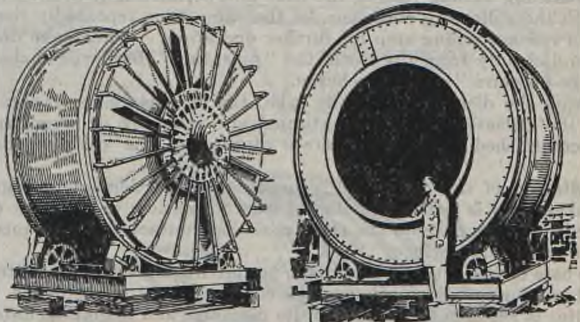


FIG. 20.—Dorrco filter.

The feed is arranged to enter at either end of the drum and is distributed uniformly on a line, the full length of the drum, by means of an inclined riffled launder or some other simple feed distributor.

The inner face of the drum is composed of a series of individual filter panels. In the case of a fabric filtering medium, the cloth is loosely stretched over the panel surface and conveniently held by packing rope on the four sides of the panel. This packing rope is pressed down into narrow grooves leaving a smooth and practically continuous exposure of active filtering surface. These panel coverings are supported on a punched plate which is rigidly spaced away from the outer shell to allow passage for air and filtrate, and the filtering medium is in turn held free from the punched plate by an intermediate layer of backing cloth. The cloth can be applied as a one-piece blanket for the filter, or, by simply cutting the cloth in rectangular strips, the individual panels may be separately reclothed. The panel compartments communicate through the end of the drum with externally located piping, leading to individual ports in the replaceable rotating wearing plate of the automatic head valve, so that the vacuum, pressure, steam, etc., can be applied in proper sequence during the cycle.

Operation. As the filter revolves, the cloth passes down into the pulp bath underneath the line of feed and the cake-forming vacuum is applied automatically as adjusted by the setting of bridges in the head valve, permitting, where desired, the precoating of the cloth with the coarsest, most rapidly segregating material in the feed. As the cloth emerges from the bath, the cake is drained out and washing sprays may be brought to bear upon it while it is still resting in an inclined position against the drum. Following this early application of washing sprays, the cake is dried by vacuum until discharge.

As the cake passes over the hopper which guards the discharge conveyor, the drying vacuum is cut off and the panel is brought into direct connection with the special discharge port. This port leads by separate piping from the filter valve to a small four-way valve bringing the port into direct connection with the inlet and outlet of a small blower, and at the same time alternately opening the outlet and inlet connections of the blower to the atmosphere. This imparts to the panel an alternating pulsation with a gentle breathing action, freeing the cake and permitting it to drop away by gravity to the collecting conveyor, at the same time repeatedly flexing the cloth as a reconditioning step for further operation. When cake discharges readily and cleanly from the cloth, the "pulsating" discharge is changed to simple low-pressure "blow" discharge.

Following the discharging of the cake, the cake cloth is exposed and, if desired, the pulsation may be continued and spraying or steaming of the cloth accomplished with the separate division of the valve available for this use.

No agitation of the pulp bath is used as any settlement or segregation which takes place is in the direction of cake formation, and neither the pulp in the bath nor the cake on the cloth comes in contact with any mechanical device or stationary surface.

In the case of free-filtering materials, permitting high speeds of filter operation and handling of large tonnages on comparatively small drums, the face of the filter is short in proportion to its diameter and a simple chute is substituted for the belt conveyor. The short-faced filter lends itself particularly well to the distribution of extremely coarse heterogeneous feeds where any appreciable depth of bath is difficult to maintain, the coarse material being merely sluiced on to the narrow path of filtering surface revolving under it. When operating conditions prevent the use of cotton filter cloth, a special type Dorrco filter has been designed to use woven-wire cloth. The separate panels in this design are inwardly convex, instead of uniformly concave like those in the standard Dorrco. These panels have special clamping bars and plates to hold the wire cloth in place.

Oliver Top-feed Filter. On this filter (Fig. 21) the feed is applied on the ascending face of the drum and a chute receives the cake when discharged. The end flanges extend radially beyond the face of the drum in order to retain the feed. These departures from ordinary rotary vacuum-filter construction are shown by the illustration. The cake is carried around the filter nearly to the point of cake formation before discharging, or about 315 deg. travel on the circumference.

In cases where minimum moisture concentrations are needed, a hood is provided in which the incoming air is heated before it is drawn through the cake thus obtaining a dry product by direct evaporation with high thermal efficiency.

Large volumes of air are drawn through the cake at low vacuum (2 to 4 in. Hg) and the power consumption is reduced to a minimum by designing the filter with the least possible internal resistance—no piping being used in the vacuum conduits of the filter.

Since very low moisture concentration in a filtered product is usually required, the design of the Oliver top-feed filters has evolved methods for the application of heat to the filter cake in various convenient ways.

Where indirect heat at lower temperature is needed, the air drawn through the filter cake is first passed through a heat exchanger built on the multitubular plan, using steam.

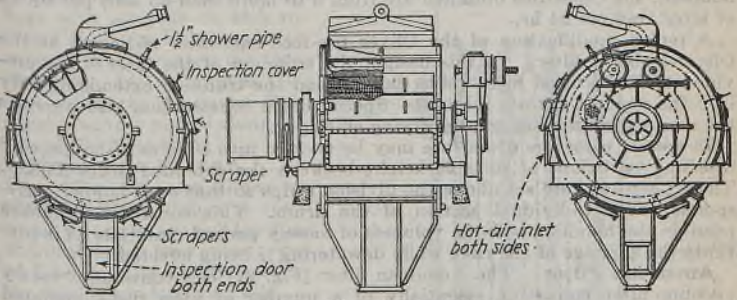


Fig. 21.—Oliver top-feed filter.

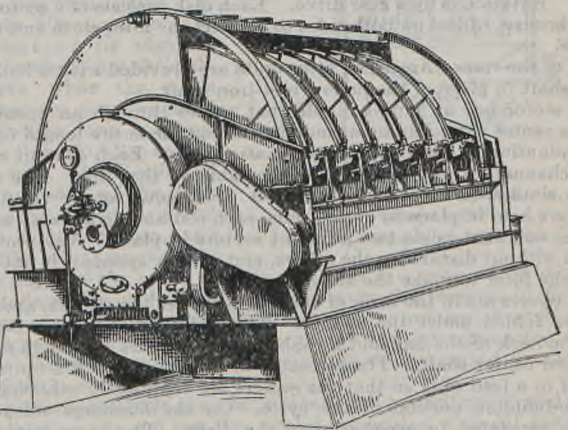


Fig. 22.—American filter.

Higher temperatures may be obtained through use of direct steam or of heaters supplied with oil, gas, or coal fuel, which provide direct use of gases of combustion. "Bone dry" salt crystals are readily produced on the Oliver top-feed filter. The dewatering of crystal magma, such as salt crystals in brine, and other crystalline chemical products are a special field of the Oliver top-feed filter.

Other similar mineral products, such as metallurgical table concentrates and phosphate sands are successfully handled by it also. Its use includes handling such uncommon substances as Cellophane-waste product, which occurs in tiny micaceous flakes.

A special economic advantage of the Oliver top-feed salt filter is obtained by using vacuum exhausters that are direct-driven from steam turbines, the exhaust from which is used in vacuum-pan evaporators.

The tonnage that can be handled on the Oliver top-feed filter of a given area depends chiefly upon the fineness of the material and the final degree of moisture content required. On materials for which it is particularly adapted, the capacities obtained are from 5 to more than 35 tons per sq. ft. of filter area per 24 hr.

A recent modification of the Oliver top-feed salt filter is known as the Oliver-Robison filter. In this design, the trunnion of the filter is not provided with the usual filter valve, but instead the trunnion extends directly into the vacuum filtrate receiver. Special drum construction also provides the means of obtaining greater drying efficiency.

In special cases the drum face may be divided into a series of hoppers by inserting partitions of suitable height between the extended drum flanges. These partitions are set above the division strips so that each hopper corresponds to an individual section of the drum. This construction makes possible the handling of large volumes of loosely packed materials as it prevents the slippage of the cake while dewatering is being effected.

American Filter. The American filter (Fig. 22) is a continuous rotary vacuum filter consisting essentially of a number of filter disks mounted at regular intervals around a hollow cast-iron center shaft, as illustrated in Fig. 22. Rotation is by a gear drive. Each disk consists of a sector of wood, iron, or bronze, ribbed on both sides to support the filter cloth and to provide drainage.

Some of the recent American filter units are provided with a built-up steel central shaft in place of the hollow cast-iron shaft.

Each sector has an outlet nipple that passes through an opening in the cast-iron center shaft joining a conduit running the entire length of the shaft and terminating in a port at the automatic valve. Each conduit serves as a filtrate channel for all sectors along the shaft on that line. The automatic valve is similar to those used for other continuous rotary vacuum filters. Sectors are held in place by radial rods, each rod having a clamp and nut on the outer end that holds two adjacent sectors in place. Any sector can be replaced without disturbing the others, and at slow speeds it is not necessary to stop the filter to make the change.

Filter covers are in the form of bags slipped over the sectors, and the outer edges are folded under the clamps. At the filtrate nipple, a cord is tied round the neck of the bag and a rubber washer makes a tight joint between nipple and center shaft. The assembly of filter disks on the center shaft is mounted in a feed tank so that the sectors are completely submerged during the cake-building portion of the cycle. On the discharge side, the filter tank is crenelated to accommodate the disks. The space between these divisions is utilized for cake discharge; scrapers or tapered discharge rolls for each disk are mounted at the top of the tank. In some cases discharge is effected by fine water jets under pressure.

Operation. During operation, the feed is supplied at the bottom of the tank through a manifold pipe having one supply nozzle under each disk. A homogeneous mixture is maintained by forcing a steady stream of feed through these nozzles, the excess pulp returning to the supply tank through an overflow in the filter tank. The disks rotate slowly, and, as soon as the sectors are submerged, vacuum is applied by the action of the automatic valve.

A layer of cake solids forms upon the cloth on both sides of the sectors and the filtrate passes from the sector through the conduit in the center shaft and out through the automatic valve. Vacuum is still maintained when the sectors emerge and are exposed to the air, and wash is applied if required. As each sector reaches the scraper, or discharge, roll, vacuum is cut off and a gentle air blast is applied. This causes the filter bag to inflate as it is not held fast to the sector by any grid or wiring. Contact of the bags with scrapers or with rotating discharge rolls causes the cake to drop between the tank divisions. In some uses, the feed enters the American filter from a launder placed along the rim of the filter tank.

Streamline Filter. This filter (Fig. 23), also known as the **edge filter,** differs in operating principle as well as mechanical design from other types of filters and is widely used in renovation of insulating oils.

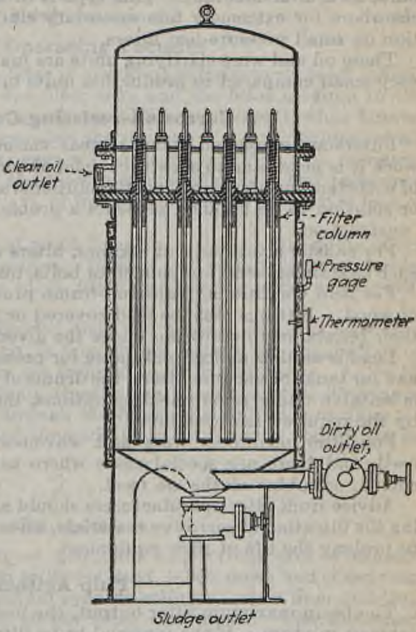


FIG. 23.—Streamline filter (section).

Closely compressed disks of specially prepared paper are used as the filter medium, the filtrate passing by edgewise filtration between, not through, these disks, while the solids collect upon the outer edges of the disks which form a hollow column or "pack" when assembled in a unit. Application of vacuum to inside of the pack draws the oil through it, all traces of sludge, carbon, and other solids being retained on the outer edges of the paper disks in the form of a cake. Discharge of this cake is made at the end of the day's operation by reversal of flow, using compressed air. The **streamline filter** uses a number of such elements ("packs") mounted together in a container and interconnected for joint operation. In certain cases the streamline filter is arranged to operate as a pressure filter instead of by vacuum.

Wide applications of this filter have been found in purification and dehydration of insulating oils and the like. Solid impurities, colloidal or otherwise, are removed by passing through the streamline filter, and, by the dehydration effected, the oil regains the standard dielectric strength for transformer and circuit-breaker oils. Dehydration occurs through the joint effect of heating the oil followed by subjecting it to vacuum.

The streamline filter has been extensively adopted in England and abroad for such uses.

Mention should be made of the various small individual filter units mounted on internal-combustion engines for clarifying continuously the crankcase oil. These are usually provided with a cotton-flannel tubular labyrinth into which the dirty oil is pumped. The impurities are retained within the cotton tube, while clarified oil passing through the fabric walls of the tube is returned to the crankcase.

Another small type of filter used in commercial production is the pressure unit, which depends upon specially prepared asbestos material, in pads or in bulk, as a filter medium. This type is often used in the wine industry and elsewhere for extremely fine secondary clarification following primary filtration on small pressure-leaf filters.

These oil and wine clarifying units are made in many forms, but are always very small compared to production units in other industrial uses.

Corrosion-resisting Construction*

Filters are employed under a great variety of conditions, and in chemical work it is necessary to construct them of corrosion-resisting materials capable of withstanding attack by caustic solutions, alkalies, or acids. Each substance or solution to be handled presents a problem calling for special materials of construction.

For caustic solutions and alkalies, filters are built of cast iron, using alloys such as stainless steel and monel for bolts, nuts, fittings, and the filter medium.

For acid conditions, plate-and-frame presses and vacuum filters are made of wood. Fittings may be lead covered or made of special alloys chosen for their resistance to corrosion under the given conditions.

Lead is used as an internal lining for pressure filters of the closed-shell type and for tanks of vacuum filters, the drums of which may be cast lead. Rubber is suitable under severe acid conditions, but it is limited in its applications by the required temperature.

For filter mediums, wool and woven-wire monel cloth are extensively used, and there are special cases where asbestos fiber, fiber glass, nitrated cotton, or rubber cloths are used.

Advice from filter manufacturers should always be obtained when considering the filtration of corrosive materials, since they constantly conduct research to prolong the life of their equipment.

Pulp Agitation

To obtain maximum filter output, the feed must be maintained as a homogeneous mixture. If the material to be filtered contains particles varying in size and density, segregation takes place unless there is sufficient agitation.

In shell-type pressure filters, segregation is prevented by maintaining a circulation of excess feed through the filter. When the proportion of solids in the feed is small, this is essential for the formation of even cakes.

The rotating-disk vacuum filter (American) utilizes feed circulation to provide agitation by forcing the feed through inlets in the bottom of the tank directly under each disk, the overflow returning to the feed-supply tank. This gives excellent results with materials of high specific gravity and of uneven size.

In rotary-drum vacuum filters (Oliver), agitators in the form of longitudinal rakes oscillating under the drum in the tank are most satisfactory.

For materials that dewater quickly causing thickening of the feed in the filter tank, the oscillating agitator may have hollow rakes through which the returned filtrate is forced, thereby preventing segregation. The amount of filtrate used for this purpose is adjusted to maintain a balance between solids and liquids to give the required density of filter feed.

* See pp. 2091–2166.

Vacuum leaf filters of the Moore and the Butters types use air lifts at the sides of the tank between the leaves as a means of circulating the feed and preventing settlement; or a centrifugal pump may be used.

In the Dorrco filter, agitators are never used and segregation is encouraged as an aid to filtration. This is a result of applying the filter medium on the interior of a drum acting as its own pulp container.

Range in Operating Vacuum

The vacuum registered when a filter is operating satisfactorily is an indication of the resistance offered by the filter cake and the filter medium to the passage of air. For instance, a crystalline solid like grainer salt, when filtered from the mother liquor of brine, requires the passage of a large volume of air through the cake, yet the vacuum gage will indicate only 2 to 4 in. mercury. In this case, the thick cake and the filter medium offer very low resistance to the passage of as much as 40 cu. ft. of free air per minute per square foot of filter area.

Precipitated calcium sulfate is of much finer grain size and therefore offers a correspondingly increased resistance. It does not require large volumes, usually 1 to 3 cu. ft. per min. per sq. ft. filter area but the vacuum gage will indicate 10 to 14 in. mercury.

Cane-sugar mud is so finely divided that the passage of air is restricted to very small volumes, about ⅓ cu. ft. free air per minute per square foot of filter area. The vacuum readings are from 25 to 27 in. mercury.

Vacuum readings serve to indicate the friction encountered in drawing a given amount of filtrate and air through the filter cake and filter medium.

The range of vacuum employed depends upon barometric conditions at the location of the equipment.

Temperature.
Character of material being handled.
Size of vacuum pump in relation to filter equipment.

Cyanide Pulp. The recovery of gold and silver from ores is usually effected by grinding the crude ore to 90 per cent −200 mesh and dissolving its precious-metal content with sodium cyanide solution. Still finer grinding is now done in some cases up to 90 per cent −325 mesh. The thickened pulp resulting from this operation is usually filtered and washed on continuous vacuum filters for a thorough recovery of gold and silver.

Although the ore is usually quartz, there are often enough clay and other semiplastic material present to reduce the porosity of the mass when filtered. Therefore, the capacity of such filters varies directly with the proportion of clean quartz in the pulp and, in some cases, capacities of 2000 lb. and upward dry weight per square foot of filter area have been handled on continuous filters in cyanide-slime plants when treating clean quartz ores.

The oxidized and clayey ores will lower such capacities to 450 to 600 lb. per sq. ft. of filter area per 24 hr.

Cyanide-slime filtration requires both filtration and thorough washing of the filter cake before discharge to remove the dissolved gold and silver, and for the latter either, or both, sprays or drip-wash equipment are provided. Wash equipment must be effective and subject to observation, as the character of the washing is revealed by the appearance of the cake surface during the operation.

The pulp fed to cyanide filters is usually 40 to 50 per cent solids, while discharged-cake moistures vary with the nature of the crude ore, ranging in most cases from 15 to 25 per cent.

As the cyanide solutions handled are alkaline with lime, the filter cloths become gradually clogged with calcium carbonate, and this requires at intervals the use of hydrochloric acid-solution wash to remove the coating on the cloth and to maintain its porosity. Cotton fiber covers are universally used for this work, while the filters are built of steel, cast iron, and wood. No brass is used as it is attacked by cyanide solution.

Metallurgical Flotation Concentrates. These products are obtained by the flotation of ore ground to a fineness varying from 65 per cent passing 200 mesh to as much as 90 per cent passing 350 mesh. Usually over 50 per cent of the particles composing the feed are metallic sulfides such as galena (PbS), zinc blende or sphalerite (ZnS), chalcopyrite ($CuFeS_2$), of high specific gravity 7.5, 4.1, and 4.2, respectively. The capacity of the filter will be determined by the fineness of the particles, their specific gravity, the density of the feed, and the amount of gangue or non-metallic material present in the form of slime.

Capacities of 300 lb. per sq. ft. per day are obtained upon material containing considerable gangue and slime; 1400 lb. or more is obtained on the same material, even with finer grinding, provided a clean concentrate is made.

The object of filtration in this case is simply to dewater the concentrate preparatory to smelting, and cake moistures run from 7.5 to 16 per cent. Pressure filters are unsuitable because the proportion of solids in the feed is high, necessitating frequent opening and discharging and thus making operation and maintenance charges exorbitant.

Rotary vacuum filters, with their continuous discharge of cake, handle such material with minimum of attention and low operating and maintenance charges.

Metallurgical Gravity Concentrates and Sands. These products are obtained by water classification of ores in which a part of the mineral particles is liberated without fine grinding. Screen analyses of the feed selected at random show 10 to 15 per cent held on 20 mesh and 34 to 72 per cent held on 65 mesh.

With such coarse feed it is difficult for a rotary vacuum filter to pick up and hold a cake against the action of gravity when the position of the filter medium becomes such that the undrained cake is on the under side and liable to fall off by reason of its weight. Suitable types of vacuum filters give high capacity ranging from 5 to 35 tons per sq. ft. filter area per day when handling such feed. The Oliver top-feed filter is best suited for this use.

Cane-sugar Syrups. For the clarification of liquor and syrups in sugar refineries, pressure filters are especially suitable and, conversely, vacuum filters are unsuitable. This is because the proportion of solids in suspension is so low as to require a filter aid to enable clarification to be effected. The proportion of filter aid added is about three times the weight of solid to be removed and this makes a reasonable thickness of cake. Plate-and-frame presses and shell-type pressure filters are used.

Paper Pulp. This material, because of its fibrous nature, is free filtering and is readily handled on open-mesh wire screen. It is bulky being of low specific gravity, and the quantity to be handled by any unit is large. At the same time, large volumes of water or solution must be removed and therefore the only filters suitable are vacuum filters with continuous discharge. There are many stages in the manufacture of pulp and paper at which filtration is performed such as in washing the stock after cooking in the digesters, deckering after screening, thickening prior to bleaching, washing after bleaching, and to recover fiber from paper-machine waste white water.

The output obtained varies with the nature of the work and is from 200 to 1200 lb. per sq. ft. filter area per day, and the water, or solution filtered, will vary from 1½ to 20 gal. per sq. ft. per min.

Sewage Sludge. All types of sewage sludges produced at municipal sewage-treatment plants such as raw, activated, digested, and various mixtures are now being dewatered successfully by means of vacuum filters. Dewatering in this manner is impractical without sludge conditioning, and some sludges are difficult to dewater even after treatment. The suspended particles are finely divided and are compressible and distorted by pressure to such an extent that ⅛ to ½ in. is the usual limit of cake thickness.

Vacuum filters are particularly suitable for handling this material since they accomplish dewatering at a relatively low pressure: 21 in. mercury. Under this pressure (10½ lb. per sq. in.), there is least distortion of the compressible feed particles. A fair filtrate is obtained, and the cakes are automatically discharged.

Filter feed ranging from 1 to 10 per cent cake-forming solids gives a capacity of 25 to 250 lb. dry weight of cake per square foot of filter area per 24-hr. day. Cake moistures are 65 to 85 per cent, depending on the type and character of the sludge handled. Raw sludge cake is usually incinerated; activated is incinerated or dried for fertilizer; and digested is incinerated or spread on land for soil conditioning.

Cement Slurry. In portland-cement manufacture, the raw material is ground either with or without addition of water and the wet-ground material, known as slurry, is the type of feed used in most cement plants today. Whether a wet or a dry mixture is used, it must then be calcined and burned in a rotary kiln, the water content in the wet mix requiring the use of fuel to evaporate it during the process. To avoid this use of fuel, filtration of the wet slurry before burning has been widely adopted.

The ground material in wet cement slurry usually is chiefly limestone with some shale or clay for the alumina constituent, but there are also plants operating on marl, which is a natural mixture, and on blast-furnace slag and limestone. In all these cases, filter plants have been successfully used to reduce the water in the feed to the cement kilns. The wet-ground material usually enters the filters with 30 to 50 per cent water content, and the filter cake discharged from the filter will range from 17 to 25 per cent moisture. The moisture content in the feed to the filters is usually kept as low as is compatible with the easy handling of the pulp in the pumps.

The filtration of such pulp on vacuum filters shows wide ranges in capacity, as do individual plants. This is caused by variations in raw material from the quarry, so that cement-filter installations should be calculated with prudent capacity factors. Usual filtering rates are 400 to 1000 lb. dry weight per square foot of filter area per 24 hr.; under exceptional circumstances, rates are considerably higher.

Besides the fuel economy noted, the use of filters enables a given size of cement kiln to burn more clinker than before and thus to increase the capacity without an increase of kiln installation. This provides a reduction in cost per unit of product and is a valuable factor in the economy of cement-plant operation.

In many ways the filtration of cement slurry resembles cyanide-pulp filtration, as both are finely ground mineral pulps in aqueous suspension, and the filter cakes produced are quite similar. The density of cement slurries prevents segregation of solids in the suspension, which may occur in more dilute mixtures.

Salt and Crystals. This class of material is being separated from accompanying solution and dried with success on continuous vacuum filters.

Formerly, this work was done by centrifuges of various types, but the continuous vacuum filter has now been widely adopted for this service.

The materials handled under this heading may be any practically slime-free aggregate of non-deformable solids whose particle size is usually in a narrow range which assures uniform cake formation. There are cases, however, where heterogeneous masses with wide range of sizes are successfully filtered on salt-type filters.

This plan has been adapted in practice for sandy pulps and for many crystalline precipitates in various solutions.

The ratio of solids to liquids in the filter feed does not affect the filter performance greatly, as is the case with plastic pulps; and the moisture in the discharged cake is always low.

Capacities are usually high: in the case of sodium chloride, 4 to 6 tons per sq. ft. filter area per 24 hr. being capacities often reached. In special cases much higher capacities have been reached in regular work.

The liquid in the cake is removed by the use of vacuum and heated air drawn through the cake while on the filter. The salt is therefore readily discharged at 2 to 2.5 per cent water content and, by special equipment, can be discharged from the filter below 1 per cent water; "bone-dry" salt is now being produced on Oliver-Robison top-feed filters.

Filters of this type usually require cast-iron construction with woven metal-cloth covers and differ from conventional design by small drum dimensions together with large internal conduits in the drum. Vacuum pumps used handle large volumes of air at low-gage readings, 5 in. Hg, or less, being usual.

Beet-sugar Plant Filtration (Saccharate and Carbonation Mud). The recovery of dissolved sugar from beet juice includes the use of filters at two stages: first, in the separation of the carbonated beet juice from the insoluble impurities and lime sludge produced by carbonation. In this stage, the filters remove a clear first and second carbonation juice and wash the resulting lime cake to a low sucrose content in a single filtration.

The second stage consists of the separation and washing of calcium saccharate in the Steffens process. This is a two-stage operation wherein one filtration removes the trisaccharate, and, after further lime treatment, a second filtration removes the mono- and disaccharate. This latter step completes the removal of sugar from the beet molasses, since the first precipitation as trisaccharate does not remove the entire sugar content.

Before continuous vacuum filters became the standard for beet-sugar factories, pressure filtration was the rule, but economies gained in substituting continuous vacuum filters in this work have made the use of pressure filters for this duty a rarity.

Summarizing, the results of continuous vacuum filtration in beet-sugar factories are:

Minimum sucrose left in washed lime cake.
Minimum impurities left in washed saccharate cake.
Minimum use of wash water and evaporation cost.
Minimum cost for filter-cloth renewals and acid treatment.
Minimum labor requirements and use of skilled labor unnecessary.
Clarity of filtered carbonation juice is high.

For a typical Steffens A factory installation, using 3.75 per cent CaO addition, a vacuum filter on carbonation mud will handle about 1.35 equivalent tons of sliced beets per square foot of filter area per 24 hr.; and a "milk-of lime" plant, at 2.50 per cent CaO addition, will handle 2.00 equivalent tons; variations in the amount of lime affect the capacities of the filter proportionately.

Calcium saccharate filtration is based directly upon the amount of molasses treated and gives a rate of about 190 lb. molasses per square foot of filter area per 24 hr. on tricalcium saccharate.

The filter feed on carbonation-mud-filter units is usually about 40° Brix. or about 17.5 per cent cake-forming solids; the pulp is filtered and washed on external drum filters to 1.5 to 2.0 per cent sucrose on CaO. To effect this, as little as 110 per cent wash water is used in terms of wet cake.

The feed to the tricalcium saccharate filters is about 12.5° Brix. with 6.4 per cent sugar, and the wash water from the filter contains about 0.70 to 0.75 per cent sugar.

Petroleum Products. The major problems involving filtration in the refining and processing of petroleum oils include:

1. Contact filtration of contacted petroleum products such as lubricating stocks.
2. Clarification of cracking-still residuum.
3. Filtration of chilled oils to remove wax.
4. Miscellaneous problems, including secondary clarification after contacting, or the "polishing" of products to ensure absolute freedom from solids just prior to shipment, emulsion breaking, etc.

In item 1, finely divided bleaching clay is added (frequently after acid-treating the oil), thoroughly agitated with the oil at the required temperature, and then removed by filtration which produces the required color. The amount of clay employed and the temperatures used vary with the different oils. Usually from 1 to 3 lb. clay per barrel of gasoline will suffice, while in the case of high-viscosity lubricating oils from 4 to 15 per cent of clay (based on the weight of oil) may be necessary. Shell-type filters are generally employed, as the percentage of clay is either very low, as in the case of gasoline, or else the temperatures are near the flash point of the oil (or the naphtha used in cutting back the more viscous oils prior to filtration). Rates per square foot of filter area will vary from as much as 75 gal. on gasoline with little clay, to as little as 2 to 3 gal. on very viscous oils. Continuous removal of contact clay from lubricating oils is now more effectively done by precoat filters. Continuous drum filters, such as the vacuum precoat, are also finding application on problems in item 4.

Not all cracking-still residuums have been successfully filtered. On certain residuums, rates as high as 7 gal. per sq. ft. filter area are being obtained. In some cases filter-aid material is being used. Shell-type filters are generally employed as temperatures frequently range as high as 650°F.

Item 3, as noted above, includes the recovery of marketable paraffin wax and also (secondly) the filtration of oils, using a special solvent, for the primary purpose of lowering the cold test. In the first case, special high-pressure plate filters with heavy canvas-filter pads are generally employed and rates are low and pressures range from 300 to 400 lb. per sq. in. In lowering the pour point of oils, either continuous vacuum filters or shell-type pressure filters are employed depending upon the requirements of the individual problem.

Shell-type filters have generally been employed in division 4, as the percentage of solids is usually very low and filter aids are frequently employed.

Settled Cane-mud (Cachaza) Filtration. In producing sugar from sugar cane in defecation mills making raw sugar, and also in defecation-sulfitation mills making plantation white sugar, the common practice is to settle the hot, treated sugar juice; to draw the clarifier juice from the upper parts of the settling tanks; and likewise to draw the settled muds from the lower parts of the settling tanks. These muds are subsequently filtered, often after additional "liming" and sometimes settling. This filtration is decidedly difficult in the large majority of cane mills.

The rapidity with which filter cloths become clogged has resulted in the retention of the plate-and-frame type of filter although there are some notable installations of leaf-type pressure filters, particularly in Java and in the Philippine Islands. A plate-and-frame filter-press station in a sugar mill is usually the highest loss, highest cost, most unsightly, and most objectionable station in the mill.

In recent years continuous vacuum filters are rapidly replacing plate-and frame-press installations; seldom are plate-and-frame presses installed now in a new factory, preference being given to vacuum filters. The method now generally adopted is to send the settled muds either from open defecators or continuous clarifiers after hot liming treatment to specially designed vacuum drum filters using perforated metal plates as the filter medium, the filtrate being divided into two parts, the first or cloudy filtrate being returned to process and the second or main filtrate going direct to the evaporators.

Filter capacities vary between wide limits in different localities and in different mills in the same locality, being determined to a large extent by factors outside the control of the operating staff, such as the nature of soil and the climatic conditions, and also by other factors under control, such as chemical control and mill conditions. The two primary requisites for vacuum-filter operation are relatively thick muds and sufficient fine cane fiber in the muds for formation of a filter cake. Capacities will ordinarily be from a minimum of 5 to a maximum of 10 tons (2000 lb. per ton) of cane per square foot of filter area per 24 hr.

The cake on vacuum filters is washed in the usual manner for the removal of sugar solution and no difficulty is experienced in reducing the sucrose content of the cake to a maximum of 1.25 per cent. With care, the sucrose content can in all cases be reduced to the point where no further advantages are gained by further lowering the sucrose content.

The advantages of vacuum filters are:

1. Continuous and automatic filter operation.
2. Absolute elimination of leaks and unknown losses at filter station.
3. Complete change from the most unsightly and objectionable station to a clean and entirely presentable station.
4. Large reduction in filter area.
5. Large reduction in space required for filter station.
6. Entire elimination of filter-cloth troubles and frequent washing and renewals; when a punched-plate filter medium is used, no treatment is required from beginning to end of grinding season, and the estimated life of the medium is 4 years.
7. Minimum possible loss of sucrose in filter cake.
8. Large reduction in labor requirements.
9. Minimum dilution of juice owing to consistent low wash-water consumption. This results in decreased fuel requirements and in increased capacity of evaporators.
10. Steadily consistent results because the human element is largely obviated.
11. Rapid, efficient filtration with practical elimination of inversion losses and maintenance of best possible purities.
12. Practical elimination of overtaxing the filters by increasing the milling capacity without providing additional filters. Vacuum filters can be speeded up to a certain point to provide for increased capacity with small loss in efficiency, but the time required for washing the filter cake cannot be eliminated or reduced as with plate-and-frame filters, nor is it advantageous to native labor to forget the washing, or otherwise mis-

treat the filter station and convenience in running sugar juice to waste is entirely prevented.

Food Products. In the field of food production nearly every important product undergoes filtration at some stage. Important instances are starch and corn syrup derived from conversion of corn. Another field is in the manufacture of beer and wine, while the fruit juice, soft-drink, and vinegar industries supply many other applications. Starch is handled on rotary disk filters, and corn syrup finds the vacuum precoat filter useful. The liquids in the varied beverage industries are usually handled to best advantage by some type of pressure-filter unit with or without use of filter aid (Fig. 24).

Caustic Soda. After thickening, the calcium carbonate, precipitated in the manufacture of caustic soda by the lime-soda ash process, is generally dewatered and washed on vacuum filters.

This precipitate filters quite readily from a feed moisture of 50 to 70 per cent moisture to a cake containing from 30 to 40 per cent moisture. Capacity ranges from 500 to 1200 lb. per sq. ft. per 24 hr., and the vacuum requirements correspond to about 6 cu. ft. of free air per minute per square foot of filtration area.

Metallic filter mediums are used. Iron and steel must be used for the filter proper.

Phosphoric Acid. A precipitate of anhydrous calcium sulfate is formed in the manufacture of phosphoric acid by digesting phosphate rock with sulfuric acid. The filtration and washing of this precipitate on vacuum filters at 150° to 165°F. require the use of special corrosion-resisting materials for the filtration mediums and for the filter proper when handling 20° to 35°Bé. acid. Nitrated cotton cloth and KA$_2$ metal cloth have been used as filter mediums. Cast lead, rubber-covered steel, and KA$_2$ metal have been found satisfactory for filter construction. When handling up to 20°Bé. acid, filter construction of wood, with lead, rubber, or KA$_2$ fittings, and wool cloth has been found satisfactory.

The precipitate filters readily from a moisture content of 65 per cent in the filter feed to a cake moisture of 30 to 35 per cent. Capacity ranges from 2000 to 4000 lb. dry $CaSO_4$ per square foot.

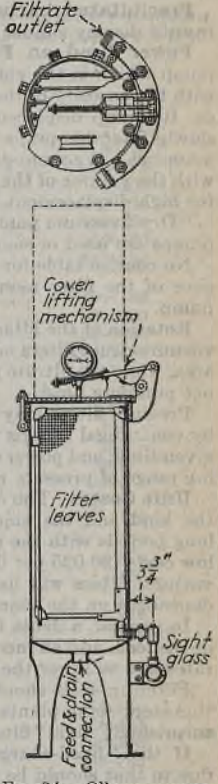


FIG. 24.—Oliver pressure filter.

Pigments. Lithopone is a fine white pigment of about 70 per cent barium sulfate and 30 per cent zinc sulfide, simultaneously precipitated by mixing solutions of barium sulfide and zinc sulfate. Vacuum filters are used for dewatering the thickened raw lithopone and the finished lithopone after calcination and grinding.

Lithopone is representative of the group of chemically precipitated mineral pigments. It filters slowly, is extremely finely divided, and must be filtered in the form of a very thin cake.

The thickened feed generally enters the vacuum filter at 50 to 70 per cent moisture and is discharged at 35 to 50 per cent moisture; capacity ranges from 200 to 400 lb. per sq. ft. per 24 hr.

Corrosion-resisting construction must be used, particularly in the filtration of the raw lithopone. Approved materials of construction are wood, monel metal, aluminum-zinc alloys, and rubber-covered iron and steel.

Precipitated barium sulfate, white lead and titanium-base pigments display similar filtration characteristics to lithopone.

Power Used on Filters. Vacuum-filter operation requires a vacuum pump which is the chief consumer of power. The pump size varies widely with the porosity of the cake formed on the filter and will range from about ½ cu. ft. pump displacement per square foot of filter area for dense cakes of slowly filterable pulps up to as high as 40 to 60 cu. ft. per sq. ft. for crystalline materials. Vacuum-gage readings are nearly always found at inverse ratio with the volume of the pump displacement, which avoids excessive power costs for high displacement.

"Dry" vacuum pumps (piston type) are preferred, although rotary-vacuum pumps are used in some special cases.

No concise table for such power consumptions can be compiled, the experience of the filter manufacturer being the safest criterion for the choice of pump.

Rotation of the filter itself consumes very little power, the larger size rotary vacuum-drum filters not requiring more than about 0.005 h.p. per sq. ft. filter area, while the filtrate pumps are chosen for volume of filtrate handled and the net pumping head.

Pressure filters may be operated by gravity head on the filter-feed line, or by centrifugal pumps whose size depends on the volume to be handled in a given time; and power depends upon volume handled, size of filter, and operating range of pressure required.

Unit Costs. The continuous-rotary-vacuum-filter costs vary widely with the kind and the amount of feed handled. While consistent figures over long periods with the rotary-vacuum filter have, in a given case, shown very low costs ($0.025 per dry ton of filter cake), it is likely that most continuous vacuum filters will have cost figures ranging from $0.05 to $0.15 per ton, dependent on the tonnage handled and the material itself.

In general, a dense feed to the filter, an absence of pasty or colloidal solids in the feed, and an increase of pulp temperatures serve to increase the filtering rates and to lower the unit filtering costs, while the converse increases them.

Filtering costs should include only those operations directly pertaining to this step; some plants include other operations therein which explains some surprisingly high "filtering costs" in otherwise well-supervised plants.

If the "fixed-charges" burden is included in the cost item, the amount due to that should be separately indicated.

The cost of pressure filtration varies widely with the materials handled, and as such installations are usually small, compared with continuous vacuum filters, the costs are consequently higher, but the difficult kind of work performed justifies the higher operating costs in most cases.

Floor Space Available. The floor space available affects the choice of type of filter in some cases. For example, if a large filter area is required to handle the product, and the floor space available is restricted, there is at once an advantage in the disk type of vacuum filter, other factors being approximately equal. The saving in building costs and heating expense is often a considerable factor.

Plate-and-frame presses occupy less floor space than shell-type pressure filters for the same filter area. Shell-type filters require more room for opening and closing but, if large units are employed, space may be economized by installing the twin-type units.

Cost of Equipment. Prices on filters vary so widely, depending upon size, type of filter, and material of construction, it is impractical to set a unit price common to all such varieties.

In the case of continuous rotary vacuum filters, a typical price per square foot of filter area (not including wash apparatus or accessories), would, in the case of a plain dewatering type, be about $16 to $18 per sq. ft. filter area in the smaller sizes, and about $10 to $12 per sq. ft. in the larger sizes.

Table 1. Range of Filter Sizes

	Area, Sq. Ft.
Oliver	
1-ft. diam. × 1-ft. face	3
3-ft. diam. × 6-in. face to 4-ft. face	4– 36
4-ft. diam. × 2-ft. face to 6-ft. face	25– 75
5-ft. various in. diam. × 4-ft. face to 12-ft. face	65–200
6-ft. diam. × 1-ft. face to 12-ft. face	18–226
8-ft. diam. × 6-ft. face to 14-ft. face	150–350
11½-ft. diam. × 10-ft. face to 16-ft. face	360–570
14-ft. diam. × 14-ft. face to 18-ft. face	610–790
American	
4-ft. diam. × 1 disk to 4 disk	22– 86
6-ft. diam. × 1 disk to 6 disk	50– 300
8-ft. 6-in. diam. × 2 disk to 10 disk	185– 925
12-ft. 6-in. diam. × 5 disk to 12 disk	1000–2400
Sweetland	
No. 1	8½
No. 2	47
No. 5	191
No. 7	261
No. 10	538
No. 12	1020
Kelly	
No. 30	33
No. 50	50
No. 250	270
No. 450	459
No. 650	652
No. 900	918
No. 1300	1304
Dorrco	
4-ft. diam. × 1-ft. and 2-ft. face width	12– 24
6-ft. diam. × 2-ft. and 3-ft. face width	37– 55
8-ft. diam. × 3-ft. to 14-ft. face width	74–345
10-ft. diam. × 4-ft. to 16-ft. face width	124–494
12-ft. diam. × 5-ft. to 16-ft. face width	190–596
14-ft. diam. × 16-ft. to 18-ft. face width	696–783

For the special designs required in the wood-construction filters for acid-resisting use, and those built for work with caustic material, the smaller sizes cost about $30 to $40 per sq. ft. filter area, and the larger sizes about $20 per sq. ft. filter area. These prices are suitable for rough preliminary estimations only.

Cost of accessory vacuum equipment, pumps, etc., for vacuum-filter installations will range around 25 to 33 per cent of the filter cost, additional.

All costs mentioned herein are on the basis of quotation f.o.b. point of manufacture.

Table 2. Factors Affecting Selection of Type of Filter and Character of Pulps Handled

Typical materials	Character	In. Hg vacuum or lb. pressure	Approx. filter capacity, lb. per sq. ft. per day	Type of filter suitable		
				Plate and frame	Shell type	Continuous vacuum
Cyanide slime	Finely ground quartz ores	18–25 in.	400– 2,000	..	..	X
Flotation concentrates	Minerals, finely ground	18–25 in.	400– 1,800	..	..	X
Gravity concentrates and sand.	Metallic and non-metallic minerals almost free from slime	2– 6 in.	10,000–70,000	..	..	X
Cement slurry	Finely ground limestone and shale, or clay, etc.	18–25 in.	400– 2,000	..	..	X
Pulp and paper	Free-filtering fibers	6–20 in.	200–1,200 and 1½–20 gal. water per sq. ft. per min.	..	..	X
Crystals, salt, etc	Granular, crystalline	2– 6 in.	3,000–12,000	..	..	X
Cane-sugar-liquor clarification, beverages, etc.	Syrups and solution with small percentage of solids with filter aid	40–50 lb.	36–1,400 gal. per sq. ft. per day	X	X	
Pigments	Smeary, sticky, finely divided, non-crystalline	20–27 in.	200– 500	..	..	X
		40–50 lb.	Batch operation	X	X	
Sewage sludge	Colloidal and slimy	22–24 in.	25 to 250	..	..	X
Varnish	Cloudy viscous liquid, filter aid used for clarification. Filtered hot	15–16 lb.	5 gal. per sq. ft. per hr.	X		
Mineral oils, with or without wax.	Removal of bleaching clay from petroleum products. 1 to 20% clay used	50-lb. max. pressure	3–30 gal. per sq. ft. per hr. (lubricating oils)	..	X	
			25–75 gal. per sq. ft. per hr. (gasoline)	..	X	
Cane mud	Vegetable fiber and cane juice	....	....	..	..	X

Corrosion. When handling corrosive material, vacuum filters are more affected than other types because there are more parts exposed and the passage of air through the filter increases oxidation and corrosion. Pressure filters are not exposed to the same extent and because of simplicity lend themselves more readily to corrosion-resisting construction.

The use of special alloys, however, makes it possible to construct filters to suit given conditions of corrosion.

Filter Auxiliaries

Pressure Pumps. For supplying feed to pressure filters, centrifugal and plunger pumps and, sometimes, hydrostatic head pumps are used. Montejus, or pressure tanks, are used in special cases.

Open-impeller centrifugal pumps are recommended where it is desired to start filtration with low pressure and to increase the pressure as the cake thickness increases.

Plunger pumps deliver feed at a regular rate depending upon the pump displacement. They give high initial pressure which may be maintained throughout the cycle.

Montejus were largely used in the early days of pressure filtration. A monteju is a pressure tank which may be filled with the filter feed, after which compressed air is admitted at the top and forces the feed through an outlet pipe leading to the pressure filter. Montejus are installed in pairs so that the feed can be supplied steadily, one being filled while the other is being emptied.

Vacuum Pumps. There are two classes of vacuum pumps, wet and dry. The former handle both the filtrate and the entrained air while the latter handle the air only. Wet vacuum pumps are usually of the rotary type of which the Nash Hytor and Connersville cycloidal are examples.

The Nash pump (Fig. 25) is well adapted for handling large volumes of air with considerable liquid at a moderate vacuum (16 in. Hg) and is much used in the pulp-and-paper industry where this type of service is required.

The Connersville cycloidal type handles large volumes of air at low vacuum (2 to 6 in. Hg) with entrained liquid at lower power consumption. This type pump is particularly useful when filtering granular or crystalline products such as salt, and the air is either heated or is at normal temperature. These pumps are also used when both air and filtrate are handled together.

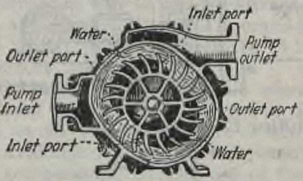


FIG. 25*a*.—Nash vacuum pump.

FIG. 25*b*.—Nash vacuum pump (dismantled).

Dry vacuum pumps are available in both reciprocating and rotary types; the designs of each are well standardized and there are many excellent ones to choose from.

Receivers or Separators. When the filtrate and air from a vacuum filter are discharged separately, a receiver or separator is used consisting of a cylindrical tank usually installed with its long axis vertical (Fig. 26*a*). The incoming filtrate and air enter at the side, air is withdrawn through the vacuum line at the top, and filtrate drains away or is pumped out through a connection at the bottom.

Moisture Traps and Condensers. If a dry vacuum pump is used, it is necessary to install a moisture trap or condenser (Fig. 26*b*), the purpose being to prevent filtrate or condensed moisture from entering the pump. A moisture trap suffices when filtering at normal temperatures, but a condenser is necessary to maintain the vacuum-pump efficiency when operating at temperatures approaching the flash point of the filtrate at the vacuum employed.

A trap is merely a small receiver, and a condenser is similar but is equipped with cold-water showers and baffles. The installation may be made at a height allowing at least 30 ft. vertical between the bottom of the trap or condenser and the seal pit in which the discharge drain pipe terminates (Figs. 25 *a* and *b*). If preferred, the outlet of the drain pipe may be connected to the suction of the filtrate pump.

Filtrate Pumps. These pumps are usually centrifugals of the closed-impeller type. They pump filtrate from the receiver and discharge it at the desired height. They should be specially designed, since they are called upon to handle both water and air under vacuum and usually discharge against positive hydraulic heads. The manufacturer of the filter equipment

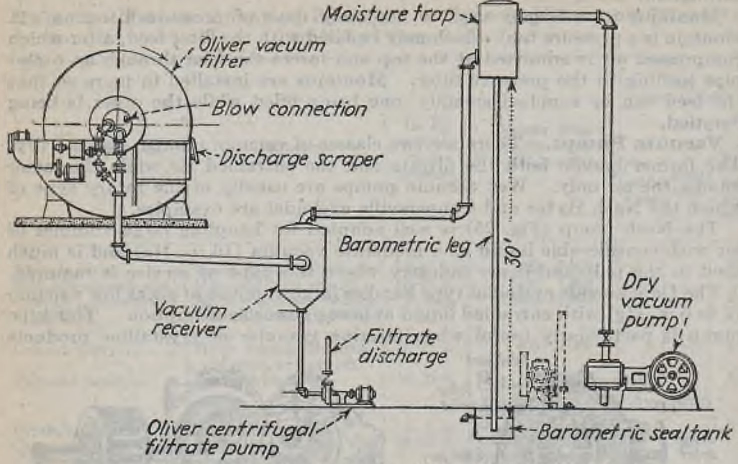


FIG. 26*a*.—Vacuum filtration unit. (*Oliver United Filters, Inc.*)

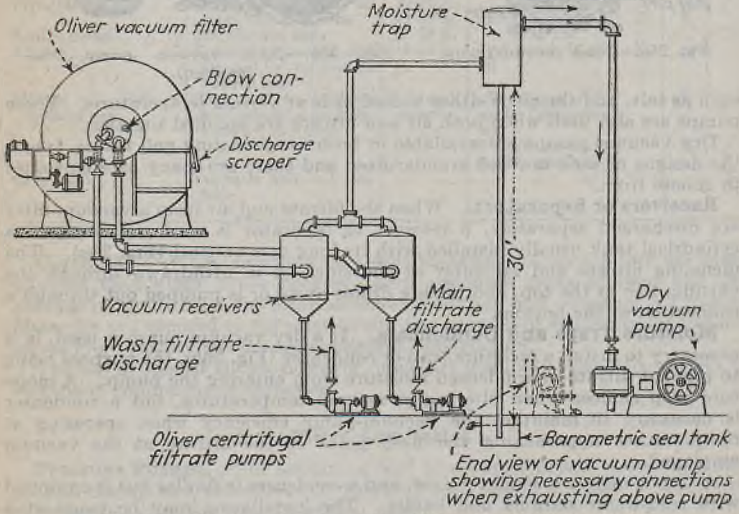


FIG. 26*b*.—Vacuum filtration unit. (*Oliver United Filters, Inc.*)

should always be consulted as to the design of pump best suited to the equipment involved.

Barometric Legs. Where the product is free filtering and the amount of liquid to be removed is large, the use of vacuum pumps may be avoided by discharging the filtrate through a barometric leg. This consists of a vertical pipe which discharges into a seal box, the overflow from which goes to waste. The length of vertical discharge should equal the height of the water-barometer, and if a vacuum pump is employed to handle the air from the filter there is no danger of water being drawn into the system.

The vacuum created depends upon the velocity of the water in the pipe, therefore the pipe diameter and the height must be adjusted to give the filtrate sufficient velocity to draw out the entrained air, thus creating a vacuum at the filter. The leg should be as nearly vertical as possible.

In pulp and paper mills many filter installations operate with barometric discharge of the filtrate. It is the simplest way to handle filtrate and air collectively and avoid installation and operating expense of vacuum and filtrate pumps; usually 10 to 20 in. vacuum is obtained by this method.

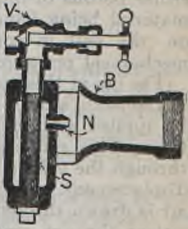


FIG. 27.—Air-injectc blower. (*New Jerse Meter Co.*)

Blowers. Most materials when handled by vacuum filters require an air blast to assist the discharge of the cake. The volume of air used is small, and usually a low pressure is sufficient. Where compressed-air service is not available a small air compressor, or a rotary blower, may be employed. Air compressors, if used, are of the ordinary displacement pump types, but rotary blowers are best suited for pressures under 3 lb. per sq. in.

The disk type of filter requires only a few ounces pressure and an air-injector blower will give this (Fig. 27).

The pulsating air blow used for the Dorrco filter merely requires a rotating valve in the air line to produce air pulsations under the filter medium.

The function of an air blow is quickly to fill the section with air at sufficient pressure to flex the cloth, and to issue through the pores of cloth, thereby dislodging the cake and freeing the fine particles which might otherwise remain and cause cloth blinding.

Flappers. A flapper is an appliance used with the external-drum type of vacuum filter to reduce the cake moisture (Fig. 28). It is installed above, and parallel to, the drum axis and consists of a shaft to which are attached two pieces of heavy fabric of such length that they strike the cake a regular succession of blows when the shaft is rotated. The blows cause a rearrangement of particles in the cake, closing the cracks and liberating moisture which is then drawn through into the vacuum system.

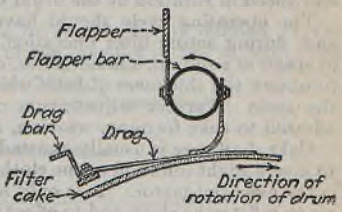


FIG. 28.—Flapper. (*Oliver United Filters, Inc.*)

To prevent the cake from being dislodged by the blows of the flapper, a piece of heavy fabric is fastened so that it drags upon the cake as the drum revolves and receives the blows given by the flapper. The flapper reduces the moisture from two to four units which may be 10 to 20 per cent of that in the unflapped cake.

Compression Belts. A cake-compression belt is an endless belt which rides over a system of rollers mounted in a frame above the drum of an external-drum type of vacuum filter. The belt comes in contact with the cake at a point on the ascending side of the drum and leaves at the descending side above the cake discharge. It is driven by contact with the cake, and lateral adjustment is provided by a tracking roller. The belt may be either porous or impervious depending upon the material being handled. Weighted rolls riding on the belt in contact with the cake enable mechanical pressures to be applied.

The use of this appliance tends to close cracks in the cake, and in some instances the belt may be applied soon enough to prevent their formation. This reduces the amount of air drawn through the cake and reduces the vacuum-pump displacement. Moisture is reduced since all the air is drawn through the cake and cannot short-circuit through cracks. Pressure from the rolls expels additional moisture. This arrangement is useful when handling pigments and other finely divided materials.

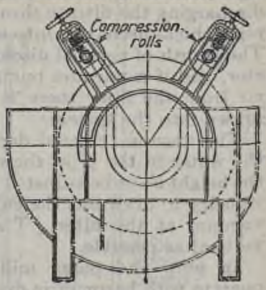


FIG. 29.—Compression rolls.

Compression Rolls. Compression rolls without belts are used for reducing the moisture in crude bicarbonate of soda, paper pulp, etc. (Fig. 29).

Filter Operation

In operating pressure filters, the process is intermittent and the product obtained from each cycle when compared with that previously obtained indicates whether there has been a change of conditions. During the cycle, however, the operator should control the feed pressure, note the clarity and quantity of the filtrate issuing, and any other factors that influence the particular problem.

In vacuum filtration there are advantages in that the controlling factors are visible and accessible at all times. The feed is in an open container, and cake thickness can be controlled by regulating the submergence, vacuum, and speed of rotation of the drum or disk.

The operating cycle should have been determined by preliminary tests, and, during actual filter operation, changes and adjustments may be made in speed of rotation, submergence, cake-forming vacuum, and agitator speed, to obtain the thickness of cake which will wash and dry in the remainder of the cycle. Further adjustments may be made in the proportion of time allotted to cake forming, washing, and drying.

Cake discharge is usually assisted by an air blast which should be regulated to give a slight inflation of the cloth. The volume of air used for this purpose is the essential factor. It is only necessary completely to fill the sector with air to inflate the cloth and to effect the discharge of the cake. Air pressure is a secondary matter. Generally, the automatic valve should be set so that the air blast occurs as the leading edge of the section comes in contact with the scraper at the point of discharge. The filter medium should be maintained in a clean condition at all times, and this is possible only by a complete removal of cake at each revolution. If the material being filtered tends to blind the medium, it will be necessary to wash it at regular intervals to maintain capacity. Experience in operating a filter upon a specific problem is the best guide.

MISCELLANEOUS METHODS OF SIZING AND CONCENTRATION OF MATERIALS

SCREENING

By Kenneth H. Donaldson

DEFINITIONS

Screening, or screen sizing, is the separation of a mixture of various sizes of grains into two or more portions, by means of a screen; the grains of any one portion being of a more uniform size than those of the original mixture.

Dry screening refers to a process where the material handled on the screen contains a natural amount of moisture or where the natural moisture has been removed by drying before screening.

Wet screening refers to a screening process where water is added to the material before it is delivered to the screen or after it reaches the screen, or both.

Mesh. In the coarser sizes of screens the term *mesh* means either the distance between adjacent wires or rods or the distance between centers of adjacent wires or rods. In the finer sizes of screens the mesh means the number of openings per linear inch. Where square openings are not used the screen size is frequently designated by numbers.

Oversize and Undersize. Where the separation is such that only two products are made, the material which fails to pass through the screen is called the *oversize* or *plus* material; that which passes through the screen openings is called the *undersize* or *minus* material.

Close Sizing. When the limiting and retaining screens are of nearly the same size of opening then close sizing is said to be practiced.

Sieve Scale. In a special sizing operation, the list of apertures of successively smaller screens is said to constitute a sieve scale.

Clear Area. The clear area of a screen is the combined area of all of the openings presented and the ratio of the clear area to the total screen surface is designated as percentage of opening.

Aperture or Screen Size. Aperture or screen size is defined as the minimum clear space between the edges of the opening in the screen. The

Table 1. Needle Mesh*

Needle number	Largest diameter of head, in.	Equivalent screen
1	0.0735	10 mesh
2	0.0428	16 mesh; No. 3 slot
3	0.0395	16–18 mesh; No. 3–No. 4 slot
4	0.0355	20–24 mesh; No. 4–No. 5 slot (nearer the last)
5	0.0335	20–24 mesh; No. 4–No. 5 slot (nearer the last)
6	0.0300	24 mesh; No. 5–No. 6 slot
7	0.0265	24–26 mesh; No. 6–No. 7 slot
8	0.0230	26 mesh; No. 8 slot
9	0.0203	29 mesh; No. 9 slot

* From Taggart, "Hand Book of Ore Dressing," p. 506, Wiley, New York, 1927.

aperture A, mesh M, and wire diameter D are interrelated by the following equation in which P is the percentage of opening:

$$P = \frac{A^2}{(A + D)^2} = (1 - MD)^2$$

Needle Mesh. Needle mesh is a number indicating the size of needle which is of the same diameter as that of the opening. Table 1 gives the diameter of opening in inches and the mesh of screen corresponding with the first nine needle numbers. This term is, however, not customarily used in chemical engineering work.

Multiple Screening. Where more than two sizes are made, the product may be referred to as oversize; first, second, and third intermediate, etc.; and undersize; or the product may be referred to as oversize and then the various grades expressed in terms according to the openings employed in making the separations as for example:

Oversize, $\frac{1}{4}$ in. $= +\frac{1}{4}$ in. $= \frac{1}{4}$ in.
Through, $\frac{1}{4}$ in. on $\frac{1}{8}$ in. $= -\frac{1}{4}$ in. $+ \frac{1}{8}$ in. $= \frac{1}{4}$ in. $\times \frac{1}{8}$ in.
Through, $\frac{1}{8}$ in. on $\frac{1}{16}$ in. $= -\frac{1}{8}$ in. $+ \frac{1}{16}$ in. $= \frac{1}{8}$ in. $\times \frac{1}{16}$ in.
Undersize $= -\frac{1}{16}$ in.

Screen Efficiency. The efficiency of screens may be considered from two standpoints:

1. The percentage of the undersize in the feed to the screen (as determined by testing sieves) which is found in the undersize of the screening operation. This is the important consideration where it is desirable to separate as large a percentage as possible of the undersize in the feed to the screen.
2. The percentage of the oversize of the feed to the screen (as determined by testing sieves) which is found in the oversize as produced by the screening operation. This is the important consideration when it is desired to make an oversize containing as little undersize as possible.

Screening efficiency is affected by the rate of feed. The higher the feed rate the less efficiency, and vice versa. If the material being screened is slightly moist, the efficiency is greatly lowered; if the feed material contains relatively large amounts of oversize or of difficult grains, close to the size of the mesh openings, the efficiency is also lowered.

It would seem fairer to calculate the screen efficiency by two methods. The first method might be called **plant screen efficiency** and the second method, **theoretical efficiency of screen.**

To determine the plant screen efficiency, the material to be tested should be taken to the test sieves directly and in the same condition as it exists in the plant. To determine the theoretical efficiency of the screen, the sample to be tested should be washed through the finest sieve to be used in the test. The oversize from this washing should be dried and then screened through the various test sieves including the one the sample was washed through, as the dried sample may contain a considerable amount of material which will pass through the screen used for washing.

Under certain conditions it is impossible to wash the material through the screen owing to some physical or chemical changes which may take place when water is added to the material.

Screening efficiency may be determined by sieve analysis without taking any tonnage samples with probably as great accuracy as if tonnage samples were taken.

From the standpoint of undersize the screen efficiency is given in the formula

$$\text{Per cent efficiency} = \frac{100 \times \text{actual undersize}}{\text{true undersize}}$$

where actual undersize means undersize as obtained in the plant, and true undersize is the undersize as determined by test-sieve analysis.

From the standpoint of oversize the screen efficiency is given by the formula

$$\text{Per cent efficiency} = \frac{\text{true oversize} \times 100}{\text{actual oversize}}$$

where true oversize means oversize as determined by test-sieve analysis, and actual oversize means oversize as obtained in the plant.

The W. S. Tyler Company gives the following formula for screen efficiency:

R = fractional recovery of fines in screening.
E = percentage of fines recovered in screening.
a = percentage of coarse in feed to screen.
b = percentage of fines in feed to screen.
c = percentage of coarse in oversize after screening.

$$R = \frac{100(c - a)}{(bc)}$$

$$E = 100R = \frac{100(c - a)}{bc} \times 100$$

Certain other formulas for the derivation of screen efficiency are used. Taggart, in his "Handbook of Ore Dressing," suggests the formula:

$$E = 100\left[\frac{100(u - o)}{u(100 - o)}\right]$$

where E is equal to the efficiency of screen, u is the percentage of undersize in the feed, and o is the percentage of undersize in the screen oversize. Wiard states that the average commercial efficiency of screening is about 60 per cent and that 75 per cent is unusually good, and 90 per cent about the limit that may be reached by careful hand screening. Taggart's handbook also gives the following formula derived by Warner:

$$E = \frac{u(f - o)}{f(u - o)}$$

where f, u, and o are the percentages of "difficult grain" in the feed, the undersize and the oversize, respectively. Difficult grains are defined by Warner as those particles which are held on the next finer Tyler standard sieve scale, the apertures of which are less than 83 per cent of the aperture of the sieve in question.

Screen Capacity. Leading manufacturers of screening equipment state that it has never proved possible to develop reliable capacity tables or formulas. Capacity and efficiency in screening operations are closely related. If a low efficiency is not objectionable, the capacity may be large. Usually as the tonnage of the feed to the screen is increased, the efficiency is decreased.

A formula having somewhat wider applicability is given by Newton [*Rock Products*, **35**, 26 (1932)] as follows:

$$E = \frac{(o - b)(u - a)}{ou(1 - a - b)}$$

where E = screen efficiency (as a decimal).
o = per cent oversize in feed (as a decimal).
u = per cent undersize in feed (as a decimal).

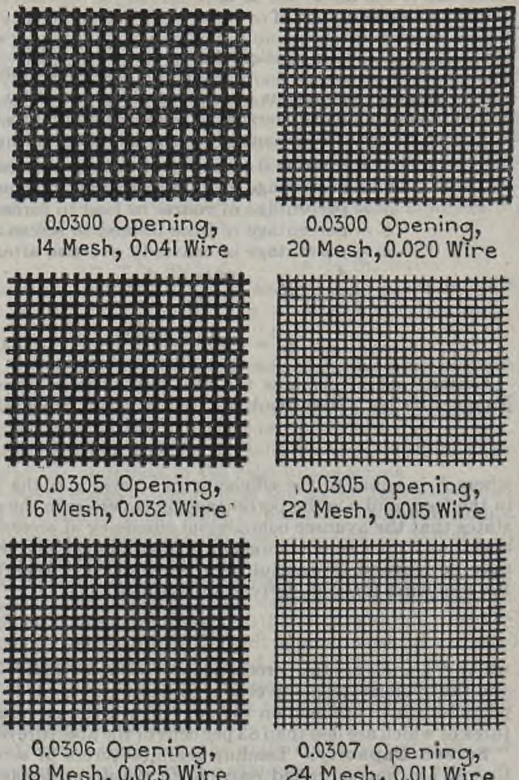
a = per cent undersize in tailings (as a decimal).
b = per cent oversize in fines (as a decimal).

In this case, "oversize" is material larger than the screen cloth opening, "undersize" is material smaller, "tailings" is that which passes over the screen, and "fines" is that material which passes through the screen cloth.

With a constant efficiency desired the following factors affect the capacity of the screen:

1. The ability of the screening device to prevent "blinding" of the screen surface is probably the most important single factor in the capacity of the screen. By blinding is meant the fact that grains, or masses of grains which adhere together, will not pass through the opening but wedge in it thus reducing the open area available for passing fines. The vibration of the screen cloth undoubtedly reduces the blinding to a minimum.

2. In "dry" screening, the greater the amount of moisture in any particular material the lower the capacity of the screen; but the finer the material the more moisture it may carry owing to the greater surface exposed, without materially lowering the capacity.



0.0300 Opening, 14 Mesh, 0.041 Wire

0.0300 Opening, 20 Mesh, 0.020 Wire

0.0305 Opening, 16 Mesh, 0.032 Wire

0.0305 Opening, 22 Mesh, 0.015 Wire

0.0306 Opening, 18 Mesh, 0.025 Wire

0.0307 Opening, 24 Mesh, 0.011 Wire

FIG. 1.—Diagram showing effect of wire size in per cent of open area.

3. The greater the amount of "near-mesh" size the lower the capacity of the screen. For example, if the size of the opening is ¼ in. and there is a large proportion of $\frac{3}{16}$-in. grains in the material to be screened, the capacity will be much lower than if most of the material is finer than $\frac{1}{32}$ in. in size.

4. The finer the screen used the lower the capacity.

5. The percentage which the open area is of the total area has an important bearing on the capacity of the screen.

Figure 1 shows six screens with the same opening but different sizes of wire. The top screen has 14 meshes to the linear inch, while the bottom screen has

Table 2. Steel-wire Screens and Cloths

Heavy Steel-wire Screens

Size of opening, in.	Range in diameter of wire, in.	Range in approx. weight, lb. per sq. ft.	Range in % open area
	Max. Min.	Max. Min.	Min. Max.
4	1 – ⅜	13.70–2.16	64.0–83.6
3¾	1 – 5/16	14.44–1.61	62.3–85.2
3½	1 – 5/16	15.29–1.72	60.5–84.3
3¼	1 – 5/16	16.23–1.74	58.5–83.2
3	1 – ¼	17.35–1.29	56.3–85.2
2¾	1 – ¼	18.49–1.40	53.7–84.0
2½	1 –0.225	19.93–1.16	51.0–84.1
2¼	1 –0.207	21.67–1.17	47.9–83.8
2	1 –0.192	23.69–1.14	44.4–83.2
1¾	1 –0.192	26.02–1.30	40.5–81.2
1½	1 –0.177	28.92–1.24	36.0–80.0
1¼	¾ –0.177	20.22–1.47	39.1–76.7
1	¾ –0.162	23.40–1.50	32.6–74.0
⅞	⅝ –0.148	18.97–1.41	34.0–73.5
¾	⅝ –0.148	20.53–1.63	29.7–69.8
⅝	9/16–0.105	17.91–1.01	27.7–73.4
½	7/16–0.105	14.08–1.22	28.4–68.3
7/16	5/16–0.105	9.48–1.37	34.0–65.0
⅜	5/16–0.105	10.42–1.33	29.7–61.0
5/16	0.225 –0.092	6.87–1.42	33.8–59.6
¼	¼ –0.092	8.96–1.66	25.0–53.4
3/16	0.192 –0.092	7.60–2.04	24.4–45.1

Steel-wire Cloth

Size of opening, mesh	Range in diameter of wire, in.	Range in width of opening		Range in approx. wt., lb. per sq. ft.	Range in % open area
		In.	Mm.		
	Max. Min.	Min. Max.	Min. Max.	Max. Min.	Min. Max.
1	0.307 –0.072	0.693 –0.928	17.60 –23.57	7.52–0.39	48.0–86.1
⅝	0.283 –0.063	0.467 –0.687	11.86 –17.45	7.75–0.39	38.8–83.9
¾	0.263 –0.047	0.362 –0.578	9.20 –14.68	8.51–0.31	33.5–85.5
2	0.225 –0.041	0.275 –0.459	6.99 –11.66	7.62–0.24	30.3–84.3
2½	0.192 –0.035	0.208 –0.365	5.28 – 9.27	7.02–0.22	27.0–83.3
3	0.162 –0.032	0.171 –0.301	4.34 – 7.65	5.78–0.23	26.3–81.5
3½	0.148 –0.032	0.138 –0.254	3.51 – 6.45	5.62–0.24	23.3–79.0
4	0.135 –0.028	0.115 –0.222	2.92 – 5.64	5.03–0.22	21.2–78.9
4½	0.120 –0.025	0.102 –0.197	2.59 – 5.00	4.64–0.20	21.1–78.6
5	0.105 –0.023	0.095 –0.177	2.41 – 4.50	3.84–0.19	22.6–78.3
6	0.092 –0.020	0.075 –0.147	1.91 – 3.73	3.49–0.17	20.2–77.8
7	0.080 –0.018	0.063 –0.125	1.60 – 3.18	3.15–0.16	19.5–76.6
8	0.072 –0.017	0.053 –0.108	1.35 – 2.74	2.78–0.20	18.0–74.6
9	0.063 –0.016	0.048 –0.095	1.22 – 2.41	2.52–0.16	18.7–73.1
10	0.054 –0.015	0.046 –0.085	1.17 – 2.16	1.96–0.16	21.2–72.3
14	0.041 –0.010	0.030 –0.061	0.762– 1.549	1.51–0.10	17.6–72.9
20	0.028 –0.009	0.022 –0.041	0.559– 1.041	1.07–0.11	19.4–67.2
28	0.017 –0.009	0.0187–0.0267	0.475– 0.678	0.57–0.15	27.4–55.9
35	0.016 –0.009	0.0126–0.0196	0.320– 0.498	0.59–0.20	19.4–47.1
50	0.010 –0.008	0.0100–0.0120	0.254– 0.305	....	25.0–36.0
64	0.0075–0.00675	0.0081–0.0089	0.206– 0.226	....	26.9–32.4
100	0.0040	0.0060	0.152	....	36.0

24 meshes to the linear inch; therefore the screen with the coarser wire has less than 40 per cent of the open area of the screen with the finer wire, and the capacity of the screen with the coarser wire will be less than 40 per cent of that with the finer wire.

It must be remembered that the life of the screen with the finer wire is much less than where a heavier wire is used. If the material to be screened contains a large amount of abrasive, it may not be practical to use a screen with a fine wire; while, if the material is damp and sticky, the life of the screen may be of only secondary importance.

Table 2 shows the largest and smallest diameter of rod or wire for some of the various woven screens manufactured by the W. S. Tyler Co., Cleveland, Ohio. (See catalogue for the intermediate sizes of screens as well as rods or wires.) See Fig. 1.

SCREENING MEDIUMS

Woven Wire Screens. Woven wire screens may have a square opening as shown in Table 2 or may have a rectangular opening. The various sizes of screens with rectangular openings are frequently designated by numbers rather than by mesh.

For shaking and vibrating screens, the rectangular opening has considerable advantage over the square mesh in that the percentage of open area is larger and that grains which should just pass through the screen cannot touch more than three sides and many will not touch more than two sides of the opening; while, with the square mesh, these grains are almost sure to touch three sides and may touch four sides of the opening. The screen with a rectangular opening does not tend to blind to the same extent as do square-mesh screens.

Square-mesh woven wire screens are obtainable in a variety of metals and alloys such as steel, brass, bronze, copper, nickel, monel metal, and stainless steel. Steel resists abrasion and attrition to the greatest extent, and the other materials should be used only when their corrosion-resisting qualities are of relatively greater importance than their mechanical strength and durability.

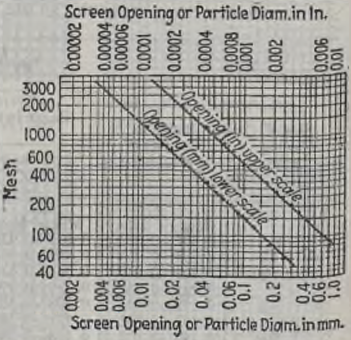


FIG. 2.—Screen opening and mesh for testing sieves, Tyler series.

Although present-day practice prefers the designation of wire sizes by number, showing their diameters in decimal parts of an inch, there are a number of different wire-gage numbers used by different manufacturers. On p. 110 are tabulated a series of wire-gage numbers and their equivalents in decimal parts of an inch according to the standards of various manufacturers and countries.

Punched Metal Plates. Punched metal plates were formerly used quite extensively even in the finer sizes of openings for stationary screens and trommels, but the finer sizes have been largely replaced by woven wire screens for trommels and in many cases for stationary screens where the slope of the screen will permit their use.

It has been shown that a square hole in a wire screen will pass particles of the same size as a circular-hole punched plate, the diameter of which is 1.23 times the length of a side of a square hole. The advantages of a punched plate may be summarized as strength, long life, and ruggedness, while the disadvantages are chiefly great weight per unit of area and difficulty of handling. Punched-plate screens are chiefly used in metal- and coal-mining

Table 3. Punching Steel Plates*†

Diameter of hole			U. S. Standard		
Mm.	In.	Decimal of an inch	No.	Thickness, In.	Weight per sq. ft.
3/4		0.02952	26	0.018	0.765
1		0.03937	24	0.025	1.02
	3/64	0.04687	22	0.031	1.275
1 1/4		0.04921	20	0.037	1.53
1 1/2		0.05906	18	0.050	2.04
	1/16	0.06250	18	0.050	2.04
	5/64	0.07812	16	0.0625	2.55
2		0.07874	16	0.0625	2.55
2 1/4		0.08858	16	0.0625	2.55
	3/32	0.09375	16	0.0625	2.55
2 1/2		0.09843	16	0.0625	2.55
3		0.11811	14	0.078	3.187
	1/8	0.125	14	0.078	3.187
3 1/4		0.12795	14	0.078	3.187
3 1/2		0.1378	12	0.109	4.46
	9/64	0.14062	12	0.109	4.46
4		0.15748	12	0.109	4.46
4 1/2		0.17717	10	0.1406	5.737
	3/16	0.18750	10	0.1406	5.737
5		0.19685	10	0.1406	5.737
5 1/2		0.21654	10	0.1406	5.737
6		0.23622	10	0.1406	5.737
	1/4	0.25	8	0.1718	7.012
6 1/2		0.25591	8	0.1718	7.012
7		0.27559	3/16	0.1875	7.65
	9/32	0.28125	3/16	0.1875	7.65
	5/16	0.3125	3/16	0.1875	7.65
8		0.31496	3/16	0.1875	7.65
9		0.35433	3/16	0.1875	7.65
	3/8	0.375	3/16	0.1875	7.65
10		0.3937	1/4	0.25	10.2
11		0.43307	1/4	0.25	10.2
	7/16	0.4375	1/4	0.25	10.2
12		0.47244	1/4	0.25	10.2
	1/2	0.5	1/4	0.25	10.2
13		0.51181	1/4	0.25	10.2
14		0 55518	1/4	0.25	10.2
15		0.59055	1/4	0.25	10.2
	19/32	0.59375	1/4	0.25	10.2
	5/8	0.625	1/4	0.25	10.2
19		0.74803	1/4	0.25	10.2
	3/4	0.75	1/4	0.25	10.2
22		0.86614	1/4	0.25	10.2
	7/8	0.875	1/4	0.25	10.2
25		0.98425	1/4	0.25	10.2
	1		5/16	0.312	12.75

* *Allis Chalmers Co. Bull.*

† This table gives the greatest thickness of steel in which it is advisable to punch round or square holes of given diameters or sizes. Spacing, strain upon the plate, wear of dies, and other considerations determine what is advisable. While the table is offered as a convenient guide in ordering, thinner plates will generally answer every requirement and cost less.

operations, crushed-stone and gravel sizing, and so forth, where the openings are ½ in. in diameter and greater. Woven wire screens are preferred where the opening is less than ½ in. and finer.

Bolting Cloth. Bolting cloth is useful in the dry screening of paints, pigments, ceramic materials, and chemical precipitates of relatively fine size. Table 4 shows the sizes of apertures in bolting cloth, the corresponding meshes per inch, and the percentage of opening.

Table 4. Sizes of Apertures in Dufour's Bolting Cloth[*]

Maker's No.	Meshes per in.	Aperture, in.	Screen area, %
0000	18	0.0478	74.03
000	23	0.0342	61.87
00	29	0.0282	66.40
0	38	0.0204	60.09
1	48	0.0159	58.25
2	54	0.0131	50.04
3	58	0.0124	51.72
4	62	0.0116	51.72
5	66	0.0107	49.87
6	74	0.0096	50.46
7	82	0.0082	45.21
8	86	0.0072	38.34
9	97	0.0068	43.51
10	109	0.0058	39.97
11	116	0.0052	36.38
12	125	0.0049	36.00
13	129	0.0045	33.70
14	139	0.0039	29.39
15	150	0.0036	29.17
16	157	0.0035	30.20
17	163	0.0031	25.53
18	166	0.0028	21.60
19	169	0.0029	24.02

[*] *Trans. Am. Inst. Min.*, **19**, 580 (1909–1910).

INDUSTRIAL SCREENS

By C. P. Cabell

Screens may be divided into five main classes, grizzlies and stationary screens, trommels and reels, shaking screens, vibrating screens, and oscillating screens. Grizzlies, trommels, and shaking screens are customarily used for separations above about 1 in., but vibrating screens are now competing in this field. Reels are used for flour and similar free-flowing materials, but capacity and efficiency are considered low. The main interest of the chemical engineer is in separations from about 4 mesh to 325 mesh, a field which is almost exclusively held by vibrating and oscillating screens.

Grizzly. A grizzly consists of a set of parallel bars held apart by spacers through which rods pass that hold the bars at some predetermined interval. Figure 3 shows a typical grizzly and the shape of bar most commonly used in its construction. Owing to the wear of the bars they are frequently made of manganese steel (12 per cent Mn). A grizzly is frequently used before a primary crusher in rock- or ore-crushing plants to remove the fines before the ore or rock enters the crusher.

In Table 5 are shown some of the sizes of grizzlies for sale by manufacturers.

One of the chief disadvantages of a grizzly is the tendency of the cross pieces to retard the flow of material and to cause clogging of the openings. Certain engineers have partially avoided this difficulty by placing the cross pieces at a considerable distance from the parallel bars which make up the screen surface.

Table 5. Various Sizes of Grizzlies*

Size, ft.		Openings, in.	Weight, lb.
Width	Length		
3	8	1	1125
3	8	1½	955
3	8	1¾	865
4	8	½	1950
4	8	1	1505
4	8	1½	1235
4	10	1	1885
4	10	1¼	1645
4	10	1½	1540
4	10	2	1365

* From Catalog 1, Allis-Chalmers Co.

The size of the grizzly depends on the capacity desired. The slope, or angle with the horizontal, of the grizzly varies with the type of material between 20 and 50 deg.

Flat grizzlies in which the parallel bars are in a horizontal plane are used on tops of ore and coal bins and under unloading trestles. Inclined grizzlies are more generally used, and the slope of these varies with the angle of repose of the material and the velocity at which the material strikes the grizzly. In general it may be said that moist material requires a greater slope of grizzly bar than dry material and that the larger the size of the material fed to the grizzly, the greater the slope, and vice versa.

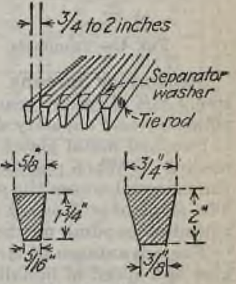


FIG. 3.—Grizzly.

While the simple horizontal or sloping grizzly is still the most generally used, a number of improved types provided with mechanical features are now available and used to a certain extent in the metallurgical and mineral aggregate fields. At the Copper Queen concentrator of the Phelps-Dodge Corporation, a self-cleaning grizzly is used in which the bars are bent to form one-quarter of the circumference of a circle between which heavy metal arms attached to a shaft revolve, preventing oversize from clogging between the bars. In the moving-bar grizzly, the bars are mounted at one end on eccentrics, these eccentrics being 180 deg. apart for adjacent bars. When the eccentric shaft is driven at a speed of about 50 r.p.m., the material is moved gently across the screening surface, and a better opportunity is given for fine material to pass through to a hopper below. In the chain type of grizzly, a series of endless chains passing over sheaves replaces the grizzly bars. Adjacent chains are mounted on sheaves of slightly different diameter, thus tending to break up lumps and to prevent blinding of the open area. The traveling-bar grizzly consists of a multiplicity of short lengths of bar running transversely between two slowly moving sprocket chains. The undersize passes through the upper run of bars and is diverted into hoppers by chutes directly under the upper run and directly above the lower. The distinctive feature of the shaking grizzly is the pivoting of the bars on a horizontal shaft at the upper end and the rapid raising and lowering of the lower ends of the bars by a connecting member actuated by an overhead eccentric.

Capacity. The capacity of a grizzly varies with the efficiency desired and the spacing of the bars as well as its slope. The lower the efficiency desired, the greater the distance between bars; and the greater the slope, the greater the capacity of the grizzly.

The *Eng. Mining J.*, **117**, 307, 1924, is authority for the statement that a grizzly in which the bars are spaced to give 1 in. of clear opening may be expected to handle 100 to 150 tons of material per square foot per 24 hr.

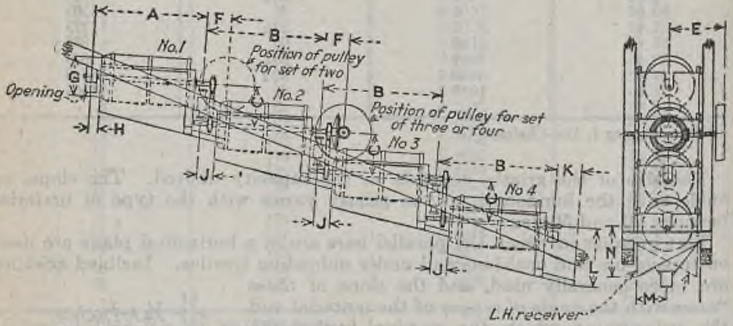


FIG. 4.—Trommels. (*Courtesy Allis-Chalmers Manufacturing Company.*)

Stationary Screens. Stationary screens were formerly used quite extensively but of recent years have not been in such general use due to their low efficiency and capacity and to the head room they require.

Punched metal plates or woven wire screens may be used as the screening medium. When punched metal plates are used, the slope necessary to carry the material across the screen is less than where woven wire screens are used. The slope of stationary screens varies between 20 and 50 deg. from the horizontal, depending on the material to be screened.

The advantages of stationary screens are low initial and repair costs, and ease and speed of installation.

The disadvantages of stationary screens are low efficiency and capacity in the finer sizes of screens, great tendency to blind, and the large amount of head room necessary.

Capacity. The capacity of stationary screens is stated to be from 1 to 5 tons per sq. ft. per 24 hr. per mm. aperture. Woven wire screens of the stationary type are generally mounted at a steeper angle than stationary screens of the punched-plate or bar type.

Revolving Screens. *Simple Trommels.* Figure 4 shows a train of trommels, the feed passing into the upper trommel; the undersize of the first passing to the second trommel and the undersize of the second passing to the

Table 6. Trommels or Revolving Screens—Clearance Dimensions*

Size, in.		Dimensions, in.														Set of two				Set of three				Set of four				R.p.m.		H.p.		
																Pulley		Belt		Pulley		Belt		Pulley		Belt						
Diameter	Length	A	B	C	D	E	F	G	H	J	K	L	M	N	P	Diameter	Face	Width	Ply	Diameter	Face	Width	Ply	Diameter	Face	Width	Ply	Trommel	Center shaft	Set of two	Set of three	Set of four
30	48	74½	76½	27⅞	43¾	38½	18¼	24½	7	9	15⅛	38¾	19½	33¾	5⅛	30	6½	6	4	44	6½	6	4	44	6½	6	4	25	50	2½	3½	4½
30	60	86½	88½	31¹⁄₁₆	43¾	38½	18¼	24½	7	9	15⅛	41½	19½	33¾	5	30	6½	6	4	44	6½	6	4	44	6½	6	4	25	50	2½	3½	4½
36	60	84¼	89½	30¼	49¾	42	18	29¾	7	12½	18¼	44½	22½	39	4¹³⁄₁₆	30	6½	6	4	36	6½	6	4	42	6½	6	4	21	62	2½	3½	4½
36	66	92½	95½	32⅛	49¾	42	18	29¾	7	10	15⅞	45½	22½	39	4¹³⁄₁₆	30	6½	6	4	36	6½	6	4	42	6½	6	4	21	62	2½	3½	4½
36	72	98½	101½	32¼	49¾	42	18	29¾	7	10	16⅛	46¾	22½	39	4½	30	6½	6	4	36	6½	6	4	42	6½	6	4	21	62	2½	3½	4½
36	84	110½	113½	35	49¾	42	18	29¾	7	10	16	48⅞	22½	39	4⁷⁄₁₆	36	6½	6	4	42	6½	6	4	48	6½	6	4	21	62	2½	4	5
36	96	124½	128	37⁷⁄₁₆	49¾	42	19½	29¾	8	11½	17¾	51⅝	22½	39	4¼	36	6½	6	4	42	6½	6	4	48	6½	6	4	21	62	2½	4	5
42	60	84¼	91¼	31¹⁄₁₆	55¾	46	18⁵⁄₁₆	35¾	7	14	20	47½	25½	44¼	4¹³⁄₁₆	36	6½	6	4	36	8½	8	5	42	8½	8	5	18	54	3	4½	6
42	72	96¼	103¼	33⅞	55¾	46	18⁵⁄₁₆	35¾	7	14	20	49½	25½	44¼	4¹¹⁄₁₆	36	6½	6	4	36	8½	8	5	42	8½	8	5	18	54	3	4½	6
42	84	108¼	114¼	34¼	55¾	46	18⁵⁄₁₆	35¾	7	13	18¼	51¾	25½	44¼	4⁵⁄₁₆	36	6½	6	4	36	8½	8	5	42	8½	8	5	18	54	3	4½	6
42	96	124½	128	35⅞	55¾	46	19¹³⁄₁₆	35¾	8	11½	17½	53	25½	44¼	4⅛	42	6½	6	4	42	8½	8	5	48	8½	8	5	18	54	4	5½	7½
48	60	85½	92	33¹⁄₁₆	61¾	49	18⁵⁄₁₆	42	7	13½	19¼	51½	28½	49½	5¹⁄₁₆	36	6½	6	4	36	8½	8	5	42	8½	8	5	16	48	3	4½	6
48	72	97½	104	35	61¾	49	18⁵⁄₁₆	42	7	13½	19¼	53⅞	28½	49½	4¹³⁄₁₆	36	6½	6	4	36	8½	8	5	42	8½	8	5	16	48	3	4½	6
48	84	109½	116	37¹³⁄₁₆	61¾	49	18⁵⁄₁₆	42	7	13½	19½	56	28½	49½	4¹¹⁄₁₆	36	6½	6	4	36	8½	8	5	42	8½	8	5	16	48	3	4½	6
48	96	125¾	129¾	39¹³⁄₁₆	61¾	49	19¹³⁄₁₆	42	8	12	18	58⅝	28½	49½	4⁷⁄₁₆	42	6½	6	4	42	8½	8	5	48	8½	8	5	16	48	4	5½	7½

NOTE: Trommels to be set at a pitch of ¾ in. per ft. for wet work, 1¼ in. for dry.
Supporting timbers to be set at an angle *P* in. per ft. in relation to horizontal line.

* Courtesy Allis-Chalmers Manufacturing Co.

third trommel; thus four products are made with three trommels: oversize of the first; undersize of the first but oversize of the second; undersize of the second but oversize of the third; and undersize of the third trommel.

If only one trommel is used, of course, only two products are made, oversize and undersize.

Trommels are used for both wet and dry screening but have a small capacity for the finer sizes. For sizes coarser than ½ in., where high efficiency is not a necessity, trommels have been found to be satisfactory.

The slopes of trommels vary from ¾ to 3 in. per ft., the smaller slopes being more common. The greater the slope the thinner the bed on the screen and, therefore, the better the chance for the fines to pass through the openings.

Compound trommels are trommels in which two or more concentric screening surfaces are mounted on the same shaft. The coarsest screening surface is the innermost and the finest the outermost, intermediate sizes arranging themselves between these two limits. Provision is made for the removal separately of the oversize from each screening surface, and arrangements are made so that the undersize of each screen becomes the feed to the screen of the next smaller aperture.

Conical trommels have the shape of a truncated cone and are generally mounted with their axes horizontal. Since the diameter of the ends of conical trommels are different, the inclination of the screening surface permits the material to flow through them by gravity without inclination of the axis.

Prismatic trommels may be of any convenient shape but are generally hexagonal. The screening surface consists of a multiplicity of plain surfaces which facilitates the application of screening mediums, particularly if woven wire cloth is used. The sides of the prismatic trommel may be parallel elements, in which case the unit is mounted with its axis slightly inclined to the horizontal; or the sides may be non-parallel elements of a pyramid, in which case the axis is horizontal and gravity flow is obtained as in the conical trommel.

The maximum revolutions per minute for a trommel according to Simons ("Ore Dressing: Principles and Practice," p. 101, McGraw-Hill, New York, 1924) may be obtained by the formula

$$\text{Max. r.p.m.} = \sqrt{\frac{60^2 g}{\pi 2D}} = \frac{77}{\sqrt{D}}$$

where D = diameter of trommel in feet; g = gravitational constant = 32.2; π = 3.1416.

The above formula gives the maximum revolutions per minute of a trommel without resulting in material adhering to the interior screening surface and being carried around by it through centrifugal force. This is also the theoretical optimum speed for a trommel since capacity increases not only with the diameter, slope, and size of aperture, but also with the speed. It may therefore be stated that the efficiency and capacity increase steadily with an increase of speed until the point is reached where the screen substance is carried through a complete revolution by centrifugal force.

Taggart's "Handbook of Ore Dressing" states that the capacity of a trommel is 0.6 ton per sq. ft. per 24 hr. per mm. aperture for dry screening and 1.0 ton for wet screening. Truscott places the capacity for dry screening at 0.5 ton per 24 hr. per sq. ft. per mm. aperture.

Reels are trommels using very fine silk or wire cloth. Many are in use in flour mills. This field has recently been invaded by special types of oscillating screens (such as the Allis-Chalmers gyratory sifter and several others), which are said to offer higher capacity. A stationary vertical screen with interior turbines (Abbe turbo-sifter) is likewise making progress.

Shaking Screens. Shaking screens and shaking grizzles have been used to a considerable extent when it is desired to make a separation and the head room available is not sufficient to permit the use of a vibrating screen. The shaking motion may be such as to make the material advance across the screening medium when the screen is at a slope of not more than ¾ in. per ft.

Shaking screens give a fairly high efficiency in sizes coarser than ½ in., but in the finer sizes the capacity is low if a high efficiency is desired.

The chief disadvantage of shaking screens is the high cost of maintenance both of the screen and of the supporting structure. Shaking screens may be used both for screening and for conveying—this being particularly true in the anthracite-coal industry where innumerable shaking troughs with perforated bottoms are used both for grading the fine anthracite particles and for conveying them to subsequent treatment steps.

Shaking screens require from 0.05 to 0.10 h.p. per sq. ft. screening surface. Speeds range from 60 to 400 strokes per minute with an average of about 100 strokes. These screens are generally inclined at an angle of 18 to 20 deg. with an average of 14 deg., and the length of stroke ranges from 9 in. to a fraction of an inch. A greater number of strokes per minute is generally employed where length of stroke is small and vice versa. Taggart's "Handbook of Ore Dressing" is authority for the statement that capacity ranges from 2 to 8 tons per sq. ft. per 24 hr. per mm. aperture.

Vibrating and Oscillating Screens

Where large capacity and high efficiency are desired, the use of vibrating and oscillating screens is standard practice. The capacity, especially in the finer sizes, is so much greater than any of the other screens that they have practically replaced all other types where the efficiency of the screen is an important factor. An advantage of these screens is that the vibrations of the screen cloth reduce the blinding effect to a minimum. Care should be taken to have sufficient vibration in the coarser sizes.

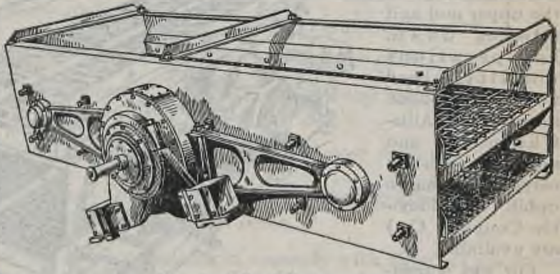


FIG. 5.—The Selectro screen.

Vibrating Screens. A vibrating screen consists essentially of a flat or slightly convex screening surface to which is applied a rapid vibration normal or nearly normal to the surface. The vibrating means may be eccentric shafts, an unbalanced flywheel, a cam and tappet arrangement, or an electromagnet. There are a very great number of vibrating screens on the market, but they can be classified as follows:

Type 1. *High-amplitude Normal Vibration.* In this type screen a high-amplitude (about $\frac{9}{64}$-in.) vibration is applied by a mechanical vibration source (eccentric shafts, unbalanced flywheels, or cam and tappet) operating at about 1200 to 1800 r.p.m.

One of the better known screens employing high-amplitude normal vibration is the Selectro, manufactured by the Productive Equipment Corp., Chicago (see Fig. 5). Vibration is supplied by a horizontal, adjustable, positive eccentric.

Two popular type 1 screens, the Tyler Niagara and the Ty-Rock, are made by the W. S. Tyler Co., Cleveland, Ohio. These are positive eccentric machines with the amplitude of vibration set at the factory. Improvements embodied in the Ty-Rock screen are said to give it longer life for heavy work.

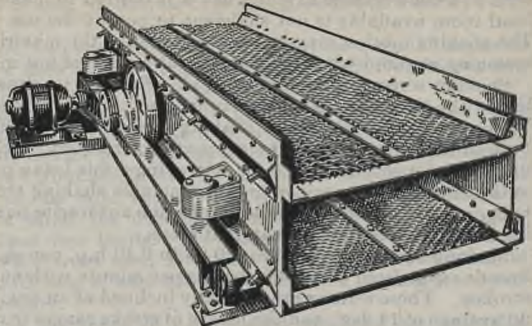


FIG. 6a.—Tyler Niagara screen.

The type 1 vibrating screen is the only vibrating screen used for heavy work (above about 1-in. opening). Special heavy-duty screens are available for such service. In dry screening in chemical work, they are normally used for screening from 1 in. down to about 35 mesh. The screen usually operates at an angle of about 20 deg., with feed entering at the upper end and traveling down. For operations with little head room, low-head screens (Allis-Chalmers) and screens which carry the charge uphill (Ajax Flexible Coupling Co.) are available.

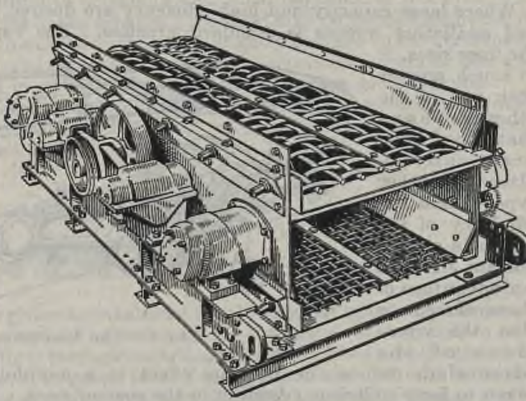


FIG. 6b.—Ty-Rock screen.

For wet screening the type 1 screen is very popular. A low angle (5 to 10 deg.) is used, and feed normally enters at the top of screen. However, in many operations it is found desirable to feed the slurry at the lower end of the screen and have the dried material discharge at the top.

Type 2. *Low-amplitude Normal Vibration.* This type is the electric screen. A low-amplitude vibration is applied normal to the screen cloth by electromagnets at a frequency of 1800 to 7200 vibrations per minute, the low frequency for coarser screening (8 mesh) and the high for fine screening below about 80 mesh. A typical screen is the Hum-mer (see Fig. 6) made by the W. S. Tyler Co.

These screens are not normally used for screening any coarser than about 8 mesh. They are, however, used for separations at 100 mesh and finer. For such fine work two or more small screens are used in series.

Type 2 screens have been very successful in wet screening to remove 1 to 2 per cent of a slurry. They are not recommended for heavy materials having high separable-solids content.

Capacity of Vibrating Screens. Table 7 shows the actual tonnage handled in various plants per hour per square foot of screening surface using a vibrating screen.

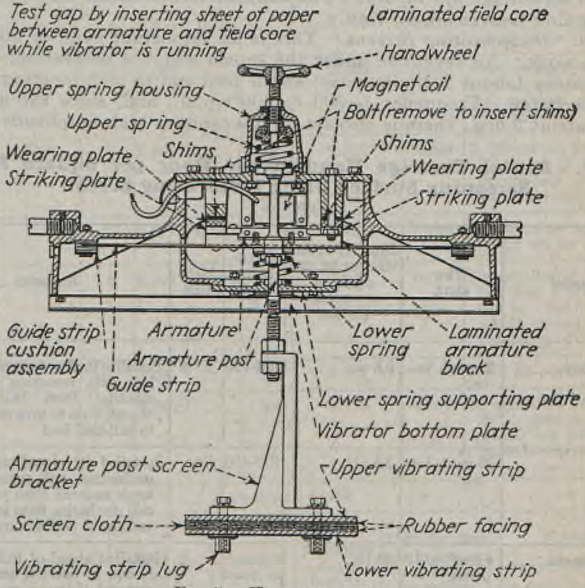


FIG. 7.—Hum-mer screen.

The figures show the great variations in capacity with different materials as well as, in some cases, similar materials, which is the reason why many screen manufacturers do not publish capacity figures. It is to be noted that the tonnages in Table 7 are not necessarily the maximum which could be handled but are simply the actual tonnages put over a certain screen in plant operation.

Oscillating Screens. This type screen is characterized by low-speed (300 to 400 r.p.m.) oscillation in a plane essentially parallel to the screen cloth.

Screens in this group are usually used from ½ in. to 60 mesh. Some light, free-flowing materials, however, can be separated at 200 to 300 mesh. Silk cloths are often used.

Type 1. *Sifters.* A sifter is a boxlike machine, with a series of screen cloths nested atop one another. Oscillation, supplied by eccentrics or counterweights, is in a circular or near-circular orbit. In some machines a supplementary whipping action is set up. Most devices of type 1 have an auxiliary vibration caused by balls bouncing against the lower surface of the screen cloth.

The sifter is a relatively new development which is just entering the chemical field. It is used for dry-screening light materials. Typical machines are made by the Wolfe Co. and the Allis-Chalmers Co.

Type 2. Riddles. A riddle is a screen driven in an oscillating path by mechanism attached to the sole support of the screen, a vertical bar extending from the top of the screen box.

The riddle is the cheapest screen on the market. It is intended normally for batch screening, wet or dry. One supplier is the Great Western Manufacturing Co., Leavenworth, Kan.

Type 3. Reciprocating Screens. This is an important type of screen for chemical work. An eccentric under the screen supplies oscillation, ranging from gyratory (about 2-in. diameter) at the feed end to reciprocating motion at the discharge. Frequency is 500 to 600 r.p.m., and, since the screen is inclined about 5 deg., there is also set up a secondary high-amplitude normal

Table 7. Actual Tonnage Handled per Hour per Square Foot of Screening Surface Using a Vibrating Screen

Cement

Material	Wire cloth	Capacity undersize per hr. per sq. ft. of screening surface	Circulating load, %	Moisture	Remarks
Cement slurry........	20-mesh ton-cap	0.6 ton	100–150	33%	Installed in closed circuit with wet ball mill, removing −20-mesh material from ball-mill discharge, fines to tube mill, tailings to ball-mill feed
Clinker..............	20-mesh ton-cap	2.5 bbl.	100–150	Dry	Installed in closed circuit with preliminary mill, removing −20-mesh material from preliminary-mill discharge, fines to tube mill, tailings to preliminary mill feed
Finished cement......	4 mesh to 12 mesh	20 bbl.	....	......	Installed ahead of Fuller-Kinyon pumps or babbing machines for removal of foreign material

Chemicals

Material	Wire cloth			Capacity, tons per hr. per sq. ft. of screening surface in		Moisture	Remarks
	Mesh	Opening, in.	Wire diam., in.	Feed	Oversize		
Powdered soap............	7	0.096	0.047	0.100	15 to 50% depending upon weather	Varies	Bulky material
Trisodium phosphate........	20 35	0.0320 0.0151	0.018 0.0135	0.065	Oversize, ⅕ Intermediates, ⅗ Fines, ⅕	.... Dry	Hard to screen; therefore low capacity

Table 7. Actual Tonnage Handled per Hour per Square Foot of Screening Surface Using a Vibrating Screen—(*Continued*)

Clay

Material	Wire cloth				Capacity, tons per hr. per sq. ft. of screening surface	Condition of material	Remarks
	Mesh	Opening, in.	Wire diam., in.	T.C.			
Clay...............	4	0.196	0.054	...	0.169	Damp	About 2% of undersize remains in the oversize
Clay...............	10	0.070	.....	740	0.500	Damp	3⅓% oversize; 6⅓% fines. A considerable percentage of undersize remains in oversize
Clay...............	8	0.098	.....	599	0.525	Varies	0.125 ton per hr. oversize; 0.4 ton per hr. fines. Approximately 2% undersize in oversize
Clay...............	10	0.077	0.023	...	0.625	Varies	0.3125 ton per hr. oversize; 0.3125 ton per hr. fines
Clay...............	8	0.093	0.032	...	0.650	Varies	Oversize 0.25–0.275 ton per hr., fines 0.375–0.40 ton per hr. Feed is coarse, which accounts for high percentage of oversize
Clay...............	35	0.020	.....	523	0.125	....	0.075 ton per hr. tailings
Clay...............	40	0.0168	.....	166	0.150	Damp	80% product over screen; 20% product through screen; 3% undersize remains in oversize

Coal and Coke

Material	Kind of plant	Separate made, in.	Maximum size of feed, in.	Capacity, tons per hr. per sq. ft. of screening surface	Products	
					% Over	% Through
Coal...................	Tipple	1¼	3	5.0	30	70
Coal...................	....	¾	2	4.7	35	65
Coal...................	....	¼	1¼	3.0	50	50
Coal...................	Dry-cleaning plant	⅛	½	2.0	30	70
Coal...................	....	1/16	¼	1.5	40	60
Coke...................	Coke plant	1	2	1.5	60	40
Coke...................	Gas plant	⅝	1⅝	1.3	50	50
Coke...................	....	⅛	⅜	1.0	60	40
Coke...................	Coke plant	⅛	⅝	0.5	50	50

Table 7. Actual Tonnage Handled per Hour per Square Foot of Screening Surface Using a Vibrating Screen—(*Continued*)

Crushed Stone and Limestone

Material	Wire cloth				Capacity, tons per sq. ft. of screening surface	Product, tons per hr. per sq. ft. of screening surface		Moisture	Remarks
	Mesh	Opening, in.	Wire diam., in.	Ton cap.		Over	Through		
Crushed stone........	..	1½	¾ rod	....	7.50	2.5	5.0	Dry	5% undersize remains in oversize
Crushed stone........	3	0.265	.....	1017	2.50	Oversize, 1.25 Intermediates, 0.625 Fines, 0.625		Dry	
	10	0.070	.....	740	....				
Limestone............	..	⅜	0.027	....	3.75	65%	35%	Varies	5% undersize remains in oversize
Hydrated gypsum.....	6	0.095	0.072	....	1.0–1.75	2%	98%	Dry	

Metallic Ores

Material	Wire cloth				Capacity, tons per sq. ft. of screening surface	Product, tons per hr. per sq. ft. of screening surface		Moisture	Remarks
	Mesh	Opening, in.	Wire diam., in.	Ton cap.		Over	Through		
Chrome ore..........	10	0.080	.....	38	0.275	0.075	0.2	Varies	
Concentrated iron ore	.....	0.506	0.244	....	3.75	1.75	2.0	Dry	
Flue dust............	1 in.	1	.....	1455	2.5	.....	...	Dry	5 to 1% undersize remains in oversize

Minerals and Non-metallic Ores

Material	Wire cloth			Capacity, tons per hr. per sq. ft. of screening surface	Product, tons per hr. per sq. ft. of screening surface		Moisture	Remarks
	Mesh	Opening, in.	Wire diam., in.		Over	Through		
Insulating material:								
1st surface............	10	0.065	0.035	0.15	....	....	Varies	3 to 5% undersize remains in oversize
2d surface............	12	0.051	0.032					
3d surface............	16	0.0375	0.025					
Powdered enamel.........	30	0.0198	0.0135	0.4	5%	95%	Dry	
Talc....................	55	0.0087	0.0095	0.5	0.3	0.2	Dry	

Table 7. Actual Tonnage Handled per Hour per Square Foot of Screening Surface Using a Vibrating Screen—(*Concluded*)

Sand and Gravel

Material	Wire cloth			Capacity, tons per hr. per sq. ft. of screening surface	Product tons per hr. per sq. ft. of screening surface		Moisture	Remarks
	Mesh	Opening, in.	Ton cap.		Over	Through		
Sand	7⁄16 in.	0.445	1397	2.0	0.75	1.25	Varies	
Sand and gravel	6	0.151	767	0.750	0.15	0.60	Varies	
Sand:								
1st surface	14	0.059	566	0.600	50%	50%	Dry	
2d surface	20	0.041	365					
Foundry sand	24	0.030	520	0.60–0.80	....	....	Wet	Less than 1% of undersize remains in oversize

Stock Yards and Packing-house By-products

Material	Wire cloth				Capacity, tons per hr. per sq. ft. of screening surface	Oversize, tons per hr.	Fines, tons per hr.	Moisture	Remarks
	Mesh	Opening, in.	Wire diam., in.	Ton cap.					
Tankage	8	0.078	0.047	....	1.00–1.25	0.625–0.815	0.375–0.438	Varies	
Tankage	9	5⁄8	5⁄8-in. backing cloth	....	0.20–0.25	0.1–0.125	0.1–0.125	Varies	Approximately 5% undersize remains in oversize
Tankage	4	0.192	....	1167	0.333	0.133	0.2	Greasy	5% undersize remains in oversize
Meat scrap and bone crackling	5	0.120	0.080	....	0.20	0.15	0.05	Greasy	5% undersize remains in oversize
Add phosphate	3	0.253	0.080	....	1.0–1.5	0.25–0.5	0.75–1.0	....	5% undersize remains in oversize
Commercial fertilizer	3	0.0253	0.080	....	1.75	3%	97%	Varies	

Sugar

Material	Wire cloth			Capacity, tons per hr. per sq. ft. of screening surface	Product, tons per hr. per sq. ft. of screening surface		Remarks
	Mesh	Opening, in.	Wire diam., in.		Over	Through	
Sugar	12	0.060	0.023	0.502	20%	80%	

vibration of about $\frac{1}{10}$ in. Further vibration is caused by balls bouncing against the lower surface of the screen cloth.

This type screen is very popular in this country and England. It is used for a variety of chemicals, usually dry, and has been employed for very fine separations (300 mesh) when screening some light materials such as aluminum powder. A typical machine is the Rotex, made by the Orville-Simpson Co., Cincinnati, Ohio (see Fig. 8). Wet screening with this type machine is not common, but at least one manufacturer claims successful operation.

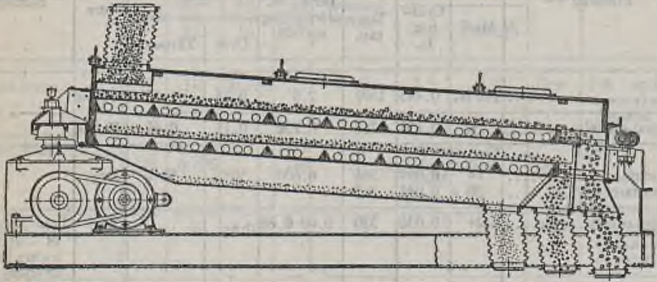


FIG. 8.—Rotex screen.

SIEVE TESTING

Many specifications now call for definite sizes of material and the use of test sieves is required.

Test sieves are very generally used to determine the efficiency of screening devices and the work of crushing and grinding machinery. However, it has been recently shown [Weber and Moran, *Ind. Eng. Chem.*, anal. ed., **10**, 180 (1938). MacCalman, *Ind. Chemist*, **13, 14, 15** (1937–1939)] that sieves meeting present specifications often do not give accurate analyses. Calibration of sieves against a standard set is one means of getting more accurate results.

It is essential that standard screens, with standard siz of opening, be used for sieve analysis. The time of screening and the method of agitating the material on the screen should also be standard, and in many industries the practice of specifying the size of opening (or a certain standard screen) and the time and the method of screening are specified. There are at least three standard screen-scale sieves in use in the United States. In 1939, the A.S.T.M. and the A.S.A. adopted a single standard specification, A.S.T.M. Designation E-11-39, A.S.A. Z 23.1–1939.

The Tyler Standard Screen-scale Sieve. This standard has as its base a 200-mesh screen in which the opening is 0.0029 in. and the wire diameter 0.0021 in. This screen has been adopted by the U. S. Bureau of Standards as the standard 200-mesh screen. In the Tyler standard, the width of opening for each succeeding screen is increased by the square root of 2 or 1.414; thus the area of the succeeding opening is twice that of the preceding screen (see Fig. 2).

Table 8 shows the Tyler standard screen-scale sieves, Catalogue 48, The W. S. Tyler Co., Cleveland, Ohio.

Mechanical Shakers. The Ro-Tap shaker manufactured by the W. S. Tyler Co. is the standard machine for carrying out automatically sieve-test procedure with accuracy and dependability. This device consists of a receptacle to hold a series of superimposed standard sieves and a motor-driven shaker which imparts to the sieves both a circular and a tapping motion.

Table 8. The Tyler Standard Screen-scale Sieves

1	2	3	4	5	6	7	8	9
Tyler standard screen scale $\sqrt{2}$ or 1.414 openings, in.	Every other sieve from 0.0029 to 0.742 in. ratio of 2 to 1	Every other sieve from 0.0041 to 1.050 in. ratio of 2 to 1	Every 4th sieve from 0.0029 to 0.742 in. ratio of 4 to 1	For closer sizing sieves from 0.0029 to 1.050 in. ratio $\sqrt[4]{2}$ or 1.189	Openings, mm.	Opening in fractions of an inch (approximate.)	Mesh	Diameter of wire, in.
1.050	......	1.050	......	1.050	26.67	1	.....	0.148
				0.883	22.43	7/8	.....	0.135
0.742	0.742	......	0.742	0.742	18.85	3/4	.....	0.135
				0.624	15.85	5/8	.....	0.120
0.525	......	0.525	......	0.525	13.33	1/2	.....	0.105
				0.441	11.20	7/16	.....	0.105
0.371	0.371	......	......	0.371	9.423	3/8	.....	0.092
				0.312	7.925	5/16	2½	0.088
0.263	......	0.263	......	0.263	6.680	1/4	3	0.070
				0.221	5.613	7/32	3½	0.065
0.185	0.185	......	0.185	0.185	4.699	3/16	4	0.065
				0.156	3.962	5/32	5	0.044
0.131	......	0.131	......	0.131	3.327	1/8	6	0.036
				0.110	2.794	7/64	7	0.0328
0.093	0.093	......	......	0.093	2.362	3/32	8	0.032
				0.078	1.981	5/64	9	0.033
0.065	......	0.065	......	0.065	1.651	1/16	10	0.035
				0.055	1.397	....	12	0.028
0.046	0.046	......	0.046	0.046	1.168	3/64	14	0.025
				0.0390	0.991	....	16	0.0235
0.0328	......	0.0328	......	0.0328	0.833	1/32	20	0.0172
				0.0276	0.701	....	24	0.0141
0.0232	0.0232	......	......	0.0232	0.589	....	28	0.0125
				0.0195	0.495	....	32	0.0118
0.0164	......	0.0164	......	0.0164	0.417	1/64	35	0.0122
				0.0138	0.351	....	42	0.0100
0.0116	0.0116	......	0.0116	0.0116	0.295	....	48	0.0092
				0.0097	0.246	....	60	0.0070
0.0082	......	0.0082	......	0.0082	0.208	....	65	0.0072
				0.0069	0.175	....	80	0.0056
0.0058	0.0058	......	......	0.0058	0.147	....	100	0.0042
				0.0049	0.124	....	115	0.0038
0.0041	......	0.0041	......	0.0041	0.104	....	150	0.0026
				0.0035	0.088	....	170	0.0024
0.0029	0.0029	......	0.0029	0.0029	0.074	....	200	0.0021
For Coarser Sizing—3 to 1½-in. Opening								
						3	.....	0.807
						2	.....	0 192
						1½	.....	0.148

The Ro-Tap is equipped to handle from 1 to 13 sieves at a time and is equipped with a timing element which automatically terminates the test after any predetermined time.

Table 9. The I. M. M. Series*

Mesh	Diameter wire, in.	Aperture, in.	Aperture, mm.	Ratio of each to one below	Screening area, %
5	0.1	0.1	2.540	1.61	25.00
8	.063	.062	1.574	1.24	24.60
10	.05	.05	1.270	1.20	25.00
12	.0417	.0416	1.056	1.33	24.92
16	.0313	.0312	0.792	1.25	24.92
20	.025	.025	.635	1.25	25.00
25	.02	.02	.508	1.205	25.00
30	.0167	.0166	.421	1.17	24.80
35	.0143	.0142	.416	1.135	24.70
40	.0125	.0125	.317	1.25	25.00
50	.01	.01	.254	1.20	25.00
60	.0083	.0083	.211	1.17	24.80
70	.0071	.0071	.180	1.145	24.70
80	.0063	.0062	.157	1.24	24.60
90	.0055	.0055	.139	1.10	24.50
100	.005	.005	.127	1.51	25.00
120	.0041	.0042	.107	1.273	25.40
150	.0033	.0033	.084	1.32	24.50
200	.0025	.0025	.063	.....	25.00

* The Institute of Mining and Metallurgy, England.

Table 10. U. S. Sieve Series

Meshes per lineal inch	Sieve No.	Sieve opening, in.	Sieve opening, mm.	Wire diameter, in.	Wire diameter, mm.
2.58	2½	0.315	8.00	0.073	1.85
3.03	3	0.265	6.73	0.065	1.65
3.57	3½	0.223	5.66	0.057	1.45
4.22	4	0.187	4.76	0.050	1.27
4.98	5	0.157	4.00	0.044	1.12
5.81	6	0.132	3.36	0.040	1.02
6.80	7	0.111	2.83	0.036	0.92
7.89	8	0.0937	2.38	0.0331	0.84
9.21	10	0.0787	2.00	0.0299	0.76
10.72	12	0.0661	1.68	0.0272	0.69
12.58	14	0.0555	1.41	0.0240	0.61
14.66	16	0.0469	1.19	0.0213	0.54
17.15	18	0.0394	1.00	0.0189	0.48
20.16	20	0.0331	0.84	0.0165	0.42
23.47	25	0.0280	0.71	0.0146	0.37
27.62	30	0.0232	0.59	0.0130	0.33
32.15	35	0.0197	0.50	0.0114	0.29
38.02	40	0.0165	0.42	0.0098	0.25
44.44	45	0.0138	0.35	0.0087	0.22
52.36	50	0.0117	0.297	0.0074	0.188
61.93	60	0.0098	0.250	0.0064	0.162
72.46	70	0.0083	0.210	0.0055	0.140
85.47	80	0.0070	0.177	0.0047	0.119
101.01	100	0.0059	0.149	0.0040	0.102
120.48	120	0.0049	0.125	0.0034	0.086
142.86	140	0.0041	0.105	0.0029	0.074
166.67	170	0.0035	0.088	0.0025	0.063
200	200	0.0029	0.074	0.0021	0.053
238.10	230	0.0024	0.062	0.0018	0.046
270.26	270	0.0021	0.053	0.0016	0.041
323	325	0.0017	0.044	0.0014	0.036

Another mechanical shaker is the Newark End-shak, made by the Newark Wire Cloth Co., Newark, N. J. Sieves used are Newark test sieves, made to conform with the U. S. Standard series.

Interpretation. Sieve analyses are most readily interpreted by plotting them on special types of paper which give simple curves. Logarithmic-probability paper is quite commonly used. A number of methods are described by Austin [*Ind. Eng. Chem.*, anal. ed., **11**, 334 (1939)]. To his list should be added the method of Weinig [*Colo. School Mines Quart.*, **28**, No. 3 (1933)] and that of Roller [*J. Franklin Inst.* **223**, (1937)].

Discussions of methods of particle-size measurement are given by Heywood [*Engineering*, **146**, 192 (1938)] and Work [*Chem. & Met. Eng.*, **45**, 247 (1938)].

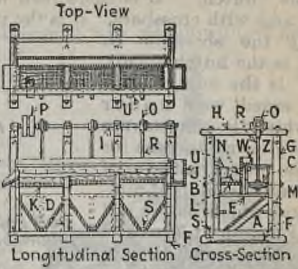


FIG. 9.—Harz jig.

JIGGING

By K. H. Donaldson and J. L. Gillson

REFERENCES: Vedensky, Use of Jigs in Placer Mining, *Miner*, May, 1938. British Patents 438,888 of 1936 to S. Dawson Ware; and 452,664 of 1937 to W. Ruoss. Oke, A Simple Hydraulic Jig, *Mining Mag.*, **54**, 207–208 (1936). Teddy, Jigs on Tin Dredges, *Bull. Dredging Assoc. Southern Malaya*, rev. in *Mining Mag.*, April, 1935, pp. 244–248. Hardy-Smith, Jigs, *Proc.* Australasian Inst. *Mining & Met.*, No. 105, pp. 1–52 (1937).

Jigging is one of the oldest processes used in separating heavy minerals from lighter gangue. A generation ago it was used widely for concentrating base-metal ores and for washing coal. Jigs were simple in operation, could be constructed locally with a low first cost and low maintenance. Power and water consumption are high and tailings losses are in general higher than in other processes, as a result of which the use of jigs, except in placer gold operations and in placer tin and tungsten deposits, is being discontinued. Many older mills, particularly in the Joplin, Mo., lead-zinc district, still employ jigs as do many of the small producers of barytes ores in the Southeastern United States.

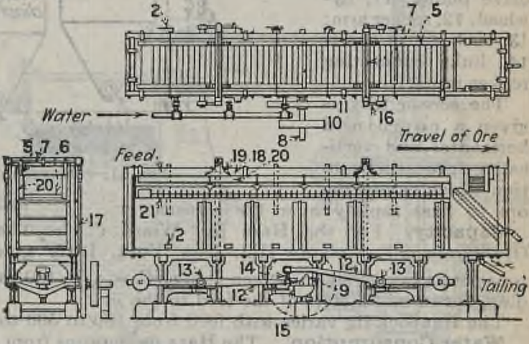


FIG. 10.—Hancock jig.

There are two principal types of jigs. Type 1 is where the sieve is stationary and water is forced up through the screen. One of these types of jigs is called the *Harz jig* and is shown in Fig. 9 (Simons, "Ore Dressing: Principles and Practice," McGraw-Hill, New York, 1924).

A is a tank the upper portion of which is divided in two sections, plunger and sieve, by the partition marked *E*. The lower portion of the tank is called the "hutch." *B* is the screen which supports the mineral. *K* is a wooden frame with crossbars. *C* is the plunger which is given a reciprocating motion by the eccentric *O*. *S* is the hutch spigot. *V* is the cup or submerged weir under which the concentrate is allowed to pass for the separation of heavy from light mineral which passes out of the jig at *T*.

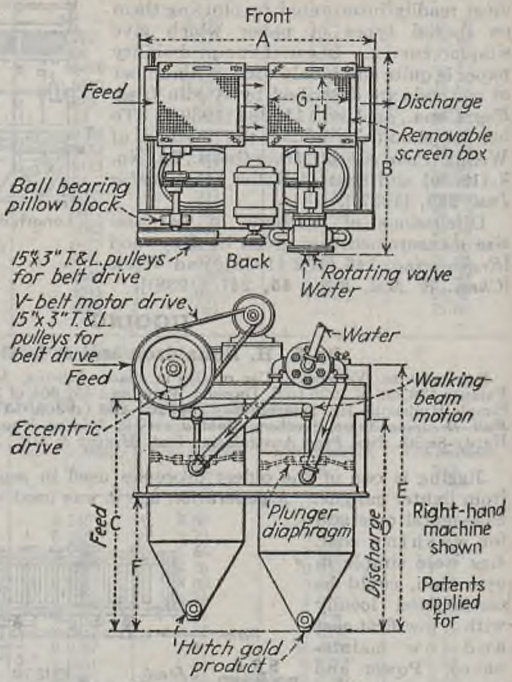


FIG. 11.—Denver mineral jig.

The second type of jig is where the sieve moves up and down in a tank of water, the one shown in Fig. 10 being a Hancock jig.

In Fig. 10: 2 is the cup for concentrate or middlings; 5 is the tray to carry the screens; 6 and 7 are supports for the tray; 8 is cam shaft; 9, three-point cam; 10, drive pulley; 11, flywheel; 12, rocker arm; 13, rocker-arm shaft; 14, links connecting rocker arms.

The screen tray is given a combined horizontal and vertical reciprocating motion which causes the ore to pass rapidly over the screens.

Capacity. For the Harz jig, Wiard ("The Theory and Practice of Ore Dressing," McGraw-Hill, New York, 1915) gives the capacity C in tons per 24 hr. per sq. in. screen area as $C = \sqrt{d/100}$ tons where d = average diameter in millimeters of the feed to the jig.

The Hancock jig varies with feed from 100 to 500 tons per 24 hr.

Water Consumption. The Harz jig requires from 10 to 20 tons of water (2350 to 4700 gal.) per ton of solid. Hancock jigs usually require about half as much water as the Harz jig.

Power. Harz jig power consumption depends on length and number of strokes per minute and upon the class of material being treated as well as the depth of bed.

Simons ("Ore Dressing: Principles and Practice," McGraw-Hill, New York, 1924) gives the following figures which may be taken as a preliminary estimate of the horse power required.

For a one-compartment jig	1–2 h.p.
For a two-compartment jig	2 h.p.
For a three-compartment jig	2½ h.p.
For a four-compartment jig	3 h.p.

Jig Feed. Jigs formerly treated material from a 2-in. size down to 2 mm., but in modern practice the minimum usually would be coarser than 3 or 4 mesh. The fine material which was formerly treated in jigs would now be treated on tables.

Jigs for Placer Gold. The commercial utilization of jigs in the recovery of placer gold started about 1914 and resulted in the design of some new jigs which may have some place also in coarse roughing of other heavy minerals. The concentration of placer gold by jigging is successful because of the extreme difference in specific gravity between the gold and the gangue materials. The high capacity of these new jigs recommends them for treating the low-grade material that must be handled in tremendous quantity in most placer operations. The use of jigs in this field was described by Malozemoff [Jigging Applied to Gold Dredging, *Eng. Mining J.*, **138,** 34–37 (1937)] and jigs are made by the Pan-American Engineering Corp., Berkeley, Calif., The Southwestern Engineering Co., Los Angeles, and the Denver Equipment Co., Denver. Diagrams showing the construction of these jigs are shown in Figs. 11, 12, and 13.

FIG. 12.—Southwestern-Kraut Hydromotor jig.

The Denver mineral jig (Denver Equipment Co.) has a number of special features which have been particularly adapted to conditions found in the grinding circuit, *i.e.*, dilution control and the handling of unsized feed. These units are easy to operate and are built in four sizes to handle practically any tonnage.

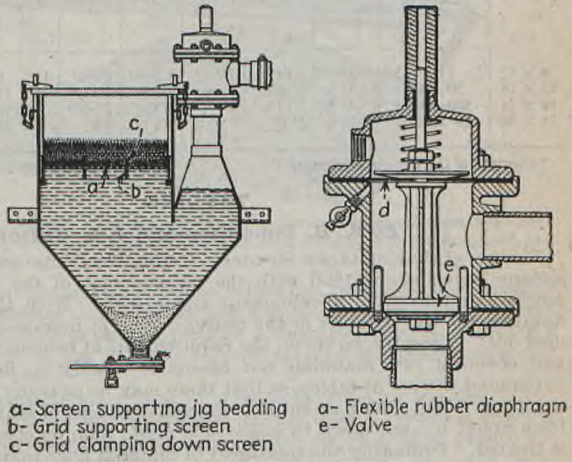


FIG. 13.—Pan-American jig.

This jig consists of two compartments and is of all-steel, welded construction, to minimize space required. In the front of the unit are the two screen compartments over which the entire mill discharge passes; in the rear are the plungers actuated by a walking beam and adjustable eccentric. The plungers are sealed with special rubber diaphragms to give positive displacement of water or solution. A rotating valve is synchronized with the movement of the walking beam so that solution is added only on one part of the stroke of the plungers to counteract the suction normally created. This gives a water action, closely approximating the perfect theoretical jigging action, with pulsations upward and periods of free settling between, with minimum suction. The speed of different jig sizes will vary according to the material being treated; average speed is from 300 to 350 r.p.m. and a stroke of approximately ¼ in.

The screen compartments are removable, as they must be cleaned up at 10- to 60-day periods, similar to any jig operation. A spare compartment is furnished so that the change can be made without shutting down the grinding circuit. A special non-blinding screen is used which practically eliminates clogging.

The metallic gold values and the high-grade concentrates are finer than the screen openings and pass into the hutch below, which has steeply sloping sides, thus allowing clean discharge of this valuable high-grade material. The discharge plug of each hutch can be securely locked to prevent theft. The tonnage shown below is that of the grinding-mill initial feed, the jig also handling the circulating load.

Table 11. Denver Mineral Jig Dimensions and Data*

Size, in.	Capacity, tons	*A*	*B*	*C*	*D*	*E*	*F*	*G*	*H*	Motor h.p.	Shipping wt., lb.	
											Belt	Motor
8 × 12	15–45	2′11″	2′11″	2′10″	2′7¼″	3′3⅝″	1′6¼″	12″	8″	¾	900	975
12 × 18	50–200	4′ 0″	3′ 6″	3′ 8⅝″	3′5⅛″	4′3⅛″	2′1¾″	18″	12″	1	1500	1625
16 × 24	200–500	5′ 0″	4′ 3″	3′10⅛″	3′4⅝″	4′3⅜″	2′3¼″	24″	16″	1½	1950	2050
24 × 36	500–1200	7′ 1″	6′ 0″	5′ 0″	4′5″	5′5″	3′6″	36″	24″	2	3000	3150

* Courtesy of the Denver Equipment Co.

TABLING

By K. H. Donaldson and J. L. Gillson

The use of shaking tables for concentrating heavy materials from worthless gangue began about 1896 with the development of the Wilfley table and reached its maximum development about 1910. With the introduction of flotation the use of tables in the concentration of base-metal sulfides rapidly died out. Recently, however, the requirements of beneficiation of many ores and chemical raw materials not amenable readily to flotation has again stimulated the use of tables, so that there may be as many in use today as at any time. Wet tables are suitable for concentration of materials in the range from about 6 mesh down to nearly 300 mesh, and, on coal, coarser material is treated. Ordinarily the treatment of material finer than 48 mesh is not so economical as flotation. The objection to tables as a method of concentration lies in the large area required for the installation of the necessary number of tables to treat fine material in a mill of large capacity and because differential separation of the several heavy minerals that occur together in many of the base metal ores is not possible.

Except where there is a large difference between the specific gravities of the materials to be separated (galena and limestone or galena, sphalerite, and limestone), it is not possible to produce both a clean concentrate and a clean tailing. Under ordinary conditions tables can be operated to produce either a clean concentrate and a middling or a rough concentrate and a clean tailing. Where the table is used to produce a clean concentrate, not over a 50 per cent recovery of the valuable mineral may be expected if only one tabling operation is used.

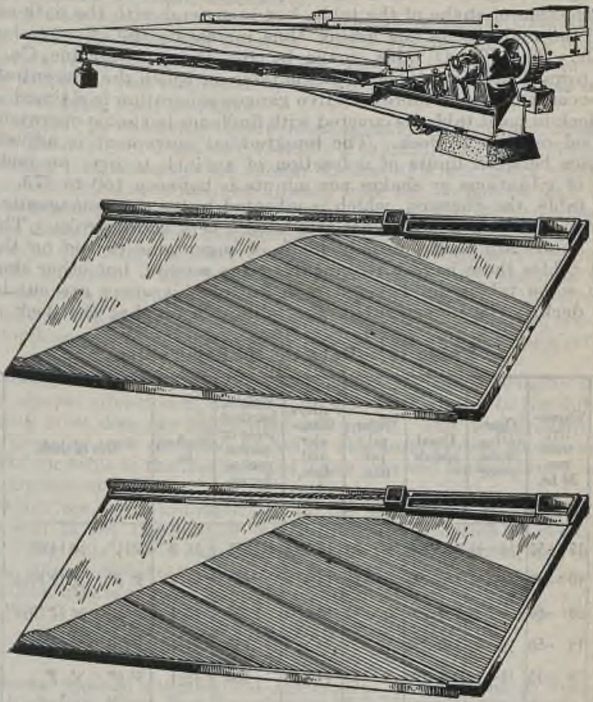


Fig. 14.—Deister-Overstrom diagonal deck table. Center, diagonal deck with pool riffle system for sand. Bottom, diagonal deck with pool riffle system for fine sand and slime.

In many plants classifiers are used to prepare the feed for the tables, but in some plants a deslimed but unclassified feed is used to produce either a medium-grade concentrate or a clean tailing which may markedly reduce the necessary crushing and/or grinding.

The heaviest grains are least affected by the cross current of water being trapped behind the riffles, lighter minerals are washed over the riffles and are discharged either at the side of the table or at the end near the low side. Launders are distributed along the side and end of the table to catch the discharge; usually a concentrate, a middlings, and a tailings are collected separately.

The shape of the deck varies with the design of the manufacturer, and few are exactly rectangular. The Deister-Overstrom table is elongated diagonally to the direction of shake of the table, but in general with the path of movement of the majority of the grains, thus increasing the effective length of the table. The Plat-O table, made by the Deister Machine Co., has a plateau toward the discharge end of the table on which the concentrate must climb, because of which more effective gangue separation is claimed.

The deck in most tables is covered with linoleum but some operators prefer a redwood or cement deck. The longitudinal movement is adjustable in most cases between limits of a fraction of an inch to over an inch. The number of vibrations or shakes per minute is between 150 to 375. In one English table, the Sherwen, which is vibrated by an electromagnetic device, the vibration is at a much higher rate with a very short stroke. The riffles in most tables are cleats, usually made of sugar pine tacked on the deck. In most tables these have a rectangular cross section, but other shapes are used; in some tables not covered with linoleum, grooves are cut into the wooden deck. The movement of the table is such that the deck reverses its direction with a maximum velocity at one end and a minimum velocity at the other end of the stroke. It is the quickness of the return which causes the material on the table to migrate toward the discharge end. This motion is given by a pitman and toggles, running in oil in a dust-tight box, except in the case of the electrically vibrated Sherwen table.

Table 12. Data on Wilfley Tables

Table No.	Capacity, tons per 24 hr.	Operating horse power	Usual speeds	Water, gal. per min.	Pulleys, diameter and face, in.	Weight, heaviest piece or pieces packed for export, lb.	Strokes, in.	Size of deck	Type of under structure
6	15 –50	½– ¾	240–290	5 –20	14 × 4	835	½–1	5′11″ × 14′9″	Steel frame
6 (special)	10 –40	½– ¾	240–290	5 –20	14 × 4	675	½–1	5′ 3″ × 12′10½″	Steel frame
9	20 –80	1 –1½	235–250	10 –40	18 × 4	1.130	½–1	5′ 3″ × 12′10½″	Steel frame
11-D (Type 6)	15 –50	½– ¾	240–290	5 –20	14 × 4	850	½–1	5′11″ × 14′9″	Concrete
12	5 –15	¼– ½	240–290	3 –15	14 × 4	300	½–1	3′ 6″ × 7′	Timber frame
13	½– 2	⅛– ⅙	300–375	½– 4	6 × 1	220	¼– ½	1′ 1½″ × 2′6″	Steel frame
14	15 –50	½– ¾	240–290	5 –20	14 × 4	890	½–1	5′11″ × 14′9″	Timber frame

The tables used most generally in commercial work have a deck from 10 to 15 ft in length and have a capacity varying from 2 to 5 tons per hr. with coarse material (and up to 10 tons per hr. on coal) and ½ to 1 ton per hr. with finer material. The installed horse power is between 1 to 3 but the running load is, in general, only half of the starting load. Water requirements average 10 and 15 gal. per min. per table.

The approximate direct cost of treating ores by tabling varies from a minimum of about 2½ cts. per ton to about 5 cts. per ton of feed, depending upon labor and power rates and the number of cleanings necessary. The installed cost of a table of standard size will run between $1000 and $2000 per table, depending upon location and the complexity of the equipment installed to handle feeds and discharges.

Vanners were extensively used in concentrating plants treating sulfide minerals before the advent of flotation, but in sulfide mills they have been largely replaced by flotation. They are used to some extent to recover minerals which are not recovered by the flotation process such as oxidized minerals of copper, lead, and zinc.

Dry Tables. A relatively recent development is the Sutton, Steele, and Steele dry table (U. S. Patents 1,574,637; 1,632,520; and 2,137,678), which has a shaking motion somewhat similar to that of a wet table, except that the direction of motion is inclined upward from the horizontal and, instead of water acting as the medium of distribution, a blast of air is driven through a perforated deck. The table has application in cases where it is desirable to treat material dry, either because of water shortage or because it is undesirable to wet the materials. The table supplements other dry methods of concentration such as electrostatic and electromagnetic methods. An advantage of the table is the ability to handle material coarser than that treated on most wet tables. Ores as coarse as ¼ in. and coal as coarse as 3 in. can be treated.

Close sizing is necessary to give good results, and until recently this has militated against adoption of the table for fine sizes, owing to the difficulties of screening most ores dry below about 40 mesh. The development of dry methods of sizing with the wind tunnel or Schramm system should promote adoption of the table to the treatment of many ores and materials now handled by other equipment.

Dry tables are used commercially in the separation of many types of minerals and in the cleaning of industrial materials such as seeds, cork, bagasse, fiber, nuts, wood chips, coffee, etc. One interesting use is in the sorting of silicon carbide by grain shapes. Flat and splintery grains are removed from others of more nearly equal dimensions.

AGGLOMERATION TABLING

By O. C. Ralston

REFERENCES: Ralston, Flotation and Agglomerate Concentration of Non-metallic Minerals, *U. S. Bur. Mines, Rept. Investigations* 3397, May, 1938, pp. 42–48. Diener, Clemmer, and Cooke, Beneficiating Cement Raw Materials by Agglomeration and Tabling, *U. S. Bur. Mines, Rept. Investigations* 3247, 1935, 6 pp. Coghill, DeVaney, Clemmer, and Cooke, Concentration of Potash Ores of Carlsbad, N. M., by Ore Dressing Methods, *U. S. Bur. Mines, Rept. Investigations* 3271, 1935, 13 pp. Part 2, Lawrence and Roca, Flotation of Low-grade Phosphate Ores, *U. S. Bur. Mines, Rept. Investigations* 3105, 1931, 9 pp. Selective Oiling and Table Concentration of Phosphatic Sands in the Land Pebble District of Florida, *U. S. Bur. Mines, Rept. Investigations* 3195, 1932, 6 pp. Laboratory Studies of the Trent Process, *U. S. Bur. Mines, Rept. Investigations* 2263, 1921, 18 pp. Ralston, Comparison of Froth with Trent Process, *Coal Age*, **22,** 911–914 (1922). O'Meara, Norman, and Hammond, Froth Flotation and Agglomerate Tabling of Feldspars, *Bull. Am. Ceramic Soc.*, **18,** 286–292, (1939).

Agglomeration tabling is a process whereby selective flocculation or agglomeration of grains of one mineral in an aggregate is caused by the addition of an agglomerating agent in a conditioning cell or in the ball-mill circuit, the slurry containing the agglomerated grains then being fed across gravity tables. The larger size and feathery texture of the floccules cause them to be washed over the side of the table by the current of cross water, while the unflocculated discrete particles remain on the table and are carried off the end in the position followed normally by the concentrate in the usual table feed. An oiled particle will tend to ride on the surface of the water and thus is more readily carried across the side of the table than an unoiled particle. Agglomeration tabling has had more application in the concentration of phosphate minerals than in any other field, although successful tests have been run on limestone, potash, and other ores. The process is applicable to materials of coarser sizes (20 to 100 mesh) than can be concentrated by froth flotation (60 to 500 mesh). Another advantage over flotation is in the cases where the mineral that can be readily agglomerated is the dominant mineral in the crude slurry. If the bulk of the material has to be lifted by flotation, there is a tendency to degrade the concentrate by entrainment of gangue in the bulky concentrate. Such entrainment does not occur so readily in agglomeration tabling, and, by filming the dominant mineral, the capacity of tables can be greatly increased. The process is not adaptable to selective concentration of more than one produce in a mixture. Oil is used as one of the principal agglomerating agents and the film of oil remaining on the product is objectionable in some industries.

ELUTRIATION

By K. H. Donaldson and J. L. Gillson

Wet Elutriation. Elutriation is a term applied usually to complete hydraulic classification of a weighed quantity of material done for the purpose of determining accurately the grain sizes, particularly in the range below the finest screen size. Three types of elutriation apparatus are illustrated.

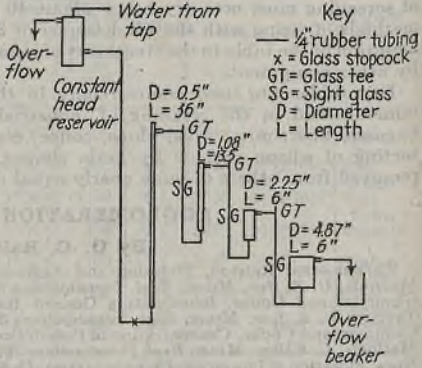


FIG. 15.—Elutriator designed by V. W. Haedrich, E. I. du Pont de Nemours & Co.

The Andrews kinetic elutriator is slightly more complicated than the other two but is said to be extremely accurate.

Elutriation is continued for a number of hours until no particles can be seen in the overflow tubes from one vessel to the next. The vessels are then emptied and the contents filtered, dried, weighed, and then examined microscopically for size, using a calibrated grid ocular or an ocular with an accurate scale reading to hundredths of a millimeter. It may be unnecessary to describe the grain sizes, referring simply to the fractions collected with a given hydraulic velocity. The weight of the several fractions collected shows the size distribution in the sample tested. Any fraction that is lost in the final overflow is recognized by the difference in total weight of the fractions as compared with the original weight.

A study of elutriation and a description of equipment, particularly of an accurate and extremely slow feeding device, are described by Clemmer and Coghill [Improved Laboratory Elutriator and Its Application to Ores, *Eng. Mining J.*, **129**, 551–554 (1930)]. Further results of investigations on elutriation are described by Cooke [Short-column Hydraulic Elutriator for Subsieve Sizes, *U. S. Bur. Mines, Rept. Investigations* 3333, pp. 39–51 (includes a bibliography)]. A description of the Andrews kinetic elutriator was given by Andrews (A New Elutriator, *Mining Mag.* (London) May, 1929, p. 301; April, 1936, pp. 232–233) and repeated later by Curtis (A New Elutriator, *Sands, Clays & Minerals*, January, 1933, pp. 59–63).

An accurate knowledge of size distribution of subscreen sizes is extremely important in the manufacture of pigments, in the dressing of clays, and in the flotation of ores.

A continuous "elutriator" for commercial use in wet tabling of mineral products is shown in Fig. 17. It is adapted especially for preparing middlings products from wet tables for further cleaning but can be used for cleaning concentrates or scavenging tailings products. The dimensions will vary according to the quantity of feed to be handled.

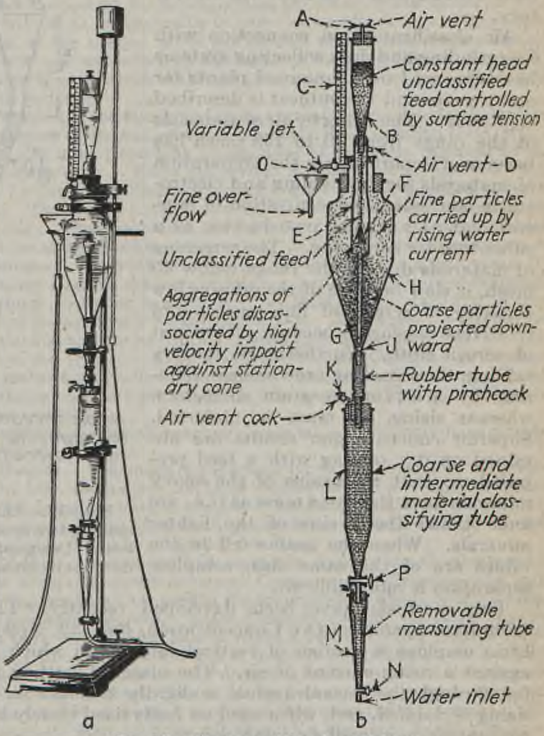


Fig. 16.—Andrews Kinetic elutriator.

Dry Elutriation and Dry Classification by Air. Equipment for and the theory of dry elutriation are described by Traxler and Baum (Determination of Particle Size Distribution in Mineral Powders by Air Elutriation, *Rock Products*, **37**, 44–47 (1934)). It is necessary in the size analysis of materials subject to hydration, such as cement, lime, or solution in the case of water soluble materials. For dry elutriation, complete dispersion of the particles must be accomplished, which in the Traxler and Baum apparatus is done by means of an air jet. Temperature and humidity conditions are controlled and maintained constant throughout each test.

Roller Particle-size Analyzer. An instrument very good for free-flowing material in the range 5–80μ is the Roller particle size analyzer, made by the American Instrument Co., Silver Spring, Md. [see Roller, *U. S. Bur. Mines, Tech. Paper* 490 (1931)]. By a controlled stream of dried air the device separates a given sample into fractions having predetermined particle-size ranges, such as 0 to 10μ, 10 to 20μ, 20 to 40μ, and 40 to 80μ. It has been used with success in at least one laboratory.

Air classification in connection with fine grinding and dust collecting systems has been used in commercial plants for many years and the subject is described on p. 1932. The sizing by air of materials in the range from 30 to 150 mesh has become of importance in the preparation of materials for dry tabling and electrostatic methods of concentration of mineral and chemical products, as a substitute for screening. The screening of materials dry, in the range below 40 mesh, is slow because of the relative low capacity of screen of fine meshes and relatively expensive because of the cost of screen cloth. Furthermore, for dry tabling, the screening process sizes materials by maximum grain diameters, whereas sizing by mass is preferred. Superior concentration results are obtained on dry tabling with a feed prepared, so that the grains of the heavy minerals have the same mass as (*i.e.*, are finer than) the grains of the lighter minerals. When the grains fed to the tables are of the same size, complete separation is more difficult.

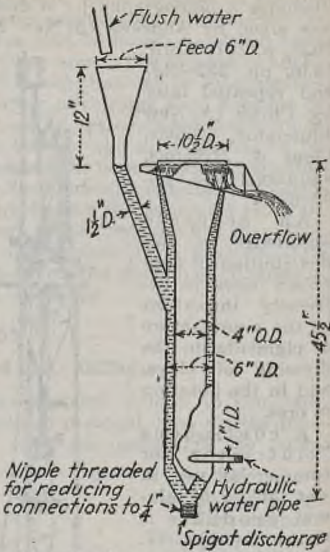


FIG. 17.—Laboratory elutriator for making two products, using dry or damp feed. Designed by J. L. Gillson, E. I. du Pont de Nemours & Co.

Two methods have been developed recently. The Schramm system (Schramm System of Ore Concentration, *Bull.* 22, New York Concentrators, Ltd.) employs a number of vertical columns in which the material fed falls against a rising current of air. The material settling in the first column is fed through the second against a slightly stronger air current. Very close sizing is claimed, and, when used on feeds sized closely by previous screening, concentration as well as sizing occurs.

Adaptation of the wind tunnel to air sizing has been made recently in the soil-analysis laboratory of the California Institute of Technology. Although designed originally for analysis of size distribution in soils, it is built for continuous operation.

MAGNETIC SEPARATION

By J. L. Gillson

The separation, by magnetic methods, of minerals, metals, and other materials is a highly developed art. There are many machines designed specifically for particular kinds of feeds. An outline of the types of machines that are in common use is as follows:

I. For removal of ferromagnetic or other strongly magnetic materials.

   A. When the amount of strongly magnetic material forms only a small part of the feed.

      1. *Dry Methods.* Magnetic pulleys, or suspended poles over moving belts or inclined chutes to remove "tramp iron," magnetite, or other magnetic materials present in the product, or purposely, or unavoidably introduced. Permanent or electromagnets are placed below chutes in some installations, as in flour mills to cause stray bits of iron fragments to remain on the chutes, which are cleaned periodically. Some agricultural products are cleaned by mixing them with iron filings. Seeds with rough surfaces to which the filings adhere are removed from others with smooth surfaces.

      2. *Wet Methods.* The removal of small amounts of iron-bearing impurities from slurries of ceramic products, pigments, dyes, flotation feeds, etc., is one of the most important applications of magnetic methods to the chemical industry. There are several machines on the market. One made by the S. G. Frantz Company of New York is called a Ferro-Filter. The Ding's Company of Milwaukee makes one known as the De-ironer. These machines have an electromagnet. Screens through which the slurry flows are magnetized, and the iron particles adhere to the screens. To avoid interrupting the flow of the slurry when the screens are cleaned, the installations are made in duplicate or triplicate. The slurry feeds for a given interval through one unit and is then switched automatically to the second unit. The current is interrupted from the first and the screens are cleaned by flushing water through them which is run to waste. The Magnetic Products Company of Trenton, N. J. makes a unit called a Separmag. The "screen" is a box of tacks, or stainless steel attractors, which are magnetized by an Alnico permanent magnet.

   B. When the amount of strongly magnetic material is a relatively large proportion of the feed.

      1. *Dry Methods.* Belt machines are used for dry separation of magnetite (Fe_3O_4) from gangue but the large volume of such a strongly magnetic material jumping toward the poles causes so much entrainment of non-magnetic grains that clean products are difficult to produce, and dry methods are to be avoided if possible.

      2. *Wet Methods.* Suspended belt machines, such as the Crockett, made by the Ding's Magnetic Separator Company of Milwaukee, and magnetic "log washers" are used in the concentrating plants of many mines producing magnetic ores. In the Crockett machine the material is fed into a tank in the upper part of which a belt is moving parallel to the length of the tank. A number of magnets are placed above the belt. The material in the feed is well dispersed in the water and the magnetic grains adhere to the underside of the belt and are moved along into another division of the tank and beyond the action of the magnets. The non-magnetic material flows out through a discharge in the bottom of the first section of the tank.

         These machines have very large capacities per unit, up to 35 or more tons per hour per machine. They are suitable only for separating very strongly magnetic materials.

II. For removal of moderately magnetic or weakly magnetic materials.

   A. *Dry Methods.* There are two types of machines available. A belt-type machine, such as the Wetherill, is adapted only to the separation of minerals that are moderately magnetic such as ilmenite, franklinite, and pyrrhotite. Some operators prefer the belt machines for these separations, particularly in cases where the desired mineral is associated with another mineral that is itself magnetic. They believe the selectivity is greater.

   The material is fed into the machine on a belt and under successive magnets. Cross belts run under these magnets and above the feed belt. The magnetic grains are lifted against the force of gravity and adhere to the underside of the cross belt which carries them laterally out of the field of the magnet.

These belt machines have a low capacity per dollar of installed cost and are not used in many plants. A large machine with an installed cost of $10,000 will handle only about one ton of feed per hour.

Rotor type machines have large capacities and can remove grains with a very low magnetic permeability, such as mica from feldspar, iron-stained grains from sands or fluorspar, rutile from zircon, etc.

Rotor-type machines are made ordinarily with a multiplicity of rotors in a tier. The upper rotors are in a field of very weak magnetism while a much stronger field is applied on the lower rotors. The capacity of these machines depends on the length of the rotor, on the grain size of the material and the degree of selectivity required. On feeds of material of plus 100 mesh, a machine with 30-in. rotors will handle from 2 to 3 tons per hr. Most machines are made with a double tier of rotors, the magnets being activated by the same coils. Such machines are made by the Ding's Magnetic Separator Company and by the Stearns Company, both of Milwaukee, and by the Exolon Company of Blasdell, N. Y.

B. Wet Methods. Magnetic ring-and-drum machines for wet feeds have been built in Germany and Sweden but are not used to any extent in the United States. In fact, as yet there seems to be no satisfactory wet separator for continuous operation on the United States market for treating only moderately strong magnetic materials.

III. For extremely weak paramagnetic or for "diamagnetic materials."

Only one machine has been built for separating exceedingly weakly magnetic materials from grains that are diamagnetic. This is the Frantz Isodynamic Magnetic Separator, made by the S. G. Frantz Company of New York; the present models of this separator have capacities of only 50 to perhaps 200 lb. per hr. Nevertheless, the separations of supposedly "non-magnetic" materials that can be made on this machine are rather remarkable. The machine is essentially a laboratory unit and has not yet been brought to commercial development.

IV. Special magnetic machines and processes.

Hundreds of special machines have been built, among which those operating with an alternating current are perhaps the most interesting. Few, if any, are in commercial use.

In general, it may be said that wet methods are applicable to material of finer grain size than are dry methods, but then only for the separation of strongly magnetic grains. Dry feeds must be granular, free flowing, and dust free.

There is probably no material that is truly non-magnetic, *i.e.*, with absolutely zero magnetic permeability. Methods and equipment of magnetic separation through a wide range of permeabilities and grain sizes are available. The limiting factor is that the magnetic forces must be strong enough to overcome other forces acting on the same grains, such as gravity, air currents, and surface tension.

FROTH FLOTATION

By J. L. Gillson

General References: Buchanan, Chemical Tools of Flotation, *Mining Met.*, **11**, 565–570 (1930). Taggart, Taylor, and Ince, Experiments with Flotation Reagents, *Trans. Am. Inst. Mining Eng.*, **87**, (1930); *Am. Inst. Mining Eng., Tech. Pub.* 204, 75 pp. Sproule, Some Fundamentals of Flotation, *Can. Mining J.*, **57**, 582–588 (1936). Whiting, Modern Flotation Reagents, Their Classes and Uses, *Mining Met.*, **19**, 185–188, (1938). Bassett, A Review of Recent Flotation Patents, *Can. Mining J.*, **58**, 185–187 (1937); May, 1937, pp. 255–256; June, 1937, pp. 304–306; July, 1937, pp. 356–359. Shorey, Patek, and Roland, Desliming Ore Pulps with Sodium Silicate as a Deflocculator, *Am. Inst. Mining Eng., Tech. Pub.* 559, Sept. 15, 1934, 14 pp. Gillies, The Story of the Bubble, *Can. Inst. Mining Met.*, 1935, pp. 349–354. Wark, Principles of Flotation, *Australasian Inst. Mining Met.*, **103**, 346 (1938). Gaudin, "Flotation," McGraw-Hill, pp. 552, 1932. Giudice, Depressors and Protective Agents in Conditioning for Flotation, *Eng. Mining J.*, **135**, 350–353 (1934). Giudice, Fundamentals of Flotation in the Light of Recent Research, *Eng. Mining J.*, **135**, 152–155 (1934). Giudice, Collection in Flotation, *Eng. Mining J.*, **135**, 213–217 (1934). Flotation Reagents, Eastman

Kodak Co., *Synthetic Organic Chemicals*, **17** (1931). Taggart, Taylor, and Knoll, Chemical Reactions in Flotation, *Am. Inst. Mining Eng., Tech. Pub.* 312, February, 1930, 33 pp. Wark and Cox, An Experimental Study of the Influence of Cyanide, Alkalis and Copper Sulfate on the Effect of Sulfur-bearing Collectors at Mineral Surfaces, *Am. Inst. Mining Eng., Tech. Pub.* 574, October, 1934, 38 pp.; Conception of Adsorption Applied to Flotation Reagents, *Am. Inst. Mining Eng., Tech. Pub.* 732, February, 1937, 16 pp. Wark and Sutherland, Influence of the Anion on Air-mineral Contact in Presence of Collectors of Xanthate Type, *Mining Tech.*, November, 1939, 23 pp. Welsch, Chemical Activity the Cause of Flotation, *Eng. Mining J.*, **133**, 529–532 (1932). Shepard, Experiments on the Cause of Bubble Attachment in Flotation, *Mining Met.*, **17**, 339 (1936). Gaudin, Groh, and Henderson, Effect of Particle Size on Flotation, *Am. Inst. Mining Eng., Tech. Pub.* 414, May, 1931, 23 pp. Brighton, Burgener, and Gross, Depression by Cyanide in Flotation Circuits, *Eng. Mining J.*, **133**, 276–278 (1932). Ince, A Study of Differential Flotation, *Am. Inst. Mining Eng., Tech. Pub.* 195, February, 1929, 23 pp. Wark and Cox, An Experimental Study of the Effect of Xanthates on Contact Angles at Mineral Surfaces, *Am. Inst. Mining Eng., Tech. Pub.* 461, February, 1932, 48 pp. Ralston and Hunter, Activation of Sphalerite for Flotation, *Am. Inst. Mining Eng., Tech. Pub.* 248, October, 1929, 15 pp. Ralston, King, and Tartaron, Copper Sulfate as Flotation Activator for Sphalerite, *Am. Inst. Mining Eng., Tech. Pub.* 247, October, 1929, 113 pp. Weinig and Carpenter, The Trend of Flotation, *Colo. School Mines Quart.*, **32,** 189 pp. (1937). Kraeber and Boppel, Uber die Wirkung von Metallsalzen beider Schwimmaufbereitung oxydischer Mineralien, *Metall Erz*, **31,** 417–427 (1934). Weinig and Carpenter, The Trend of Flotation, *Colo. School Mines Quart.*, **32** (No. 40), 60–64 (1937). Norman and Ralston, Conditioning Surfaces for Froth Flotation, *Am. Inst. Mining Eng., Tech. Pub.* 1074, published in *Mining Tech.*, May, 1939, 16 pp.

NON-SULPHIDE FLOTATION: Clemmer and O'Meara, Flotation and Depression of Non-sulphides—Calcite, Silica and Silicates, Fluorspar, Barite, Apatite and Tungsten Minerals, *U. S. Bur. Mines, Rept. Investigations* 3239, pp. 9–26, June, 1934. Dean, Clemmer, and Cooke, Use of Wetting Agents in Flotation, *U. S. Bur. Mines, Rept. Investigations* 3333, pp. 3–20, February, 1937. Halbich, Uber Neuartige Schwimmittel, *Metall Erz*, **30** (No. 21), 1–4 (1933) (use of alcohol sulfates). Growing Use of Flotation for Non-metallic Minerals, *Mining Met.*, March, 1935, pp. 129–130. Beitrag zur flotation Nicht sulfidischer Mineralien, *Metall Erz*, **27,** 527 (1930). Coghill and Clemmer, Soap Flotation of the Non-sulfides, *Am. Inst. Mining Met. Eng., Tech. Pub.* 445, 1932, 18 pp.; *Eng. Mining J.*, **133,** 136–137 (1932). Cullen and Lavers, Flotation as Applied to the Chemical Industry, *Trans. Inst. Chem. Eng.* (London), Jan. 15, 1936, reviewed in *Mining J.* (London), **192,** 65–67, 85–86, 104–105, (1936). Dean and Hersberger, New Flotation Reagents, *Am. Inst. Mining Eng., Tech. Pub.* 605, 1935. Patek, Relative Flotability of the Silicate Minerals, *Trans. Am. Inst. Mining Eng.*, **112,** 486–508 (1934); *Am. Inst. Mining Met. Eng., Tech. Pub.* 564, 22 pp. Patek, Colloidal Depressors in Soap Flotation, *Eng. Mining J.*, **137,** 558 (1936). Dean, Clemmer, and Cooke, Use of Wetting Agents in Flotation, *U. S. Bur. Mines, Rept. Investigations* 3333, pp. 3–19, February, 1937. Taggart, Flotation Application to Non-metallics, *Eng. Mining J.*, **137,** 90–91 (1936). Ralston, Flotation and Agglomerate Concentration of Nonmetallic Minerals, *U. S. Bur. Mines, Rept. Investigations* 3397, May, 1938.

Alunite: Ishikawa, Flotation of Alunite, Japanese Patent 100,390, April 4, 1933.

Apatite: Luyken and Bierbrauer, Flotative Recovery of Apatite, *Metall Erz*, **26,** 197–202, 202–203 (1929); *Mitt. Kaiser-Wilhelm Inst. Eisenforsch. Düsseldorf*, **10,** 317–321 (1928).

Arsenic: Yasyukevich and Kahn, Experiments on Scorodite Flotation, *Tsvetnuie Metall*, 1934, No. 7, pp. 26–35.

Barite: Timm, *et al.*, Barite-bearing Mill Tailing from Kamloops Homestake Mine, Ltd., Jamieson Creek, B. C., *Can. Dept. Mines, Geol. Branch, Rept.* 774, pp. 80–86, 1937. The Flotative Separation of a Mixture of Barite and Dolomite—the Differential Flotation of Earth Alkali Minerals, *Berg- u. Hüttenmänn. Jahrb.*, **81,** 139–146 (1933). Hammann, Flotation of Heavy Spar—a Preview, *Metall Erz*, **30,** 455–457 (1933). O'Meara and Coe, Froth Flotation of Southern Barite Ores, *Am. Inst. Mining Eng., Tech. Pub.* 678, 1936, 6 pp. Glembotskii, Flotation of Barite at the Salair Dressing Plant, *Gorno-Obogatitel. Zhur.*, No. 5, 1937, pp. 13–18. Rankin, Laurence, Davis, Houston, and McMurray, Concentration Tests on Tennessee Valley Barite, *Am. Inst. Mining Eng., Tech. Pub.* 880, 1938, 13 pp.

Barite and Dolomite: Ringe and Bierbrauger, The Flotative Spearation of a Mixture of Barite and Dolomite—the Differential Flotation of Earth Alkali Minerals, *Berg- u. Hüttenmänn. Jahrb.*, **81,** 139–146 (1933).

Bauxite: Lottermoser and Rumpelt, Studies on the Flotation of Bauxite from Near Bodayk (Hungary), *Kolloid Beihefte*, **35,** 372–412 (1932). Ganrud and DeVaney. Bauxite: Sink-and-float Fractionations and Flotation Experiments, *Bur. Mines, Bull.* 312, 1929, 101 pp. Preliminary Examination of Low-grade Bauxite, with Particular Reference to Flotation, *Bur. Mines, Rept. Investigations* 2906, 1928, 6 pp.

Beryl: Gisler, Selective Flotation of Beryl, Master's Thesis, University of Utah, May 15, 1936.

Borax: Flotation of Boric Acid and Borax from the Products of Treatment of the Indera Boracite, *J. Applied Chem.* (*U.S.S.R.*), **10,** 845–852 (1937). A Flotation Process for the Separation of Borax and Boric Acid from a Mixture of Salts, *J. Chem. Ind.* (*U.S.S.R.*), **12,** 277–279 (1935).

Calcium Carbonate: See Limestone.

Cassiterite: Flotation of Cassiterite, *Arch. Erzbergbau Erzaufbereit. Metallhüttenw.*, **2,** 1–20 (1932). Pol'kin and Boldyrev, The Flotation of Cassiterite, *Gorno-Obogatitel. Zhur.*, No. 6, 1936, pp. 38–39.

Clays: Sommer, Die Flotation von Kaolin, *Ber., deut. keram. Ges.*, **15,** 317–323 (1934). Weinhardt, Flotation of Clay, *Ceram. Age*, **24,** 79 (1934). Shaw, Refining of Clay by Film Flotation, *Ceram. Age*, **24,** 43–44 (1934); Clay Refining by Flotation Methods, *Bull. Am. Ceram. Soc.*, **16,** 291–294 (1937).

Coal: Yancey and Taylor, Froth Flotation of Coal, *U. S. Bur. Mines, Rept. Investigations* 3263, 1934, 20 pp. Ergebnisse der selektiven Kohlen flotation auf Kohlenchemischer Grundlage, *Glückauf*, **71,** 101–105 (1935). Goette, Grundlage der Steinkohl flotation, *Glückauf*, **70,** 293–297 (1934). Why and When to Use a Flotation Process, *Coal Age*, **38,** 333–334 (1933); **39,** 375–379 (1933); 1934, pp. 17–20. Van Iterson, Separation of Substances by Flotation, *Proc. Royal Acad. Amsterdam*, **40,** Nos. 2 and 3, 1937. Kuhlwein, Removal of Fusain by Selective Flotation of Coal, *Glückauf*, **70,** 245–252, 275–277 (1934). Yancey and Taylor, Flotation Processes for Cleaning Fine Coal, *U. S. Bur. Mines, Inf. Circ.* 6714, May, 1933, 31 (patent list). Brennstoff, The Separation of Clay from Coal, *Chem.*, **17,** 446–451 (1936). Berthelot, Modern Methods of Washing Coal, Especially Fine Coal, *Prof. 3d Internat. Conference Bituminous Coal*, **2,** 761 (1931); also *Chimie & Industrie*, **30,** 770–786 (1933). Brownlie, Mechanical Separation of the Constituents of Coal, *Steam Eng.*, **3,** 5–6, 44 (1933). Chapman, Recent Progress in Coal Cleaning Practice in Great Britain, *Prof. 3d Internat. Conference Bituminous Coal*, **2,** 741–758 (1931). Edser and Williams, The Cleaning of Coal by Froth Flotation, *Trans. Fuel Conferences World Power Conference, London*, 1928, **1,** 374–384 (1929). Hanot, Treatment of Slimes Containing Coal and Obtained from Steam Coal of High Ash Content, *Congr. Internat. Metal. Geol. Appl.*, 7th sess., Paris, October, 1935; *Mines*, **2,** 459–469.

Feldspar: Gerty, Preparation of German Feldspar Ore by Flotation, *Ber. deut. keram. Ges.*, **17,** 526–534 (1936). Ziergiebel and Gerth, Feldspar Separated from German Rocks by Flotation, *Ber. deut. keram. Ges.*, **15,** 517 (1934). Iverson, Separation of Feldspar from Quartz, *Eng. Mining J.*, **133,** 227–229, (1932). O'Meara, Norman, and Hammond, Froth Flotation and Agglomerate Tabling of Feldspars, *Bull. Am. Ceram. Soc.*, **18,** 286–292 (1939).

Fluorspar: Mitchell, Gross, and Oehler, Froth Flotation of Fluorspar, *Am. Inst. Mining Eng., Tech. Pub.* 999, 1939, 12 pp. Anonymous, War Demands for Fluorspar May Test Capacity of New Plant in Colorado, *Pit and Quarry*, **31,** 48–49 (1939). Eigeles, Flotation of Fluorspar of the Solonetchoie District, *Mineral Suire'e*, **10** (No. 10), 42–45 (1935). Anonymous, Trends in Treatment of Fluorspar Ores, *Gorno-Obogatitel. Zhur.*, 1936, (No. 2) pp. 33–42. Williams and Greeman, Concentrating Fluorspar Ore, U. S. Patent 1,785,992, Dec. 23, 1930. Gerth, Uber die flotation von flusspat und Quartz, ein Beitrag zur flotation polarer Nichterze, *Metall Erz*, **35,** 314–317 (1938). Oxidation Products of Paraffin as Flotation Agents for Fluorite Ores, *Mineral Suire'e*, **9,** 44–54 (1934). Coghill and Greeman, Flotation of Fluorspar Ores from Acid Spar, *U. S. Bur. Mines, Rept. Investigations* 2877, 1928, 3 pp. Sinclair, Fluorspar in South Africa, *Mining Mag.*, **55,** 265–270 (1936).

Graphite: Von Schoen, Flotation of Graphite, *Metall Erz*, **29,** 181–182 (1932). Menardi, Modern Flotation Plant for Graphite, *Rock Products*, **31,** 74–77 (1928). Ganrud, Coe, Benefield, and Skelton, The Flotation of Alabama Graphite Ores, *U. S. Bur. Mines, Rept. Investigations* 3225, 1934, 20 pp. Dub and Moses, Mining and Preparing Domestic Graphite for Crucible Use, *U. S. Bur. Mines, Bull.* 112, 1920, 80 pp. Black Donald Graphite Operating New Hydroelectric Plant, and Concentrator, *Eng. Mining J.*, **128,** 368 (1929). Parsons, The Concentration of Flake Graphite Ores, *Can. Dept. Mines, Mines Branch, Mem. Ser.* 26, 1926.

Gypsum: Keck and Jasberg, A Study of the Flotative Properties of Gypsum, *Am. Inst. Mining Eng., Tech. Pub.* 762, 1927.

Halite: Kuzin, Separation of Halite from Sylvinite by Flotation, *J. Applied Chem.* (*U.S.S.R.*), **10,** 457–469 (1937).

Hematite: Keck, Eggleston, and Lowry, A Study of the Flotation Properties of Hematite, *Am. Inst. Mining Eng., Tech. Pub.* 763, 1937. Adams, Kobey, and Sayres, Flotation of Hematite, a Laboratory Success, *Eng. Mining J.*, **132,** 53–55 (1931).

Iron Ores: Keck and Jasberg, A Study of the Flotative Properties of Magnetite, *Am. Inst. Mining Eng., Tech. Pub.* 801, 1937, 17 pp. Keck, Eggleston, and Lowry, A Study of the Flotative Properties of Hematite, *Am. Inst. Mining Eng., Tech. Pub.* 763, 1937.

Kyanite: O'Meara and Gandrod, Concentration of Georgia Kyanite, *Am. Inst. Mining Eng., Contrib.* 98, 1936, 3 pp. (see also *Tech. Pub.* 605). Concentration of Kyanite from

Death Rapids, B.C., *Can. Dept. Mines, Mines Branch, Bull.* 736, *Rept.* 471, 1932, pp. 238–240.

Limestone: Lee, Flotation of Limestone from Siliceous Gangue, *U. S. Bur. Mines, Rept. Investigations* 2744, 1926, 3 pp. Miller and Breerwood, Flotation Processing of Limestone, *Am. Inst. Mining Eng., Tech. Pub.* 606, 1935.

Manganese: DeVaney and Clemmer, Floating of Carbonate and Oxide Manganese Ores, *Eng. Mining J.*, **128,** 506–508 (1929). Gaudin and Behrens, A Manganese Recovery Problem and Its Solution, *Eng. Mining J.*, **138,** 40–42 (1937). Basmanov, Flotation of Manganese Slimes at Chiaturi, A Preliminary Communication, *Gorno-Obogatitel. Zhur.*, 1936 (No. 3), p. 11.

Magnesite: Doerner and Harris, Concentration of Low Grade Magnesite Ores by Flotation, *Wash. State Coll. Mining Expt. Sta., Bull. P*-1, June, 1938, 33 pp. Sinkinson and Michaelson, Flotation of California Magnesites, *Am. Inst. Mining Eng., Tech. Pub.* 733, 1936. Gerth and Baumgarten, Pretreatment of Magnesite Deposits at Zobten, *Chem. Ztg.*, **60,** 177–178 (1936). See also *Tech. Pub.* 723.

Magnetite: A Study of the Flotative Properties of Magnetite, *Am. Inst. Mining Eng., Tech. Pub.* 801, 1937.

Nepheline: Nephelite and Vanadium Concentrates from Khibin Apatite-nephelite Ore, *Novosti Tekniki Ser. Gornorudnaya Prom.*, **193** (No. 18), 9–10. Smirnov and Sheblovnskii, Concentration Apatite-nephelite Ore of the Khiba District, *Inst. Mekhanischeskoi Obrabotko Poleznuikl Meklanubr*, 1930, Part 1, pp. 184–236.

Nonmetallics: Ralston, The Nonmetallics, *Eng. Mining J.*, **138,** 93–96 (1937). Taggart, Flotation—Application to Nonmetallics, *Eng. Mining J.*, **137,** 90–91 (1936).

Oxide Ores: Wiegon, Use of Aromatic Amino Compounds, U. S. Patent Reissue No. 20,614, Jan. 4, 1938.

Pigments: Merz, Pigment Purification, U. S. Patent 2,144,115, Jan. 17, 1939.

Phosphate: Barr, Development and Application of Phosphate Flotation, *Ind. Eng. Chem.*, **26,** 811–815 (1934). O'Meara and Pamplin, Selective Oiling and Table Concentration of Phosphate Sands in the Land-pebble District of Florida, *U. S. Bur. Mines, Rept. Investigations* 3195, 1932, 6 pp. Lawrence and Roca, Flotation of Low-grade Phosphate Ores, *U. S. Bur. Mines, Rept. Investigations* 3105, 1931, 9 pp. Lawrence and DeVaney, Flotation of Low-grade Phosphate Ores, *U. S. Bur. Mines, Rept. Investigations* 2860, 1928, 4 pp. Belash, Selective Flotation of Phosphate Minerals, *Mineral Suire'e*, **11** (No. 6), 49–56 (1936); **9** (No. 6), 36–44 (1934). Pamplin, Ore Dressing Practice with Florida Pebble Phosphate, *Am. Inst. Mining Eng., Tech. Pub.* 881, January, 1938.

Phosphorites: The Flotation of Phosphorites from the Soch Deposits in Baschkir Republic, *Goruni Zuhr.*, **110** (No. 5), 58–59 (1934).

Physical Chemistry: Wark, The Physical Chemistry of Flotation: VII. Trimethylcetylammonium Bromide as a Flotation Reagent, *J. Phys. Chem.*, **40,** 661–668 (1936).

Potash Ores: Coghill, DeVaney, Clemmer, and Cooke, Concentration of the Potash Ores of Carlsbad, N. M., by Ore Dressing Methods, *U. S. Bur. Mines, Rept. Investigations*, February, 1935, 13 pp. Flotation of Langbeinite from the Potash Field of New Mexico and Texas, *U. S. Bur. Mines, Rept. Investigations* 3300, 1936, 4 pp. S. A. Kunzin, Flotation of Solykamsk Sylvinite Ores, *Kali* (*U.S.S.R*), 1937, pp. 17–27. Russian Patent 41,508, Jan. 7, 1934. Preobrajenski, Outline for the Utilization of Residues from the Working Over of Crude Sylvite, *Jour. Inst. Mech. Working Useful Minerals; Mining Concentrating J.*, (*U.S.S.R*), 1932, (No. 1), pp. 29–30. See Sodium Potassium Chlorides.

Quartz: Gerth and Baumgarten, Flotation of Quartz, *Metall Erz*, **35,** 314–317 (1938).

Scheelite: Gorodetskii, The Flotation of Scheelite, *Gorno-Obogatitel Zhur*, 1937 (No. 4), 37–38.

Selenium: Belash, Flotation of Selenium from Sulphuric Acid Plant Slimes, *Redkie Metal.*, **5** (No. 5), 15–19 (1936).

Silica: Patek, Relative Floatability of the Silica Minerals, *Am. Inst. Mining Eng., Tech. Pub.* 564, 1934; *Trans. Am. Inst. Mining Eng.*, **112,** 486, 508 (1934).

Soap Flotation: Coghill, Soap Flotation, *Eng. & Mining J.*, **135,** 125 (1934).

Sodium Fluoride: Eigeles, Separation of Sodium Fluoride from the Melt by Flotation, *Mineral Suire'e*, **9** (No. 2), 39–45 (1934).

Sodium and Potassium Chlorides: Kuzin, Separation of Sodium and Potassium Chlorides, Sulphates, and Nitrates by Flotation, *J. Applied Chem.* (*U.S.S.R*), **9,** 818–833 (1936) (abs. in French on p. 833).

Sulphur: Hazen, Recovering Sulphur, *Eng. Mining J.*, **127,** 830–831 (1929); Recovery of Sulphur from Surface Deposit, *Mining J.* (*Phoenix, Ariz.*), **13,** 7 (1930).

Talc: Norman, O'Meara, and Baumert, Froth Flotation of Talc Ores from Gouverneur, N. Y., *Bull. Am. Ceram. Soc.*, **18** (No. 8), 202–297 (1939). Carnochan and Rogers, Experimental Tests on Madoc Talc for the Separation of Dolomite, *Can. Dept. Mines, Mines Branch. Bull.* 736, *Rept.* 469, pp. 231–234, 1932. Trauffer, Froth Flotation Economically Recovers Valuable Material from Talc Waste, *Pit and Quarry*, **31,** 28–30 (1939). Talc Flotation, *Ceram. Age*, **27,** 41 (1936).

Talc-magnesite: Clemmer and Cooke, Flotation of Vermont Talc-magnesite Ores. *U. S. Bur. Mines, Rept. Investigations* 3314, 1936, 12 pp.

Tinstone: See Cassiterite.

Tourmaline: Bayula, Concentration of Tourmaline from the Tailings of Kluchevskii Ores, *Novosti Tekhniki, Ser. Gornorudnaya Prom.*, **3** (No. 22), 11–12 (1935).

Tungsten: Gorodetskii, The Flotation of Scheelite, *Gorno-Obogatitel. Zhur.*, 1937 (No. 4), pp. 37–38.

Wolframite: Winkler, Flotation of Tinstone and Wolframite, *Metall Erz*, **32**, 181–183 (1935).

Zircon Ores: Corbett, Concentrating Zircon Ores, British Patent 416,018, Feb. 12, 1934; British Patent 406,043, Feb. 12, 1934; U. S. Patent 2,082,383, June 1, 1937.

General Principles. Flotation is a process whereby the grains of one or more minerals, or chemical compounds in a pulp or slurry, are selectively caused to rise to the surface in a cell or tank by the action of bubbles of air. The grains are caught in a froth formed on the surface of the tank and are removed with the froth, while the grains which do not rise remain in the slurry and are drawn off the bottom of the cell or tank.

Floatability is a property of solids and some solids are more easily flotable than others. Sulfur, graphite, sulfides of the metals are easily floatable, whereas oxides, silica, and silicates are not as readily floatable. Floatability is a surface phenomenon. The nature of the film on the outside of the particle is the controlling factor. The selective filming of grains of one mineral in an aggregate by a specific reagent promotes floatability of these grains in preference to the others.

Reagents which will film certain minerals are known as collectors, those that induce a froth are frothers, those that assist in the selective separation of one solid from another by depressing one, or inhibiting its flotation, are depressers. Reagents which disperse slime coatings on grains, thus favoring filming, are deflocculating agents. Acids and alkalies are added to control the pH. Various inorganic reagents are used for special purposes, particularly for activating or assisting other reagents.

Collectors. The collectors used most commonly in the flotation of the native metals such as gold or copper, the sulfides, arsenides, tellurides, sulfosalts, etc., are the xanthates, dio-thio-phosphates, alpha naphthylamine, thio-carbamates, etc. A curious fact is that many rubber accelerators are collectors of sulfide minerals in froth flotation.

Collectors used for non-sulfide minerals are oleic and other fatty acids and soaps, fatty alcohol sulfates, and mineral and coal oils (usually added in an emulsified form); and for the oxides and silicates the collectors are the so-called "cationic" reagents. Kerosene and coal oils are used as collectors in coal flotation.

Cationic Reagents. Since the development of cationic reagents is new, a description is pertinent.

Lenher in U. S. Patent 2,132,902 of Oct. 11, 1938, recognized that reagents that have the surface-active constituent in the positive ion will flocculate and collect minerals that are not flocculated by the reagents such as oleic acid or soaps, in which the surface active ingredient is the negative ion. In the patent a large number of such chemicals were disclosed, principally quaternary ammonium compounds.

This permitted a classification of the minerals into two groups, positive and negative. Those that are positive presumably carry a positive electric charge on the mineral surfaces, while those that are negative carry a negative charge. The character of the charge is probably relative to the charge of the collecting reagent and is affected by acidity of the pulp and other factors. One of the most important minerals that is negative is the mineral quartz, which in all previous work had been floated, if at all, only with the greatest difficulty. Minerals that are negatively charged may occur together but one that has a stronger charge can be collected away from the other.

The U. S. Bureau of Mines took up the investigation shortly after the Lenher application was filed and found a large number of uses for these so-called cationic reagents. Norman found that short-chain amines such as di-n-butyl amine is an effective collector of minerals containing 5 per cent or more of water. Such minerals are talc, pyrophyllite, sericite, clays, and weathered mica. Anhydrous minerals and those with little water require a longer chain compound such as lauryl or stearyl amine hydrochloride, or stearyl or cetyl tri-methyl-ammonium bromide. In a more recent paper, Norman, O'Meara, and Baumert list a number of reagents in the order of their effectiveness on talc. Results of tests on concentration of magnesite ore with lauryl amine hydrochloride are described by Doerner and Harris. A long list of reagents, some of which are cationic, was described by Dean and Hersberger. These reagents are now in commercial use or in pilot-mill testing on the flotation of talc, of quartz from feldspar, of quartz and mica from limestone, of quartz and other minerals from magnesite, etc.

Kirby found that an amine hydrochloride with about eight carbon atoms is effective on potash ore.

Cleaning mineral surfaces by attrition or by the use of hydrofluoric acid was investigated by Norman and found to improve the degree of separation by flotation. The addition of some inorganic compounds such as sodium hexa-meta-phosphate or tetra-sodium phosphate also improve selectivity.

Frothers. Agents used to produce a foam are cresylic acid, pine oil, and other soft and hardwood oils, and a series of branch-chain alcohols sold under the designation of the "B series" by duPont. They are actually mixtures of alcohols and ketones.

Depressers. Sodium cyanide is used to prevent the flotation of sphalerite (ZnS) and pyrite during the flotation of galena (PbS) in mixed lead and zinc ores. Lime is added also to depress the pyrite; in copper-lead separations, cyanide and lime depress the copper sulfides. Chromate salts are used to depress galena and to permit copper minerals to be floated away from lead.

Deflocculating Agents. Caustic soda or sodium silicate serve to deflocculate slime and clean mineral surfaces.

Activation and Miscellaneous Reagents. Lime is the most common reagent used in sulfide mineral flotation to raise the pH and sulfuric acid when an acid circuit is desired. The consumption of lime for this purpose exceeds 50,000,000 lb. per year. Soda ash is used in the flotation of galena unless pyrite is present, when lime is used. Copper sulfate is added in zinc sulfide flotation as a promoter. After the lead has been removed, copper sulfate and an additional collector are added. In non-sulfide flotation a large number of reagents have been used, including glue, starches, gums, tannins, various phosphates, sodium fluoride, and silico-fluorides. Solvents for some water-insoluble reagents are employed, as are emulsifying reagents for oils, fatty acids, etc. The alcohol sulfates are suitable for this purpose.

Flotation Machines. A large number of machines of many types have been designed. Some are pneumatic, others entirely mechanical, still others are mechanical with compressed air introduced through a porous diaphragm in the bottom of the cell or at the base of a rapidly revolving impeller.

Types of four machines in common use are illustrated.

General Operations. Ores must be ground to a point of complete or nearly complete liberation, and, even though this can be accomplished by coarse crushing, grinding to finer than 20 mesh is necessary in all cases and finer than 48 mesh in most. Grinding is done in closed circuit in ball or rod mills in series with classifiers. Pulp density is important and requires close control. In a few cases warm water is used. The character of the water is important and usually better results are obtained with purer water. Hard water causes slime flocculation and in flotation with soaps and oleic acids increases greatly the reagent consumption. In the flotation of potash salts and other water-soluble chemicals requiring the use of brines, salt-water soaps and brine-resistant reagents, such as Avirol (sodium octyl sulfate), are used as are also certain of the cationic reagents such as lauryl amine hydrochloride.

In some cases conditioning or mixing of reagents and ore is necessary. Usually one or more of the reagents are added in the ball mill to promote thorough mixing, but specially designed "conditioners" are used in some mills.

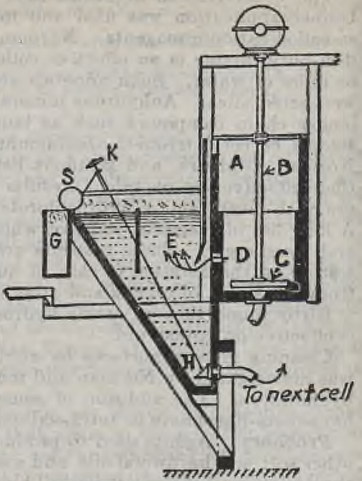


FIG. 18.—Mineral separation flotation machine.

Flotation machines are built in multiple units and the flow of the pulp through various units is adjusted for the best results. Common practice is to feed the pulp to several cells known as "roughers" which produce a barren tailing and a low-grade concentrate. The concentrate is treated, sometimes after regrinding, in "cleaner" cells and "recleaner" cells for final concentration. The tailings from the cleaner and recleaner cells are recirculated back through the system or concentrated separately in additional cells. Regrinding of these middlings is necessary in many ores. The concentrate is usually, but not always, collected in the froth from the cells. In a few cases it is the flotation tailings which is the concentrate desired.

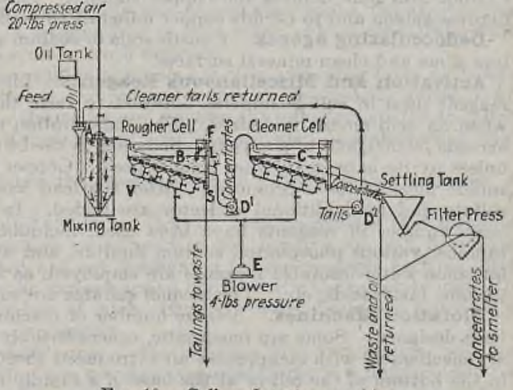


FIG. 19.—Callow flotation machine.

Flotation costs vary enormously with the type of feed. In most cases of sulfide mineral flotation the grinding is the principal item of expense. In large mills, treating thousands of tons per day, the total mill cost varies between 35 to 75 cts. per ton. In small mills, treating more limited tonnages' costs are higher. In non-sulfide flotation the reagent cost may be an item of major expense. The proportion of mineral floated in fluorspar, talc, clay ores, or coals is much larger than in most sulfide ores where the proportion of valuable mineral to gangue is generally very low. Thus the consumption of reagent is greater. Many of the special chemicals required for such purposes are more expensive than the sulfide collectors such as the xanthates and Aerofloats.

The cost of a concentrating plant using flotation of a capacity of 500 tons per day is roughly $750 per ton of daily capacity—smaller mills cost more in proportion and larger mills less. The grinding equipment costs half or more of the total in most mills.

Booth Agitair Flotation Machine (Fig. 20). Air at low pressure is blown into the cell at 9 and must pass between the prongs of the rapidly revolving impeller, causing the air to be divided into extremely fine bubbles. The stabilizer reduces to a minimum any wave action and gives an even distribution of the air throughout the cell area. The advantages of this machine are that a large amount of air is admitted to the pulp with a very small power consumption per cubic foot of air and that there is no porous medium to become clogged with sands or precipitated salts.

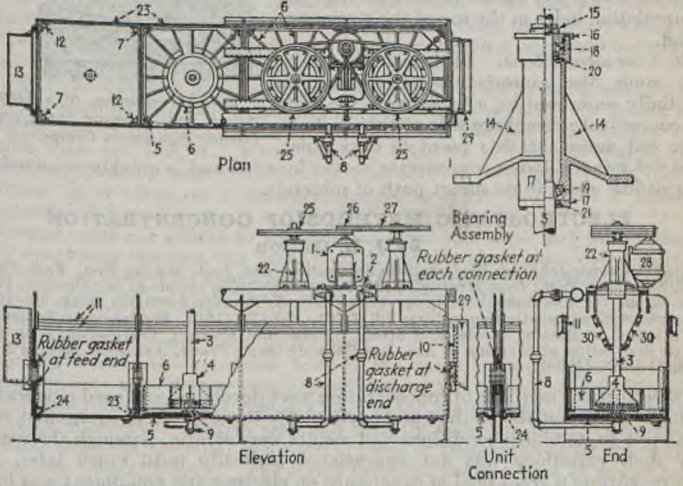


FIG. 20.—Booth Agitair flotation machine. (*Booth-Thompson Division, The Galigher Company.*)

Denver Sub-A (Fahrenwald) Flotation Machine—Operation of the Denver Sub-A Cell (Fig. 21). 1. *Mixing and Aeration Zone.* The pulp flows into the cell by gravity through the feed pipe, dropping directly on top of the rotating impeller below the stationary hood. As the pulp cascades over the impeller blades, it is thrown outward and upward by the centrifugal force of the impeller. The space between the rotating blades of the impeller and the stationary hood permits part of the pulp to cascade over the impeller blades. This creates a positive suction through the ejector principle, drawing large and controlled quantities of air down the standpipe into the heart of the cell. This action thoroughly mixes the pulp and air, producing a live pulp fully aerated with very small air bubbles. These exceedingly small, intimately diffused air bubbles support the largest number of mineral particles.

2. *Separation Zone.* In locating the impeller below the stationary hood at the bottom of the cell, agitating and mixing are confined to this zone. In the central or separation zone the action is quiet and cross currents are eliminated, thus preventing the dropping or knocking of the mineral load from the supporting air bubble, which is very important. In this zone the mineral-laden air bubbles separate from the worthless gangue, and the middling product finds its way back into the agitation zone through the recirculation holes in the top of the stationary hood.

3. *Concentrate Zone.* In the concentrate or top zone, the material being enriched is partially separated by a baffle from the spitz or concentrate discharge side of the machine. The cell action at this point is very quiet, and the mineral-laden concentrate moves forward and is quickly removed by the paddle shaft (note direct path of mineral).

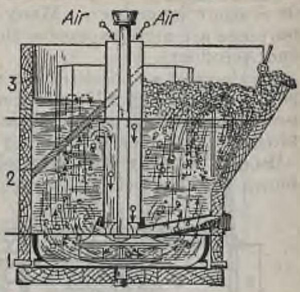


FIG. 21.—Section of a Denver Sub-A cell, showing three zones. (*Denver Equipment Company.*)

ELECTROSTATIC METHODS OF CONCENTRATION

By J. L. Gillson

REFERENCES: Johnson, Electrostatic Separation, *Am. Inst. Mining Eng., Tech. Paper* 877, February, 1938; *Eng. Mining J.*, **138,** September, 1938, pp. 37–41, 51; October, 1938, pp. 42, 43, 52; December, 1938, pp. 41–45. U. S. Patents 1,020,063, Mar. 12, 1912, Process of Electrostatic Separation; 1,017,701, Feb. 20, 1912, Electrostatic Separator, both issued to Sutton, Steele, and Steele. Fraas and Ralston, Discussion of Electrostatic Separation at Meeting of Am. Inst. Mining Eng., *Trans. Am. Inst. Mining Eng.*, **134,** 419–421 (1939).

Electrostatic methods of concentration were developed and used moderately prior to 1900, but the method and particularly the generating equipment were sensitive to humidity conditions and results were erratic, although the reason for such variations was not appreciated generally until much later. In the meantime a great deal of experience on electrostatic equipment was built up through the use of the Cottrell process for dust precipitation, and there was much improvement of electrostatic generators for other uses requiring high voltage. A number of plants concentrating lead and zinc ores employed the process until differential flotation was developed about 1914. The rejuvenation of the process occurred first with applications to the food industry, in which the process finds some application for cleaning and separating seeds.

The principle of electrostatic separation is based on the fact that, if one or more minerals in a mineral aggregate or ore can receive a surface charge in an electrostatic field, the grains will be repelled from the electrode or attracted toward it, depending upon the sign of the charge. By causing such grains to fall into separate chutes from other grains not so affected, a separation or concentration results.

H. B. Johnson has published a list of the conditions of 90 common ore minerals. He classified these into three groups; those which behave the same irrespective of the polarity of the electrode, those which will reverse their own charge in a negative field, and those which will reverse their charge in a positive field. Patents issued in 1912 covered these features, but actually, as pointed out by Fraas and Ralston, this tabulation confuses contact potential or frictional electricity with electrical conductivity. Those that are reversible positive and reversible negative are all non-conductors, whereas those classed as non-reversible are conductors or have such a low contact potential with the metal of the rotor that any slight existing electrical conductivity is the controlling factor. Those that are reversible positive are positive as a result of contact potential, while those that are negative are negative with respect to the conveying roll. Films of organic matter, or coatings of iron oxide or other compounds may change the conductivity of the grains or may affect the contact potential.

Most electrostatic separators consist of a rotor to serve both as the feeding device and as the separating electrode, and one or more charged electrodes (placed in front of the curtain of feed discharged from the rotor), which put a charge on the grains of the feed and the charged grains, are attracted toward the rotor or are repelled from it.

The critical factors in differential separation of grain in electrostatic fields are the conductivity of the grains and the voltage required for grains of a definite species to take a charge, or, having been charged, the rapidity with which they will lose it. When a grain takes on a positive charge, it is attracted toward a negative electrode and adheres to it until it has lost its charge. If it takes on a negative charge, it is repelled from a negative electrode.

Grains must not adhere to the charging electrode, since this would distort and eventually insulate the field, and, of course, they must not adhere to the rotor or separating electrode. Thus the charging electrodes, if made of metal, rotate with a brush behind the electrode to clean them, or, as in one machine, consist of a glass tube (which is an insulator) in which is a gas conducting the current, as in a neon sign.

Grains which are attracted toward the rotor or separating electrode, adhere closely to it if they retain their charge, and a brush is provided also behind the rotor to remove such grains. If the grains fail to become charged or quickly lose their charge, they fall directly from the rotor. The grains repelled from the rotor fall in a path in front of that followed by the normal trajectory from the rotor. The forces involved are the force of gravity, the tangential velocity given to the grains by the rotation of the rotor, and the actions of the grains in the electrostatic field. It is obvious that the relative mass of the individual grains must be within fairly close limits, therefore moderately close sizing of the feeds to electrostatic machines is necessary. Furthermore, material so fine as to float in air or to be blown about by air currents or material that becomes compacted cannot be treated, *i.e.*, the feed must be free flowing. The size and speed of rotation of the rotor is important since these factors control the length of time the grains are in the electrostatic field and the trajectory of the grains from the rotor.

The wave form of the rectified current is also important. In vacuum-tube rectification the primary circuit must be 50 or 60 cycle; 25 cycle gives a pulsating effect.

The contact potential is important. In one machine a needle-point electrode is used in order to impress a charge so strongly on grains that the initial frictional charge is unimportant. The U. S. Bureau of Mines is working on methods of taking advantage of the contact potential in machines so designed. Ignoring it explains why electrostatic methods have not been so successful on non-conductors as on conductors.

There are three machines now on the United States market which differ in detail. The Ritter Products Co. machine, of Rochester, N. Y., resembles the original Huff machine and consists of a series of rotors superimposed in a tier. Material not rejected by the first electrode passes to the second rotor and is again exposed to the field with the same or reversed polarity. The Sutton machine has two electrodes (Fig. 22), one a needle-point electrode that is designed to impress a strong charge on the mineral grains and a second gas-filled glass-tube electrode having the same polarity as the needle-point electrode that enlarges the electrostatic field. The third machine marketed by the Feldspathic Research Co. preconditions the feed by passing it quickly through a chamber in which it is exposed to a warm atmosphere of HF of about 1 per cent strength. The electrostatic separator itself is similar to that of the old Huff machine. Presumably the HF cleans the mineral surfaces of inhibiting films and may give the grain surfaces a positive or negative charge. Tests have proved that the subsequent attraction or repulsion of the grains so treated is more marked.

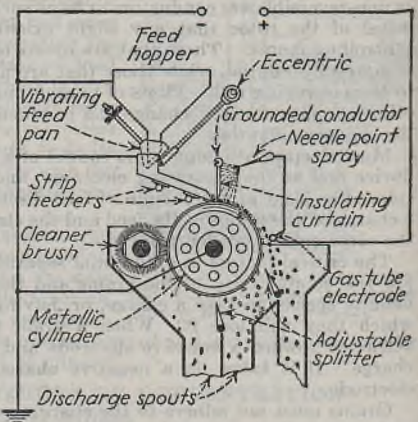


Fig. 22.—Sutton, Steele, and Steele Electro-float separator.

To control humidity conditions, which is important, the machines may be operated in an air-conditioned atmosphere. Tests on some ores showed that a relative humidity below 35 per cent was necessary for effective separation. The Sutton machine employs a heater so that the feed is raised above 150°F., at which temperature the relative humidity at the electrode is reduced. The feed from the HF preconditioner in the Feldspathic Research Co. machine is also warmed above the point of critical humidity.

Electrostatic machines are effective on ores in the range from about 10 mesh to 200 mesh (but feeds must be dust free), and for clean separation the mineral grains must be free, *i.e.*, individual particles of a single mineral. The inability of handling material finer than about 200 mesh restricts the application of the method. Most ores must be ground to 20 to 60 mesh to cause nearly complete liberation of the particles, by which grinding from 20 to 40 per cent of feed is unnecessarily, but inevitably, ground finer than 200 mesh. Fine particles fly about in the air. Some tests have been conducted on equipment to conduct the separation of fines in reduced atmospheres, but this increases the tendency for arcing across from electrode to ground, which would inhibit separation. Voltages used vary from 10,000 to 30,000, but the amperage is very low, from $\frac{1}{20}$ to $\frac{1}{4}$ milliamp.

High voltages are developed by d.c. generator sets with mechanical rectification with a cross-arm rectifier or by vacuum tubes. A simple and unique generator is the McCutcheon (made by the Communications Measurement Laboratory of New York) in which a transformer steps up a primary plant circuit and rectifies it by means of four R.C.A. No. 878 tubes.

The principal application of electrostatic methods so far is on agricultural products, but a number of ores are being treated by the method. The separation of zircon and rutile is difficult by other means because of similar physical properties of the minerals, and two operations are being conducted abroad using electrostatic equipment. The separation of feldspar and quartz for the ceramic industry is being done electrostatically by one of the large domestic feldspar companies; the advantage of the process over froth flotation is that it avoids subsequent drying. In general, it can be said that electrostatic methods are applicable as an adjunct to other dry methods of concentration where it is desirable or necessary to keep feeds dry throughout the concentration process and in a few special cases where other methods are not successful.

Electrostatic machines have capacities varying from 100 to 150 lb. per hr. per ft. of rotor length on material of 100 mesh, and 350 to 500 lb. per hr. per ft. of rotor length on material of 10 to 20 mesh. The horse power required is low, running from 1 to 2 hp. per machine, depending upon the number of rotors required. The floor space is small but the working height of an eight- to twelve- rotor machine will run from 12 to 20 ft. With a multiple-rotor machine the upper rotors are used for roughing, intermediate rotors for scavenging, and lower rotors for cleaning. The passage of material from rotor to rotor must therefore be adaptable, permitting either the discharge from the front or the back of the machine to reach the next, or a succeeding rotor. A single operator can handle three to six machines, depending upon whether feeds are continuous or whether units are installed to handle successively batches of different products accumulating in surge bins.

High voltages present physical hazards and an electrostatic machine should be enclosed so that body contact with electrodes and terminals is impossible.

CONCENTRATION OF ORES BY SINK-AND-FLOAT METHODS

By Robert Ammon*

For many years beneficiation of coal has been practiced in this country by modified sink-and-float processes such as those invented by Chance, Rheolaveur, and others. The beneficiation of ores has not been carried out on a practical scale in the United States by sink-and-float methods until very recent years. At the present time there are two outstanding developments in this field, namely, the process using heavy-density liquids such as halogenated hydrocarbons for the separating medium and the process using a heavy-density mixture of water or other liquid with a comminuted solid as the separating medium.

These processes have their limitations in that ores amenable thereto must be of such nature as to liberate in crushing a goodly portion of gangue, free, or practically free, of valuable mineral, and, moreover, ore treated by these processes must have the fines removed, but the upper range of size of materials treated is almost entirely dependent upon the liberation of the valuable mineral in crushing. These processes, from a practical standpoint, should handle any ore amenable to jigging, but in a much more efficient way metallurgically and with a larger range of sizes of feed.

---

* Chief Metallurgist, American Zinc, Lead & Smelting Co.

There has been put into practical operation in recent years three large-scale plants for the treatment of ores by the sink-and-float method of using a heavy-density separating medium composed of comminuted solids and water. Various solids may be used such as ground jig galena concentrates, flotation galena concentrates, or other solids.

This process is covered by patents [U. S. Patents Nos. 1,895,504, 1,895,505, 2,176,189. Other patents pending] and the American Zinc, Lead & Smelting Co. of St. Louis, Mo., is the exclusive agency for the licensing of the use of this process in the United States, Canada, and Mexico. During the past year approximately 1,000,000 tons of lead-free zinc ore have been treated at Mascot, Tenn., by the American Zinc Co. of Tennessee, a wholly owned subsidiary of American Zinc, Lead & Smelting Co.; approximately 2,000,000 tons of lead-zinc ores have been treated in the Tri-state District; and on the Mesabi iron range approximately 1,000,000 tons of iron ore have been treated for the production of commercial ore. The three units are now treating between 4,000,000 and 5,000,000 tons of crude ore annually. Details of the Mascot, Tenn., operation are quoted as an example of the practicability of this process.

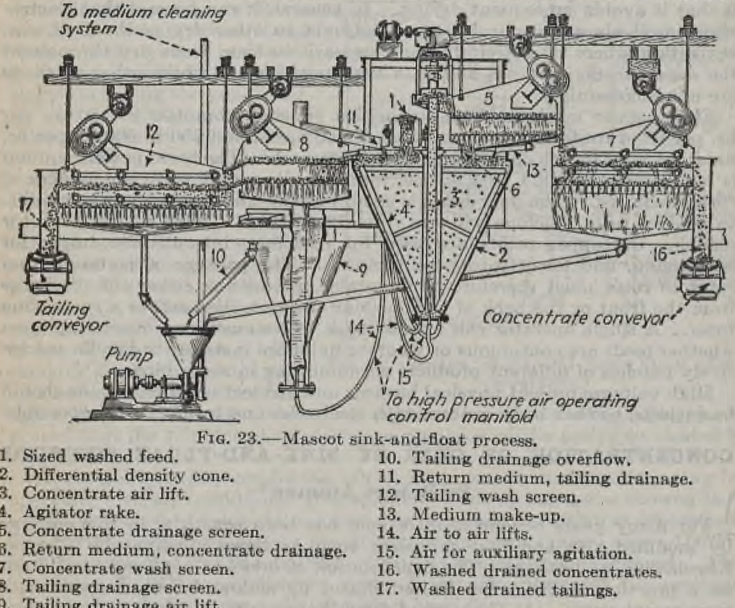


FIG. 23.—Mascot sink-and-float process.

1. Sized washed feed.
2. Differential density cone.
3. Concentrate air lift.
4. Agitator rake.
5. Concentrate drainage screen.
6. Return medium, concentrate drainage.
7. Concentrate wash screen.
8. Tailing drainage screen.
9. Tailing drainage air lift.
10. Tailing drainage overflow.
11. Return medium, tailing drainage.
12. Tailing wash screen.
13. Medium make-up.
14. Air to air lifts.
15. Air for auxiliary agitation.
16. Washed drained concentrates.
17. Washed drained tailings.

At Mascot, Tenn., a lead-free zinc ore (gangue chiefly dolomitic limestone) is treated at the rate of 3000 tons per day by the sink-and-float process, known as the differential density (D.D.) process. Prior to the advent of this process the flow sheet included jigging and flotation; 50 per cent of the original ore being rejected as ½-in. jig tailings. A striking contrast exists today at Mascot in that more than 60 per cent of the mine ore is being rejected as D.D. tailing, with the ore being crushed to 1½ in. Approximately

25 per cent of the mine ore is removed as ¼-in. fines, and the tailing from the D.D. process operating on the remaining 75 per cent of the crude ore assays 0.35 per cent zinc as against 0.65 per cent zinc formerly made on the jigs. This indicates one advantage of this D.D. process, namely, the production of a large tonnage of coarse tailings, which in certain localities is an asset for sale as a by-product.

The diagram of Fig. 23 illustrates the separatory mechanism of the process. At Mascot the ore is crushed to 1½ in. and about 25 per cent thereof removed as fines, and the remainder is thoroughly washed to remove slimes. In the illustration, 1 indicates the ore being fed to the cone, which is filled with the separating medium composed of crushed galena and water. At the top of the liquid mass, the gravity of the medium is maintained for the Mascot ore at 2.80 and at the bottom at 2.95. From the top to the bottom of the cone the specific gravity of the medium shows a constantly increasing differential.

The D.D. process is a true sink-and-float separation; the surface of the liquid mass in the cone is quiescent. No agitator is used, and only a very slow rotary motion is imparted to the mass in the cone by the slowly revolving paddles (4). Separation is accomplished by the particles of feed of greater than 2.80 specific gravity immediately sinking and particles of lighter gravity floating and being carried off in the weir overflow to the drainage screen (8).

The air lift (3) constantly removes the particles (principally mineral bearing) settling in or near the bottom of the cone, along with a portion of the medium, which medium is returned to the top of the cone after having been drained through drainage screen (5). The medium overflowing with the floated tailing is drained and returned to the top of the cone at (11) through an air lift equipped with a surge cone (9). Any deficiency in the specific gravity of the medium caused by loss of galena in the process or by the entrance of moisture with the new feed is made up through a heavy-density medium reserve supply (13) shown in the illustration. The screens (12) and (7) are washing screens for removing adhering fine galena after drainage. The galena (medium) thus removed is extremely diluted and is sent through appropriate equipment for removing colloidal slimes and sands; it is then densified and reused for make-up medium (13) along with additions of new galena required.

At Mascot the cost of operating this D.D. plant is slightly less than the cost of jigging as formerly practiced, while the recovery of zinc has been increased approximately 4 per cent over the recovery by jigging. The cost of crushing has been materially reduced in the order of 1½ in. from ½ in. About 1.6 lb. of galena are added per ton of mine ore treated, of which 50 per cent thereof is reclaimed, so that the net cost of galena is approximately 0.8 lb. per ton of mine ore, or 3 cts. per ton of mine ore. Water is in closed circuit so that practically no new water is needed. Power and labor are each approximately 1 ct. per ton of mine ore treated, so that the total operating costs are not in excess of 8 cts. per ton when operated on the basis of 3000 tons per day. Intermittent operations can be conducted, provided provision be made to keep the medium in circulation to prevent excessive settling in both the cone and the tanks.

The purification of the medium washed from the tailings and concentrates is carried out at Mascot by tabling for the removal of sands and classification by decantation for disposing of the colloidal slimes entering with the feed.

Capital costs are not high, approximately $35 per ton-day for complete installation, excluding ore crushing but including building. This cost is based on 2000 to 3000 tons per day capacity.

SAMPLING

BY J. W. STILLMAN

REFERENCES: Jones, Sampling of Materials: I. General Principles Involved, *Metal Ind. (London)*, **32,** 585–588 (1928); II. Gases and Liquids, *ibid.*, 609–612; III. Sampling of Water, Oil, Slimes, and Powdered Substances, *ibid.*, **33,** 3–4 (1928); IV. Sampling of Coal, *ibid.*, 27–28; V. Sampling of Ores and Concentrates, *ibid.*, 125–128; VI. Sampling of Ores, Metals and Alloys, *ibid.*, 199–203. Baule and Benedetti-Pichler, Sampling of Granular Materials, *Z. anal. chem.*, **74,** 442–456 (1928). Grumell, A Decade of Sampling, *Am. Inst. Mining Met. Eng., Tech. Pub.* 1044, 1939. Munch and Bidwell, What Constitutes an Adequate Sample? *J.A.O.A.C.*, **11,** 220–221 (1928). Standard Methods for the Sampling and Analyzing of Aluminum and Certain Aluminum Alloys, Aluminum Research Inst., 1932. Calkins *et al.*, Testing of Coal and Coke, *Intern. Cong. Testing Materials* 1927, II, 641–655. Grumell, Report on the Sampling of Coal with Special Reference to the Size-weight-ratio Theory. Crawford and Reed, Notes on Sampling and Analysis for Ash Content, *Brit. Standards Inst., London*, 1938. Standard Methods of Laboratory Sampling and Analysis of Coal and Coke, A.S.T.M., D 271-37. Bushell, The Sampling of Coal, *J. Chem. Met. Mining Soc. S. Africa*, **37,** 361–434, 494–495, 499–501, 566–568 (1937). "Methods of the Chemists of the U. S. Steel Corp. for the Sampling and Analysis of Coal, Coke and By-products," 3d ed., 1930. Fieldner and Selvig, Review of Standard Methods Used in Various Countries for Sampling and Analysis of Solid Fuels, *Fuel*, **17,** 266–271 (1938); Notes on the Sampling and Analysis of Coal, *U. S. Bur. Mines, Tech. Paper* 586, 1938. Sampling of Coal Tar and Its Products, *Brit. Standards Inst.*, 1938. Matthews *et al.*, Porous Solid Filters for Sampling Industrial Dusts, *Bull. Inst. Mining Met.*, No. 386, 1936. Briscoe *et al.*, The Sampling of Industrial Dusts by Means of the "Labyrinth," *Bull. Inst. Mining Met.*, No. 393, 1937. Littlefield and Shrenk, Dust Sampling with the Bureau of Mines Midget Impinger, Using a New Hand-operated Pump, *U. S. Bur. Mines, Rept. Investigations* 3387, 1938. Richardson *et al.*, Standard Methods for the Sampling and Analysis of Commercial Fats and Oils, *Ind. Eng. Chem.*, **18,** 1346–1355 (1926). Thatcher, Sampling of Boiler Flue Gases, *Power*, **64,** 774–777 (1926). "Methods of the Chemists of the U. S. Steel Corp. for the Sampling and Analysis of Gases," 3d ed., 1927. "Sampling and Analysis of Carbon and Alloy Steels—Chemists Committee of the Subsidiaries of the U. S. Steel Corp," Reinhold, 1939. "Recommended Methods for Sampling and Analysis of Cast Ferrous Metals and Alloys," 2d ed., Brit. Cast Iron Research Assoc., 1933. "Methods of the Chemists of Subsidiary Companies of the U. S. Steel Corp. for the Sampling and Analysis of Pig Iron," 3d ed., 1934. Zwicker, Note on the Significance of Sampling for the Chemical Analysis of Metal Alloys, *Chem. Ztg.*, **53,** 546–547 (1929). Standard Methods of Sampling Petroleum and Petroleum Products, A.S.T.M., D 270-33. Wittka, Sampling and Measuring the Amount of Oil from Tank Ships, *Allgem. Oel- Fett-Ztg.*, **33,** 470–478 (1936). Smither *et al.*, Standard Methods for the Sampling and Analysis of Commercial Soap and Soap Products, *Ind. Eng. Chem.*, anal. ed., **9,** 2–8 (1937). Fuller *et al.*, Sampling and Testing Lacquer Solvents and Diluents, *Proc. Am. Soc. Testing Materials*, **27,** Part 1, 870–872 (1927).

Sampling is the process of obtaining a small amount of material which shall be as nearly representative as possible of the whole mass of material which is being considered. This process usually is made up of several separate steps: (1) the collection of a comparatively large amount of material which has been selected in a systematic manner from different parts of the mass; (2) if the material is composed of solid particles, the crushing and grinding of the portion collected as described under (1) in order to reduce the size of aggregates and to provide a certain amount of mixing; and (3) the separation of this comparatively large portion into two parts by subdivision, such as by quartering, so that one part provides sufficient material for the required analysis or test and at the same time has the same average composition as the large portion before subdivision.

Since the final sample is in most cases to be used for test purposes the results of which will determine the use to which the entire mass may be put, it is obvious that all precautions which aid in making this sample representative of the original material are thoroughly justified. An analysis or test, however efficiently it is carried out, will be rendered valueless if the sample has been improperly taken or prepared. Methods of sampling have been devised with due consideration for the laws of probability and of averages. They should be applied by someone who understands the scientific aspects of sampling and who comprehends the objectives of the analysis or test to be made on the sample. Any directions which are given here are intended to supplement the experience of the sampler and guide him in selecting methods which are applicable.

In taking the gross sample, careful consideration should be given to the present condition of the material. Such questions as the following should be answered and will decide the number and location of portions to be taken to provide the gross sample:

Is the surface layer the same as the mass underneath or has it been changed by exposure to the weather or other external conditions? Has there been segregation of coarse and fine particles or of materials of different specific gravities? If the material has been transported has there been segregation? When the material is a mixture of liquids, or of liquids and solids, there is a tendency to segregation.

These questions call attention to some of the difficulties which have to be overcome in sampling heterogeneous materials. Only homogeneous materials of which very few are met in practice can be sampled at random and a representative portion obtained.

Once the gross sample has been collected, the amount of material actually required for the analysis determines the extent to which grinding, mixing, and subdivision shall be carried. Just as much care is required in these operations as for assembling the gross sample.

Where materials are bought and sold on specification, both buyer and seller are interested in obtaining an estimate of the material with respect to its specified properties. It is the customary practice to have the sampling done under conditions set by agreement. At one time, samples are taken for the buyer and seller and one or two extra portions are retained for referee samples in case of disagreement between the buyer and seller. It simplifies matters in such cases to take one gross sample and then prepare the necessary small portions at the same time and under identical conditions.

As has already been mentioned, it is easy to sample only in the case of homogeneous materials such as gases, true solutions, and finely divided solids. The difficult problems of sampling are encountered with solid materials which are practically always heterogeneous in nature, and hence this general discussion is concerned mainly with solids. Liquids and gases will be taken up in special paragraphs below. If a material can be rendered homogeneous by thorough mixing, then any part of it can be taken as a representative sample. Usually, however, with solid particles the attainment of homogeneity is difficult if not impossible, and, for that reason, in practice the different methods of sampling described below are necessary.

During the transportation of material in railroad cars, trucks, and the like, the finer materials tend to settle to the bottom leaving the larger particles on top. Certain materials are subject to oxidation when exposed to the air. The resulting oxide will, of course, be greater in amount on the surface of the mass of material. At the same time, if this oxide is a fine powder which is easily removed, it will penetrate to a greater or less extent into the pile of material as the result of erosion. These examples are typical of many heterogeneous materials which are encountered in practice, and the methods of sampling must be so designed that as far as possible the relative proportions of coarse and fine, of metal and oxide, etc., will be the same in the gross sample and in the mass of material. Once having collected the gross sample it can be crushed, ground, shredded, etc., to provide a more homogeneous mass for subdivision to the final sample. The greater the difference in size or other characteristics between the components of the material, the larger should be the gross sample taken.

When it is necessary to combine samples to make a representative composite sample, weights of the portions entering into the composite must bear the same ratio to each other as do the weights of the initial materials sampled.

Methods of Sampling

Sampling is carried out according to two general methods: hand sampling and mechanical or automatic sampling. The former, as its name implies, involves the taking of the sample by an operator using a simple tool for the purpose. On large lots this method is slow and expensive and in all cases puts great responsibility on the individual operator. In mechanical sampling a certain predetermined portion of the material is taken continuously or at regular intervals.

Grab Sampling. Grab sampling is the simplest method of sampling and is subject to the greatest inaccuracy. It consists in taking small equal portions, either at random or at regular intervals, by hand or with a scoop or shovel. The advantages of grab sampling are its economy and the speed with which it may be carried out. The disadvantage is that it is difficult to have all components truly represented when small portions are taken, especially if the material is lumpy or the sizes of particles are not uniform. The smaller the particles of the material to be sampled by this method the more accurate will be the sample. In general, grab sampling should be applied only when the material is as homogeneous as possible and only when approximate accuracy is required. The following typical examples of grab sampling may be indicative of the usefulness of this method: (1) In unloading a tank car of acid, a side-arm outlet from the main unloading pipe is opened, say every 2 min., and a small portion of 50 to 100 cc. allowed to run out into a container. This method will give for an ordinary tank car a gross sample of approximately 5 gal. which may be thoroughly mixed and reduced to the proper amount for analysis. (2) As a mixed fertilizer, ready for bagging, passes down a chute to a bin, a portion may be taken periodically with a scoop and the accumulated sample for the day mixed and worked up for analysis. (3) A large pile of material may be sampled by taking a shovel or scoop full at different parts of the entire exposed surface of the pile. The points where portions are taken should be spaced at regular intervals and one method of laying out a pile for sampling is to use a rope with knots tied in it at regular intervals corresponding to the distance between locations for sampling. The rope is thrown across the pile and a portion taken from directly below each knot. If the character of the material in the pile varies with the depth, a better plan is to take samples at regular intervals over the new face of the pile which is formed as the material is being moved away. The resulting gross sample is subdivided in the usual way.

The following methods of **grab sampling** are abstracted from the *Journal of Industrial and Engineering Chemistry*, **1**, 107 (1909) and describe methods employed by the U. S. Steel Corporation.

Car Sampling. Samples must be taken uniformly over the surface of cars by selecting a minimum of 15 places at regular intervals. These may be selected according to the parallel or zigzag system. If the former is adopted, a convenient method is to use a net with the intersecting ropes knotted at the desired locations for sampling. A sample is taken below each knot. In the zigzag system, samples are taken at regular intervals along a line drawn from one corner of the car across to a point on the opposite side about one-third of the car length from the starting point, then back to the other side at a point about two-thirds of the car length from the starting point, and then across the car to the corner diagonally opposite the first corner.

When lumps are encountered at the designated points where samples are to be taken, small portions of each lump must be chipped off. When rock occurs it must also be sampled as ore, *i.e.*, a proportionate amount of it must be taken, representing the area for which the sample is taken, together with an amount of adjacent material to bring the whole to the amount taken at each sampling point. A gross sample is taken for each 10 cars or less.

Boat Sampling. If the material is located in cone-shaped piles, portions are taken at regular intervals (about one shovel length apart) starting at a point about two shovel lengths from the side of the boat and proceeding along a line up across the apex of the cone and down the opposite side.

When a certain amount of material has been unloaded leaving the remainder with a newly exposed face, samples can be taken at regular intervals on this face. Starting at a distance of two shovel lengths from the side of the boat, the sampler proceeds up the face of the pile taking a sample at every shovel length up to the top. The next vertical line is measured four shovel lengths from the first, and in a similar manner the whole face is covered.

Round Sampling. In round sampling, one-third of the gross sample is taken, after 5 or 6 ft. of the face of the material has been exposed, in the manner described in the above paragraph. When the grabs have removed all of the material which can be reached, a second round of portions comprising two-thirds of the gross sample is taken along the face or faces which remain.

Coning and Quartering. Coning and quartering is one of the most familiar forms of hand sampling and is used generally in subdividing a gross sample to give the retained portion. It may be used on lots of material amounting to not more than 50 tons in which the particles do not exceed 2 in. in diameter. The method may be described as follows: The material to be sampled is piled into a conical heap and then spread out into a circular cake. The cake is divided into quarters, and two of the diagonally opposite quarters are taken as the sample while the two remaining quarters are rejected. The two quarters taken as a sample are collected together and the procedure of coning and quartering repeated until a lot of the material of the desired size is obtained. A complete discussion of the application of coning and quartering together with illustrations of the different steps is given in Fig. 1 and p. 1753.

During these operations care should be taken that the material is not contaminated by anything on the floor or that part of the sample is not lost through cracks or openings in the floor. Preferably the floor should be swept clean and the operation of coning and quartering carried out with the floor covered with paper or some other suitable material. The advantages of coning and quartering are that few tools are required to carry out the method and the procedure is applicable to all kinds of solid materials. Among the disadvantages of this method are that it is expensive because frequent handling of the material is required and that it does not give an accurately representative sample. The larger sizes of material roll down the sides of the cone and collect around the base, while pieces of intermediate size arrange

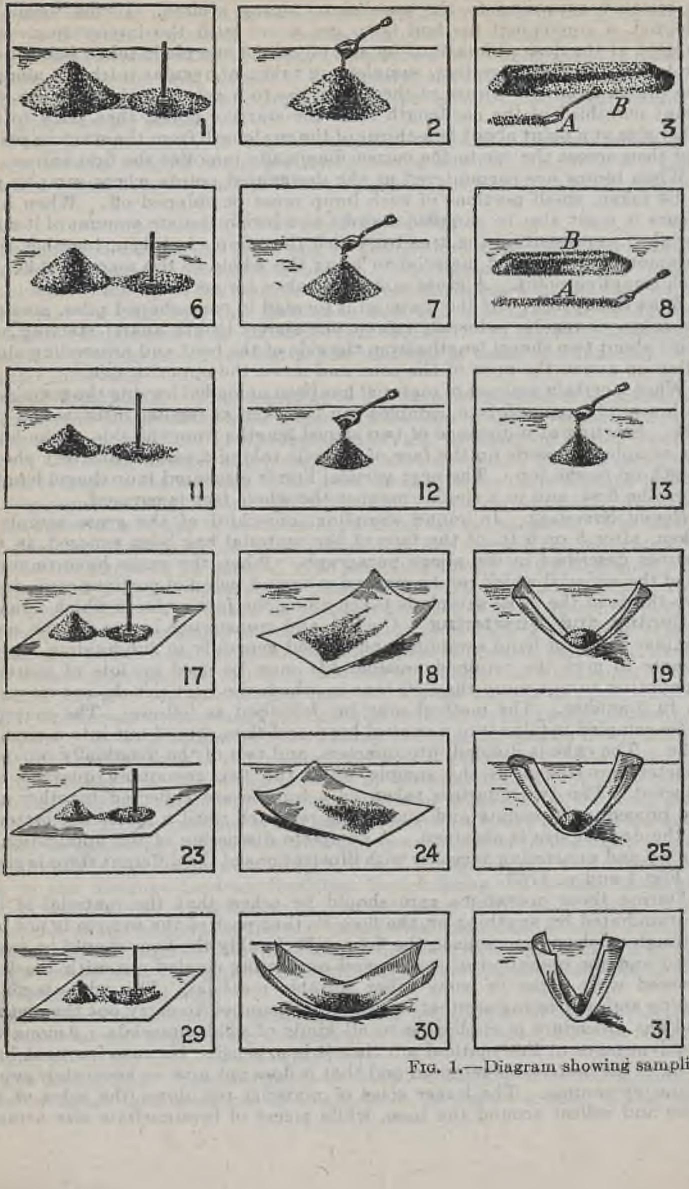
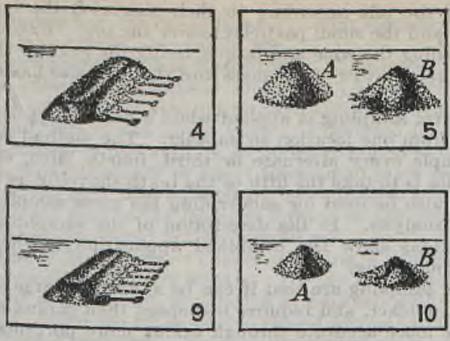
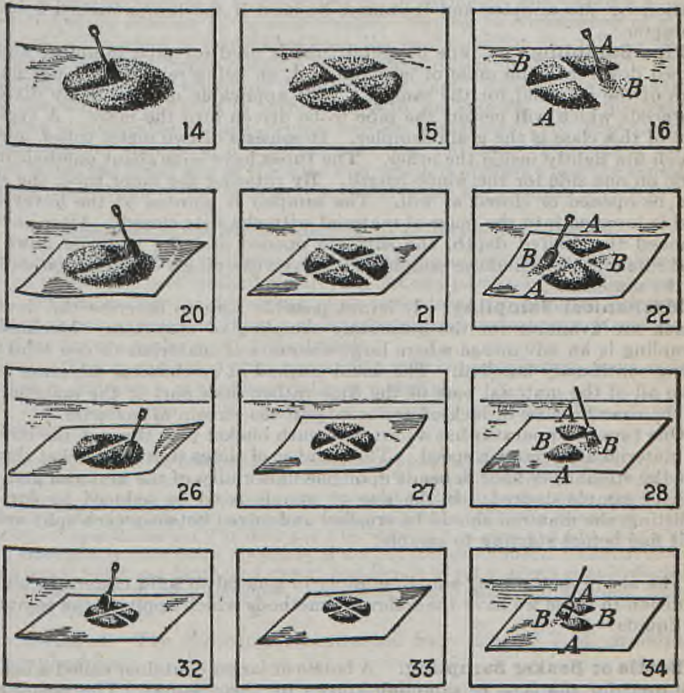


Fig. 1.—Diagram showing sampling



NOTE

Select a Hard, Clean Surface, Free of Cracks and protected from Rain, Snow, Wind and Beating Sun. Do not let Cinders, Sand, Chippings from Floor or any other Foreign Matter get into the Sample. Protect Sample from loss or gain in moisture



method by coning and quartering.

themselves on the slope of the pile according to their size with the large particles nearer the bottom and the small particles nearer the top. Extreme care must be taken in flattening the cone and in quartering the cake so that the resulting material is representative of the mass and that one size has not been segregated.

Shovel Sampling. Shovel sampling is applied when a material is being shipped or is being moved from one location to another. The method consists in taking for the sample every alternate or third, fourth, fifth, etc., shovelful. Common practice is to take the fifth or the tenth shovelful as the sample. This method can also be used for subdividing the gross sample to obtain the proper size for analysis. In the description of the sampling of coal, given below, illustrations show the combined application of shovel sampling, and coning and quartering.

The advantages of shovel sampling are that it can be applied to large lots of material. It is cheaper, quicker, and requires less space than coning and quartering. It tends to be more accurate through taking more portions in collecting the gross sample. Its disadvantages are that it is subject to manipulation by the sampler and it cannot be used if the lumps exceed 2 in. in diameter.

Pipe Sampling. A pipe or metal tube is used for pipe sampling and is forced down into the mass of material and, on being removed, brings out a core of the material for the sample. It is applicable only to finely divided materials which will permit the pipe to be driven into the mass. A typical tool of this class is the grain sampler. It consists of two metal tubes, one of which fits tightly inside the other. The tubes have slits about one-half inch wide on one side for the whole length. By rotating the inner tube, the slits can be opened or closed at will. The sampler is pointed at the lower end and is inserted into the mass of material with the slits closed. After having reached the desired depth, the slits are opened and the material flows in. The slits are closed and the sampler is removed carrying a cross-section sample of the material.

Mechanical Sampling. It is not possible here to describe the devices which are available for the automatic sampling of materials. Mechanical sampling is an advantage where large amounts of materials of one kind are being continually handled. The usual method of mechanical sampling is to take all of the material part of the time rather than part of the material all of the time because of lack of uniformity in the stream of material.

One type of apparatus has a narrow-mouth bucket pass through the stream of material at a uniform speed. The number of times that the bucket should cut the stream per hour depends upon the uniformity of the material and the size of sample desired. If the size of sample is to be reduced by further splitting, the material should be crushed and mixed between each split unless it is fine before starting to sample.

The above methods of sampling apply in general to solid materials, and in addition to these we have the following methods which apply to the sampling of liquids:

Bottle or Beaker Sampling. A bottle or larger container called a beaker are used for the type of sampling known by their names. The opening in the bottle is closed by a tight-fitting stopper which can be removed by means of a wire when the container is submerged in the liquid. The bottle is fastened to a pole and dipped into the liquid to the desired level when the stopper is removed and the liquid flows in. This method is applicable to tanks, tank cars, tank trucks, ship tanks, etc.

Continuous Liquid Sampling. Continuous sampling is applied to pipe lines, filling lines, transfer lines, etc. A pipe of small bore with a spigot on its outer end is inserted through the wall of the pipe line and in this way a small part of the liquid stream is diverted for a sample. Since the flow of a stream is not uniform at different points along the cross section of the pipe, it is customary to insert three small pipes each for a different distance within the pipe, thus obtaining a more uniform sample. A sample amounting to 0.1 per cent of the material flowing in the pipe should be taken, but the gross sample should not be over 40 gal.

Dipper Sampling. Dipper sampling is applied where there is a free flow or open discharge of a stream. By the grab-sampling method the full cross section of the stream is collected at intervals with a dipper which should hold 1 qt. Again the gross sample should amount to 0.1 per cent of the flow.

Thief Sampling. In sampling liquids a long tube or "thief" is often used. A typical thief is 3 ft. long, 1¼ in. in diameter, and tapered at the lower end to an opening ⅜ in. in diameter. It may be constructed of glass or metal. The thief is introduced into the container for the full depth of the liquid with the upper opening of the tube closed by the thumb. The thumb is released and the liquid flows into the thief. Closing the tube again with the thumb, the thief is withdrawn with the sample. This method of sampling is applicable to cans, drums, barrels, tanks, etc. For large tanks a special sampler similar to a thief sampler may be used. This has a valve which closes the lower end of the tube and which is opened when the sampler touches the bottom of the tank and is closed when the sampler is raised.

Practical Methods Employed in Different Industries

It is not possible here to discuss adequately the sampling of every material or even to give the best method of carrying out any particular sampling operation. The materials for which sampling procedures are outlined below are selected as being typical of those met in practice, and the procedures will serve as guides for use under similar conditions.

Coal. The sampling of this commodity will be described in some detail because it still is one of the essential raw materials.

At the Mine. The U. S. Bureau of Mines recommends the following procedure:

> Clean away all foreign material from the face to be sampled for a width of 5 ft. Cut away the coal from floor to roof for a width of 1 ft. and to a depth of 1 in. Discard the cuttings. In taking the sample, everything should go into it that would go into ordinary production. In the same way exclude anything ordinarily discarded. Sampling cut: Perpendicular cut 2 in. deep and 6 in. wide (or 3 in. deep and 4 in. wide in soft coals) from roof to floor down the center of the cut previously made. The cut should be uniform in width and depth, and enough coal should be cut to give a sample weighing 6 lb. for each foot of seam thickness. Sample is then crushed and screened, mixed, coned, and quartered.

As Delivered. The following is abstracted from the A.S.T.M. procedure for sampling coal:

> Gross samples of the quantities designated must be taken whether the coal to be sampled consists of a few tons or several hundred tons because of the following cardinal principle in sampling coal that must be recognized and understood; that is, the effect of the chance inclusion or exclusion of too many or too few pieces of slate or other impurities in what or from what would otherwise have been a representative sample

will cause the analysis to be in error accordingly, regardless of the tonnage sampled. For example, the chance inclusion or exclusion of 10 lb. too much or too little of impurities in or from an otherwise representative sample of 100 lb. would cause the analysis to show an error in ash content and in heat units of approximately 10 per cent where for a 1000-lb. sample the effect would be approximately only 1 per cent, being the same whether the sample is collected from a one-ton lot or from a lot consisting of several hundred tons.

Coal should be sampled as it is being loaded or unloaded from railroad cars or other conveyances, or when discharged from bins, etc. Samples from the surface of piles or bins are generally unreliable. To collect samples, a shovel or similar tool shall be used to take equal portions. For small sizes of coal, increments of 5 to 10 lb. may be taken, but, with lump or run-of-the-mine coal, increments should be at least 10 to 30 lb. The increments shall be regularly and systematically collected so that the entire quantity of coal sampled will be represented proportionately in the gross sample, and with such frequency that a gross sample of the required amount shall be collected. The standard gross sample shall be not less than 1000 lb. except that for slack coal and small sizes of anthracite, when impurities are not present in abnormal quantities, a gross sample of approximately 500 lb. shall be considered sufficient. If the coal contains an unusual amount of impurities, a gross sample of 1500 lb. or more shall be collected. The gross sample should contain the same proportion of lump coal, fine coal, and impurities as contained in the mass. A gross sample should be taken for every 500 tons or less unless special agreement is made otherwise.

After the gross sample has been collected, it shall be systematically crushed, mixed, and reduced in quantity to convenient size for the transmission to the laboratory. The sample should be protected from loss or contamination during these operations. The progressive reduction in the weight of the sample to the quantities given in Table 1 shall be carried out according to the steps which are illustrated in Fig. 1 (Plate V, *U. S. Bur. Mines, Bull.* 116).

Table 1

Weight of Sample to Be Divided, Lb.	Largest Size of Coal and Impurities Allowable in Sample before Division, In.
1000, or over	1
500	¾
250	½
125	⅜
60	¼
30	$\frac{3}{16}$, or to pass a 4760-micron (No. 4) sieve.

If it is necessary to sample a carload of coal before it is unloaded, cut a trench 2 ft. wide by 2 ft. deep along the two center axes of the car. The sample is then taken at intervals along the newly exposed faces and bottoms of the trenches.

Sampling of Coal Which Is Classified According to Ash Content. Statistical studies of the sampling of coal have shown that it is not always necessary to take as large gross samples as are specified above in order to obtain a reasonable accuracy in the test to be made. These studies have also shown that the ash content is a satisfactory criterion for the classification of coals. The Brit. Standards Inst. and the A.S.T.M. (D 492-38T) have set up tables showing the size of gross sample to be taken for different coals classified according to ash content and according to size. The A.S.T.M. method provides two procedures, one for special accuracy and one which in 95 cases out of 100 will give an accuracy of ±10 per cent in the ash content. As this new method of selecting the size of sample becomes familiar, it will no doubt displace the older method described above.

Iron Ore. *Piles.* Divide surface of pile into equal areas and select equal volume of material from each area. From all fines, select portion from center of area. From all lumps break off small pieces. From lumps and fines take proper proportion of both. Pile should be sampled at intervals of 2 to 10 ft. As suggested above, a rope with knots tied in it is convenient for locating sampling spots.

Cars. A minimum of 24 samples should be taken. Composites may be made to include up to 10 cars.

Boats. One-half pound samples may be taken at several points below and at equal distance from the apex of conical piles; or at intervals along newly exposed faces as ore is unloaded.

Pig Iron. *In Furnace.* Sampled while molten with spoon or ladle and poured into mold or on an iron plate. A sample is taken from the middle of each ladle of iron in the cast. Equal quantities from each portion are combined into one sample. Test pieces may be drilled or crushed.

From Car or Storage. One pig is taken for every 4 tons iron. If sows or scraggy pieces go into the steel-making furnace, they should be sampled in correct proportion. Otherwise they are not taken. The surface is cleaned by grinding and drilled with ½-in. drill through a metal cover to prevent contamination. The drillings are combined, weighed, and sieved through 80- and 120-mesh screens. The sample when weighed out is taken from the three sizes, classified by the sieves in proportion to their total weight.

Steel. One sample per heat is taken at a time when one-half has been poured by holding a spoon under the stream from the ladle while it is slackened. The sample is poured into a mold and drilled with ¾-in. drill. All drillings are discarded until the outer edge of the drill is buried.

Metals and Alloys. In general, metals and alloys are sampled by drilling with a ½-in. drill, using no lubricant. Any surface contamination should be removed and discarded before the sample is collected. It is often convenient to use a template in order that the peice can be drilled in a systematic manner.

Cement. According to the A.S.T.M. recommendations:

For Individual Samples. If sampled in cars, one test sample shall be taken from each 50 bbl. or less. If sampled in bins, one sample shall represent each 200 bbl.

For Composite Samples. If sampled in cars, one sample shall be taken from one sack in each 40 sacks (or 1 bbl. in each 10 bbl.) and combined to form one test sample. If sampled in bins or warehouses, one test sample shall represent not more than 200 bbl.

Soft Solids. Three sets of borings ¾-in. in diameter are taken through the depth of the material.

Lumpy Solids. Sample taken by the grab method taking approximately 0.1 per cent of the lot but not less than 50 lb. nor more than 1000 lb.

Petroleum Products.[*] *For Tanks and Tank Cars.* Four types of liquid samples are recognized: all-level sample, upper sample, middle sample, and lower sample. An all-level sample is obtained when a bottle unstoppered is lowered through the liquid to the bottom and back to the surface at such a rate that the bottle is just filled when it reaches the surface. Samples at the different levels are taken by lowering a stoppered bottle to the desired

---

[*] Abstracted from recommendations of the A.S.T.M. D 270-33.

level, releasing the stopper and allowing the liquid at that level to fill the bottle. A sample taken at a point 10 per cent below the top is an upper sample and a sample taken at a point 10 per cent above the bottom is a lower sample. The middle sample is taken at a depth of 50 per cent.

In making a composite sample for a tank, samples from the different levels should be combined as shown in Table 2.

Table 2

Sample	Vessels of uniform cross section	Horizontal cylindrical tanks (full)
Upper................	1 part	1 part
Middle................	3 parts	8 parts
Lower................	1 part	1 part

Individual samples taken with a beaker, bottle, or dipper should amount to 1 qt. Composite samples should be 5 qt. for vessels of uniform cross section and 10 qt. for horizontal cylindrical tanks. Gross liquid mixed-cargo samples are taken from the various ship's tanks in multiples of 5 or 10 qt. depending upon the shape of the tank. Liquid samples taken by the continuous or dipper method should be 0.1 per cent of the amount shipped but not less than 5 gal. nor more than 40 gal.

For Pipe Lines. The method of continuous sampling described on p. 1753 is used most frequently in the petroleum industry for the sampling of materials flowing in pipe lines. If the material to be sampled is semiliquid, the lines and receiver are warmed to keep the material just above the liquefying temperature. The gross continuous sample, as collected and amounting to not over 40 gal. as specified above, is thoroughly mixed and sampled with a thief to get a 1 qt. sample.

Lacquer Solvents. *From a tank car*, a ½-gal. gross sample in small portions of not over 1 qt. each is taken from near the top and near the bottom using the bottle method.

From drums, at least 5 per cent of the packages are sampled using a thief at center of the drum and taking not less than ½ pt. from each drum. Gross sample should be not less than 1 qt.

Water. *Reservoir.* A stoppered bottle is submerged to the required depth and the stopper released. The bottle should be rinsed several times before the sample is taken.

Stream. Immerse the bottle in the stream making an effort not to disturb silt or other solid material on bottom or sides.

For bacteriological tests the sample should be collected in a bottle which has been thoroughly sterilized by steaming, and special precautions should be taken that the sample is not contaminated.

Molten Metal. Dip the sample with a ladle, breaking through the slag and holding the ladle until it reaches the temperature of the mass.

Gases. The sampling of gases is comparatively easy, as in many cases they are homogeneous. Two kinds of samples are recognized: accumulative, which is taken continuously over a period of time (½ hr. to 24 hr.); and control, which is taken for less than ½ hr.

Where a gas is known to be homogeneous in the cross section, the sample may be withdrawn through a pet cock inserted in the wall of the container. If the gas is under pressure, the sample may be released through the pet cock.

Otherwise it may be withdrawn by means of an aspirator bottle, filled with water, mercury, or other suitable liquid.

Flue Gas. If one constituent of flue gas, as for example CO_2, is to be determined continuously and automatically, a permanent fixture is connected in the flue to conduct the gas to the CO_2 recorder.

If a complete analysis of the gas is to be made, consideration must be given to the fact that the gases move with different velocities at different points in the cross section of the flue. For sampling the gas under such conditions, a perforated pipe is used which enters the flue with an air-tight connection and extends entirely across the flue. Perforations in the pipe should be equal distances apart and to obtain an equal flow through all openings, their combined area should be less than the cross-sectional area of the pipe—3:4 being considered a safe ratio. The gas is then withdrawn through the pipe by means of suction until the desired sample has been collected.

Air or Gas in Confined Space. Fill a 2-oz. bottle with mercury, let stand for 2 min., and then pour the mercury back into the stock bottle thus filling the bottle with air at that spot.

In special cases where an instantaneous undiluted sample is required, an evacuated tube is used. The tube is pumped out and the stopcock is opened where the sample is to be taken.

Dust- or Fume-laden Air. The air is filtered through a specially prepared filter such as a Gooch crucible containing filter paper, and the residue obtained from a known volume of air is weighed. Or the air may be projected on a glass slide with some means of causing the dust to adhere with suitable provision for observing the increase in the deposit.

GENERAL RULES FOR SAMPLING

1. Sample to be taken by or under direct supervision of a person qualified by experience to recognize that sample is satisfactory for the subsequent test to be made.
2. Select the most appropriate method of sampling with due regard to the kind of material and the conditions of storage or handling.
3. Watch for special conditions which make for non-homogeneity in the material such as weathering of the outside of a pile of material, segregation of sizes, or more than one layer in liquids.
4. Take all necessary precautions to avoid contamination of sample after it has been taken.
5. Label the sample clearly with all necessary information to designate the source from which it was taken.

CRYSTALLIZATION

BY WARREN L. McCABE

Crystallization, as an industrial process, is important because of the great variety of materials that are marketed in the crystalline form. Its wide use is due basically to the fact that a crystal forming from an impure solution is itself pure (except when mixed crystals form), and crystallization affords a practical method of obtaining concentrated chemical substances in a form both pure and attractive and in satisfactory condition for packaging, handling, and storing.

Commercial Importance of Crystal Size and Shape. It is obvious that yield and purity are of importance in operating a crystallization process, but these two factors are not the only factors to be considered. The size, and often the shape, of the crystals is important; it is especially necessary that the crystals be of uniform size. Uniformity of size is important for satisfactory appearance and tends to prevent caking, allows easy washing, and results in uniform behavior in use. Large crystals are often demanded, although such demands are not usually justified by any real advantage of large crystals as compared to medium sized ones. In some cases a definite shape is also required (*i.e.*, needles rather than plates or cubes).

THEORY OF CRYSTALLIZATION

A crystal is the most highly organized type of non-living matter. It is characterized by the fact that its constituent parts (atoms or ions) are arranged in orderly array in so-called **space lattices.** The interatomic distances in a crystal of any definite material are constant and characteristic of the material.

Crystal Forms. Law of Haüy. As a result of the space-lattice arrangement of the atoms composing them, crystals, if allowed to form without hindrance from outside bodies, appear in definite polyhedral shapes and exhibit varying degrees of symmetry. It has been found that, although the relative development of the different faces of two crystals of the same material may be widely different, the interfacial angles of corresponding faces of the two crystals are all equal and characteristic of that substance. This is the Law of Haüy.

Isomorphism. A generalization that for some time seemed at variance with Haüy's law is the law of isomorphism, which states that in certain series of chemically similar substances the crystals are of the same crystalline form. Until refined methods were available for the measurement of crystal angles it was thought that isomorphic materials gave crystals with the same angles. It has been established, however, that there are small but regular differences in the corresponding angles of isomorphous substances. These differences are of the same sort that exist among other properties of elements in the same periodic group.

Crystallographic Systems. Since the crystals of a definite substance all show the same interfacial angles in spite of wide differences in the extent of development of the faces, crystal forms are classified on the basis of the angles. For example, consider a definite crystal. Take a point and draw lines through this point normal to the faces of the crystal, or to the faces produced. The resulting sheaf of lines is a function of the crystal angles, and the form of the crystal is reflected in the orientation of the lines in this sheaf. Choose three important faces as axial planes. These faces are always taken parallel to planes of symmetry, if there be such. The three intersections of the axial planes determine three non-parallel lines, and the three lines parallel to these intersections drawn through an arbitrary point of intersection are called the axes of the crystal. These axes may be all mutually perpendicular; two of them may be perpendicular to the third, but not perpendicular to each other; they may be equally inclined to each other, with the angle of inclination different from 90 or 60 deg.; or they may be mutually inclined with three different angles, all differing from 90 or 60 deg.

In addition to the three axial faces, a fourth fundamental face, intersecting the three axes, is chosen. The lengths of the segments so cut off from the three axes are expressed as ratios, the length of one of them being taken as unity. The lengths of the axes so determined may be equal for all axes; equal for two axes, but unequal for the third; or unequal for all three.

One class of crystals, showing a hexagonal cross section with 60-deg. angles between normals to the hexagonal sides, is most conveniently referred to four axes instead of the usual three. Three of the axes are at 60 deg. to each other and in the same plane, and the fourth is perpendicular to the plane of the other three.

The combinations of angles and lengths of the axes give rise to seven classes of crystals. These classes are:

1. **The Triclinic System.** Three mutually inclined and unequal axes, all three angles unequal, and other than 90, 60, or 30 deg.
2. **The Monoclinic System.** Three unequal axes, two of which are inclined, but the third is perpendicular to the other two.
3. **The Orthorhombic System.** Three unequal rectangular axes.
4. **The Tetragonal System.** Three rectangular axes, two of which are equal and different in length from the third.
5. **The Trigonal System.** Three equal and equally inclined axes.
6. **The Hexagonal System.** Three equal coplanar axes, inclined to 60 deg. to each other, and a fourth axis different in length from the other three and perpendicular to them.
7. **The Cubic System.** Three equal rectangular axes.

Liquid Crystals. While most substances are obtained in one (or more) crystalline forms, as liquid, and as vapor, some organic substances form another phase intermediate between crystal and liquid. This new phase is called the **liquid-crystal phase.** [See Friedel, *Ann. phys.*, (9), **18**, 272 (1922) and also Alexander, "Colloid Chemistry," Chap. 3, Reinhold Publishing Corp., New York, 1926.] Materials forming this phase melt at a definite temperature from a solid state to a cloudy viscous liquid, which, when further heated to some higher definite temperature, is transformed to a clear limpid liquid, which is regarded as the true liquid phase. The liquid-crystal phase is birefringent, *i.e.*, is anisotropic. The following substances are among the more than 250 substances which form liquid crystals: ethyl para-azoxybenzoate, ammonium oleate, para-azoxyphenetol, cholesteryl acetate.

Yield of a Crystallization Process. In many cases the process of crystallization is slow and the final mother liquor is in contact with a sufficiently large crystal surface so that the concentration of the mother liquor is substantially that of a saturated solution at the final temperature of the process. In such a case the yield of the process is calculated from the composition of the initial solution and the solubility of the material at the final temperature. If appreciable evaporation has taken place during the process, this must, of course, be known or estimated.

Solubility data are ordinarily given as parts by weight of anhydrous material per 100 parts by weight of total solvent, whether the crop contains water of crystallization or not.

When the rate of crystal growth is slow, a considerable time may be required to reach equilibrium. This is especially true where the solution is very viscous, or where the crystals collect in the bottom of the vessel so there is little crystal surface exposed to the supersaturated solution. In such cases the final mother liquor from the process may retain appreciable supersaturation, and the actual yield will be less than that calculated from the solubility curve unless considerable time is allowed for equilibrium to be reached. At any rate, the assumption that the mother liquor is a saturated solution gives the maximum yield of crystals that can be expected. The actual crop, after removal from the crystallizer, will in general retain some adhering mother liquor, which will give an increased weight.

In case the solid product is in the anhydrous form, the calculation of the yield is simple since the solid phase contains no water.

When the crop is hydrated, account must be taken of the water of crystallization in the crystals, since this water is withdrawn from the mother liquor and is not available for retaining the solute in solution.

The following formula can be used to calculate the theoretical yield of a crystallization process. It is valid for either hydrated or anhydrous crystals and assumes only that the mother liquor is saturated with solute at the final temperature, though this last restriction is removed if S (see below) is taken as the actual concentration of solute in the mother liquor at the end of the process. Equation (1) gives the weight of crystals as they exist in the final magma.

$$C = R\frac{100w_0 - S(H_0 - E)}{100 - S(R - 1)} \tag{1}$$

where C = weight of crystals in final magma.

$R = \dfrac{\text{molecular weight of hydrated solute}}{\text{molecular weight of anhydrous solute}}$.

S = solubility (parts by weight anhydrous solute per 100 parts by weight total solvent) of material at final temperature.

w_0 = weight of anhydrous solute in original batch.

H_0 = total weight of solvent in batch at the beginning of the process.

E = evaporation during the process.

Example 1. A 30 per cent solution of Na_2CO_3 weighing 10,000 lb. is cooled slowly to 20°C. The crystals formed are sal-soda ($Na_2CO_3.10H_2O$). The solubility of Na_2CO_3 at 20°C. is 21.5 parts of anhydrous salt per 100 parts of water. During cooling 3 per cent of the weight of the original solution is lost by evaporation. What is the weight of $Na_2CO_3.10H_2O$ formed?

Solution. Since the molecular weight of $Na_2CO_3.10H_2O$ is 286.2, and that of Na_2CO_3 is 106, $R = 286.2/106.0 = 2.70$. Also, $w_0 = (0.30)(10{,}000) = 3000$ lb.; the evaporation is $(0.03)(10{,}000) = 300$ lb.; and, therefore, $H_0 - E = 10{,}000 - 3000 - 300 = 6700$ lb. The weight of the crop is, by Eq. (1)

$$C = 2.70\left[\frac{100 \times 3000 - 21.5 \times 6700}{100 - 21.5(2.70 - 1.0)}\right]$$
$$= 6540 \text{ lb.}$$

The solubility data available in the literature are often old and inexact, even for pure substances. Solubilities may be influenced by impurities and by variations in the pH of the solution. Slight errors in the solubility of heavily hydrated solutes are sometimes magnified into larger errors in the calculated yields of such salts. If the yield of a crystallization process must be known with considerable precision, it is desirable to determine the solubility experimentally for the actual solute and solvent involved.

Purity of the Product. Although a crystal itself is necessarily pure, it retains mother liquor when removed from the final magma, and the adhering mother liquor will carry its share of the impurities present in the mother liquor. If the retained mother liquor is dried on the crystal, contamination will result.

In practice, crystals usually are centrifuged or filtered. Centrifuging leaves mother liquor amounting to 2 to 5 per cent of the weight of the crystals. Large, uniform crystals from low-viscosity mother liquors will retain a minimum proportion of mother liquor, while non-uniform, small crystals from viscous solutions will retain a considerably larger proportion. Comparable statements apply to the filtration of crystals. It is common practice to wash the crystals on the centrifuge or filter with fresh solvent; in principle such washing can reduce impurities to below almost any arbitrary figure. Purity can also be improved by recrystallization, but this method is not usually as satisfactory as that of properly washing the crystals.

Heat Effects in a Crystallization Process. The heat effect of a crystallization process is calculated by means of a heat balance. Such a balance can be computed by two methods: the individual heat effects, such as sensible heats, latent heats, and heats of crystallization, can be computed and combined into a balance equation, or an enthalpy balance can be taken in which the total enthalpy of all leaving streams minus the total enthalpy of all entering streams is equal to the heat absorbed from external sources by the process.

In the first method, the heat removed from the crystallizing solution by external means is equal to the sum of the sensible heat lost by the cooling solution and the heat evolved in the formation of the crystalline crop (heat of crystallization) minus the radiation losses and minus the heat of vaporization of solvent evaporated during the process.

Heat of Crystallization. In heat-balance calculations on crystallization processes the heat of crystallization is usually important. The heat of crystallization is the latent heat accompanying the precipitation of crystals from a saturated solution. Ordinarily the heat of crystallization is exothermic; it varies with both concentration and temperature. Rigorously the heat of crystallization is related to the heat of dilution of the solution and the heat of solution of the crystal. The heat of solution is the heat evolved when a unit mass of solid is dissolved in a very large amount of water, and such data are quite plentiful. A table of heats of solution is given on p. 1800. Heats of dilution, on the other hand, are scarce, especially for concentrated solutions, and it is usual to use the negative value of the heat of solution for the heat of crystallization. This is equivalent to neglecting heats of dilution. Ordinarily, the heat of dilution is small in comparison with that of solution, and the approximation is justified. Furthermore, the neglect of the heat of dilution leads to a conservative result because the heat of dilution is usually a heat evolution by the solution.

Example 2. The heat absorbed when 1 g. mol of $MgSO_4.7H_2O$ is dissolved isothermally at 18°C, in a large amount of water is 3180 cal. (p. 1802). What is the heat of crystallization of 1 lb. of $MgSO_4.7H_2O$ if heat of solution effects are negligible?

Solution. The molecular weight of $MgSO_4.7H_2O$ is 246.5. Since 1 cal. per g. mol = 1.8 B.t.u. per lb.-mol, the heat of crystallization of $MgSO_4.7H_2O$ is $\frac{(3180)(1.8)}{246.5}$ = 23 B.t.u. per lb.

Enthalpy-concentration Chart. The enthalpy method of calculating heat balances over crystallization processes is facilitated by the use of the enthalpy-concentration chart so constructed as to show the solid phases [Bošnjaković, *Z. ges. Kälte-Ind.*, **39**, 182 (1932)]. This method rigorously accounts for the heats of dilution and is very simple arithmetically, once the chart has been constructed. The disadvantages of the enthalpy-concentration chart are as follows: (1) considerable data are required for its construction, and these data are often not available; (2) the initial construction of the chart is time consuming and not justified for a single calculation. For substances commonly crystallized and for which adequate data are available, the enthalpy-concentration chart has considerable utility.

An enthalpy-concentration chart for the system $MgSO_4.H_2O$ is shown in Fig. 1. The use of the chart in heat-balance calculations involving solutions has been described [McCabe, *Trans. Am. Inst. Chem. Engrs.*, **31**, 129 (1935)]. In Fig. 1 the enthalpies of the solid phases from zero to 50 per cent $MgSO_4$ are shown, and the diagram can be correlated with the ordinary phase diagram shown in Fig. 2. The line *pa* represents the freezing points of ice

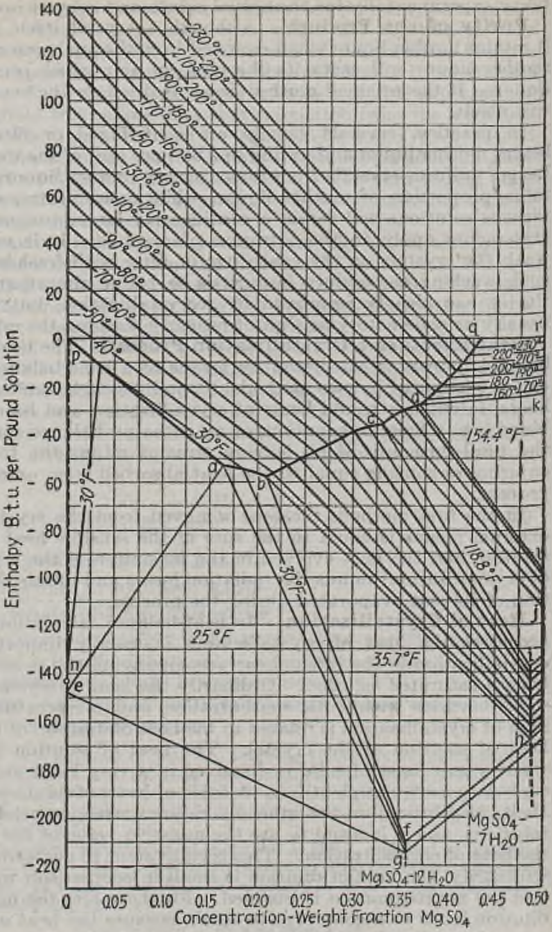


FIG. 1.—Enthalpy-concentration chart for $MgSO_4.H_2O$. For remainder of legend see page 1763.

from solutions of $MgSO_4$. Point *a* is the eutectic, and line *abcdq* is the solubility curve of various hydrates. Line *ab* is the solubility curve for $MgSO_4.12H_2O$, *bc* is the solubility curve for $MgSO_4.7H_2O$, *cd* is the solubility curve for $MgSO_4.6H_2O$, and *dq* is a portion of the curve for $MgSO_4.H_2O$. The area *aep* of Fig. 1 represents the enthalpies of all equilibrium mixtures of ice and $MgSO_4$ solution. The isothermal (25°F.) triangle *age* gives the enthalpies of all combinations of ice and partially solidified eutectic and of $MgSO_4.12H_2O$ and partially solidified eutectic. Area *abfg* contains the enthalpy-concentration coordinates of all magmas consisting of $MgSO_4.12H_2O$ crystals and its mother liquor. The isothermal (35.7°F.) area *bhf* represents the isothermal transformation of $MgSO_4.7H_2O$ to $MgSO_4.12H_2O$, and this area represents mixtures consisting of a saturated solution of concentration 21 per cent, solid $MgSO_4.7H_2O$, and solid $MgSO_4.12H_2O$. The area *cihb* represents all magmas of $MgSO_4.7H_2O$ (Epsom salt) and its mother liquor. The isothermal (118.8°F.) area *cji* represents mixtures consisting of a saturated solution containing 35 per cent $MgSO_4$, solid $MgSO_4.6H_2O$, and solid $MgSO_4.7H_2O$. Area *dljc* represents magmas of $MgSO_4.6H_2O$ and its mother liquor. The isothermal (154.4°F.) area *dkl* represents mixtures consisting of a saturated solution containing 37 per cent $MgSO_4$, solid $MgSO_4.H_2O$, and solid $MgSO_4.6H_2O$. Area *qrkd* is a part of the field representing

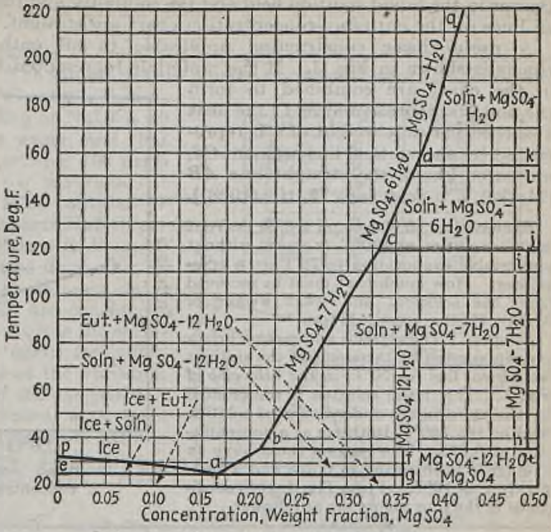


Fig. 2.—Phase diagram for $MgSO_4.H_2O$.

Legend for Figure 1.

pae = ice + solution
age = ice + eutectic solution, or $MgSO_4.12H_2O$ + eutectic solution
abfg = $MgSO_4.12H_2O$ + solution
bhf = solution + $MgSO_4.7H_2O$ + $MgSO_4.12H_2O$
bcih = $MgSO_4.7H_2O$ + solution
cji = solution + $MgSO_4.6H_2O$ + $MgSO_4.7H_2O$
dkl = solution + $MgSO_4.H_2O$ + $MgSO_4.6H_2O$
dljc = $MgSO_4.6H_2O$ + solution
dkrq = $MgSO_4.H_2O$ + solution
above *pabcdq* = solution only
below *eg* = ice + eutectic, or eutectic + $MgSO_4.12H_2O$
below *fh* = $MgSO_4.12H_2O$ + $MgSO_4.7H_2O$

saturated solutions in equilibrium with $MgSO_4.H_2O$. Except for the isotherms in the liquid solution field and the solubility and freezing point curves, all lines on the enthalpy-concentration chart are straight.

A useful basic construction applicable to all enthalpy-concentration charts is shown in Fig. 3. If the materials represented by points A and B in the chart are combined to form the material represented by C, the heat absorbed per unit weight of C is represented by the vertical line segment Cd, measured above the straight line AB [Merkel, *Z. v. deut. Ing.*, **72**, 109 (1928)].

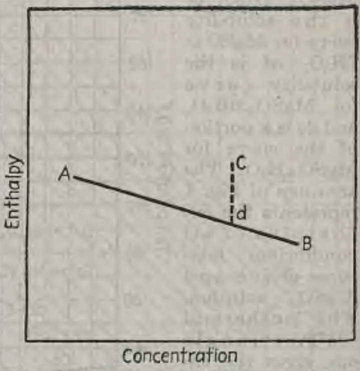


FIG. 3.—Basic construction, enthalpy-concentration chart.

Example 3. 10,000. lb. of a 32.5 per cent $MgSO_4$ solution at 120°F. is cooled without appreciable evaporation to 70°F. in a crystallizer. How much heat must be removed from the solution, and what weight of $MgSO_4.7H_2O$ crystals will form?

Solution. The crystals and mother liquor are represented by terminals of the straight isothermal line for 70°F. in the field *cihb* of Fig. 1. The initial solution is represented by the point in the undersaturated solution field on the 120°F. isotherm at a concentration of 0.325. The magma must have an average concentration of 0.325 and a temperature of 70°F. From Fig. 1, the coordinates of the four points are as follows:

	Temp., °F.	Conc.	Enthalpy
Original solution	120	0.325	−33.0
Crystals	70	.488	−51.0
Mother liquor	70	.259	−47.3
Magma	70	.325	−78.4

The heat removed from the solution is

$$10{,}000(-33.0 + 78.4) = 454{,}000 \text{ B.t.u.}$$

The same result is obtained if the "basic construction" is used. The original solution can be considered the result of a combination of mother liquor and crystals, and the vertical distance to the point of the original solutions above the line connecting the points representing the mother liquor and crystals is the heat absorbed due to such a combination, or the heat evolved from the separation of solution into crystals and mother liquor.

The yield of crystals is easily obtained by applying the "lever-arm principle" commonly used in calculations involving equilibrium diagrams.

$$\text{Weight of crystals} = 10{,}000\,\frac{0.325 - 0.259}{0.488 - 0.259} = 2880 \text{ lb.}$$

Fractional Crystallization. When two or more solutes are present in a solution, it is often possible to crystallize one of the solutes and leave the others in solution. Usually, such fractional crystallization methods are based on differences in the solubilities of the solutes.

It is a highly important fact that the solubility of a material in a solution of another solute is, in general, quite widely different from its solubility in the pure solvent. Thus, the solubility of sodium chloride at 20°C. is 36 parts per 100 parts water, and that of sodium nitrate is 88 parts per 100 parts water, but a solution saturated at 20°C. with respect to both of these salts will contain only 25 parts sodium chloride and 59 parts sodium nitrate per 100 parts water.

The mutual solubilities of the above two salts can be shown diagrammatically, as in Fig. 4. The solubilities are plotted for two different temperatures: line *DEF* is for 100°C., at which temperature the solubility of NaCl is 40 parts per 100 parts water, and that of $NaNO_3$ is 176 parts per 100 parts water. A solution saturated at 100°C. with both salts contains 17 parts NaCl and 160 parts $NaNO_3$ per 100 parts water. Points *D*, *E*, and *F* are plotted from these data, and, in the absence of more detailed data, the lines *DE* and *EF* are considered straight. The line *ACB* is the corresponding solubility curve for 20°C., and the line *EC* shows the variation with temperature of the composition of a solution saturated with both components. (Badger and Baker "Inorganic Chemical Technology," p. 82, McGraw-Hill, New York, 1928.)

Coordinates of Points

Point	Na Cl	Na NO_3
A	36	0
B	0	88
C	25	59
D	40	0
E	17	160
F	0	176
G	17	68

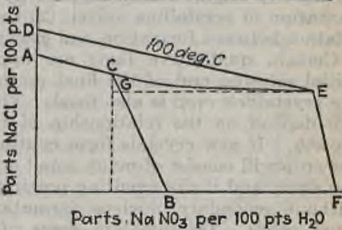


FIG. 4.—Fractional crystallization of NaCl and $NaNO_3$.

If a solution at 100°C. has a composition represented by a point on the line *DE*, the solution is saturated with respect to NaCl but not with respect to $NaNO_3$; while, if the composition of the solution is represented by a point on line *EF*, the solution is saturated with $NaNO_3$ but not with NaCl. (For a detailed treatment of such solubility relationships, see Blasdale "Equilibria in Saturated Salt Solutions," Reinhold, New York, 1927.)

Example 4. As an illustration of fractional crystallization, consider the separation of $NaNO_3$ and NaCl from a solution saturated at 100°C. with both salts and therefore represented by point *E*. If a basis of 100 lb. of water is taken, the solution contains 17 lb. NaCl and 160 lb. $NaNO_3$. Suppose the solution is cooled to 20°C. The solution becomes supersaturated with respect to $NaNO_3$, and crystallization of the latter should take place. The composition of the solution moves along the path *EG*. At 20°C., if equilibrium is reached, the composition of the solution is that represented by point *G*. If line *CB* is considered straight, the abscissa of *G* can be calculated from similar triangles as follows:

$$\text{Parts NaNO}_3 = 59 + \frac{(88 - 59)(25 - 17)}{25} = 68.3$$

On cooling along the line *EG*, there will separate 160 − 68.3 = 91.7 lb. $NaNO_3$, and all the NaCl will remain in solution. If the solution is now evaporated at 100°C. until the $NaNO_3$ concentration is brought back to 160 parts per 100 parts water, NaCl will be precipitated during the evaporation and can be removed. The concentration of the solution will again be represented by point *E*, and the cycle repeated. On each cooling, $(91.7/160)(100) = 57.3$ per cent of the nitrate in solution will crystallize, and, on each evaporation, the same percentage of the chloride will be precipitated. Various modifications of the method can be used. For example, the amount of water in the batch can be kept constant and the solution resaturated at 100°C. with fresh $NaNO_3$ after each cooling. The hot solvent will act as a selective solvent for the nitrate, since it is impoverished in $NaNO_3$, but saturated with respect to NaCl. Nitrate can therefore be dissolved and chloride left behind. The dissolved nitrate is recovered in the cooling part of the cycle.

Another method of fractional crystallization is sometimes used, where advantage is taken of different crystallization rates. Thus, a solution saturated with borax and potassium chloride will, in the absence of borax seed crystals, precipitate only potassium chloride on rapid cooling. The borax remains behind as a supersaturated solution, and the potassium chloride crystals can be removed before the slower borax crystallization starts.

CRYSTAL FORMATION

There are obviously two steps involved in the preparation of crystalline matter from a solution. The crystals must first form and then grow. The theory can therefore be conveniently considered under three heads: (1) the formation of crystalline nuclei, (2) their resulting growth, and (3) the interrelation between formation and growth.

Certain qualitative facts are apparent. If the concentrations of the initial solution and of the final mother liquor are fixed the total weight of the crystalline crop is also fixed. The distribution of this weight, however, will depend on the relationship of the two processes of formation and of growth. If new crystals form continuously and rapidly during the process, the crop will consist of many small crystals, while if but a few nuclei form at the start, and if the resulting precipitation occurs uniformly on these nuclei without secondary nucleus formation, a crop of large uniform crystals must result. Intermediate cases of simultaneous formation and growth will of course, result in intermediate average size and also a non-uniform grain, since the older crystals will be larger than the younger ones.

If the laws and data for crystal formation and growth were completely known, it should be possible to predict the size range of the crystals produced in a crystallization process. Such information is, however, at present incomplete. The following discussion is an attempt to summarize the existing knowledge that is of value in attacking problems of industrial crystallization and in analysing the operating principles of industrial crystallizers.

Supersaturation. Both crystal formation and crystal growth have a common driving force, namely, supersaturation. Unless a solution is supersaturated, crystals can neither form nor grow. The supersaturation required in a given case may be so small that it is scarcely measurable, or it may be so large that the solution is as much as 30 per cent more concentrated than called for by the solubility curve. Supersaturation affects the formation of crystals in a radically different manner than it does crystal growth, and the action of supersaturation will be discussed separately for the two processes.

Seeded vs. Unseeded Crystallization. Crystal formation can occur under either of two circumstances. The first case is where crystals form in a solution that has been carefully freed of all solid particles. Crystallization from such a solution is said to take place from an unseeded solution. The presence of dust particles, small solute crystals, or in some cases crystals of other materials, may lead to a second type of crystal formation, namely, that from seeded solutions. In practice, the seeded case is the more important. Except for closed batch crystallizers in which the solutions are heated well above the saturation temperatures before being sent to the crystallizers, completely unseeded solutions are not usually encountered. Batch vacuum crystallizers and special crystallizers, such as sugar-boiling apparatus, fall into the category of equipment that operates on unseeded solutions. On the other hand, crystallizers in which the solution has access to the atmosphere of the plant will in all probability be seeded by the plant dust which invariably will carry tiny crystals and dust particles and which will inoculate the solution. Even a solution that is unseeded at the start of the process becomes seeded the moment crystals form, and at all stages beyond the initial nucleation the process is of the seeded type.

In the case of an unseeded solution, it is possible in viscous solutions of relatively high molecular weight, such as those of sugars, to maintain a highly supersaturated solution indefinitely without the formation of nuclei. To obtain this result the solution must be carefully prepared, must be entirely free from dust particles, and must be "sterilized" or heated well above its saturation temperature for a time in a completely air-tight container. Materials of low molecular weight which form solutions of moderate viscosities cannot usually support supersaturations of any great magnitude for an indefinite period. There are undoubtedly borderline solutes to which a definite answer cannot be given concerning their ability to stay in supersaturated solutions indefinitely.

Miers' Theory. Miers and his collaborators, following an earlier suggestion by Ostwald, have elaborated the theory that there is a definite relationship between the concentration and the temperature at which crystals will spontaneously form in an initially unseeded solution. This relationship takes the form of a so-called supersolubility curve which is roughly parallel to the usual solubility curve and is located in the supersaturated field. The Miers theory, which has considerable experimental support, states that under normal conditions appreciable spontaneous formation of crystals will not occur in the area between the solubility curve and the supersolubility curve but that, whenever the concentration is brought into the labile field in which concentrations are higher than those corresponding to the supersolubility curve, sudden and copious nucleation will occur.

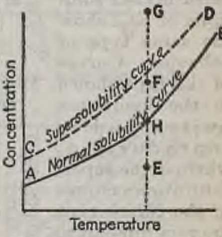


FIG. 5.—Diagrammatic representation of Miers' theory.

These relationships are shown diagrammatically in Fig. 5 wherein concentration is plotted against temperature. The normal solubility curve is shown as line AB, and the supersolubility curve is the dotted line CD. A solution at such a temperature and concentration that it is represented by a point below AB (point E, for example) is undersaturated. A solution represented by point F is metastable and will drop to concentration H if seed crystals are added to it, but it will remain at F unless seeds are present. A solution represented by point G will spontaneously crystallize, as its concentration will fall to that of point H.

In spite of considerable controversy concerning such questions as to the effects of mechanical stimulation and the effect of probability, the Miers type of supersolubility curve has been a useful concept in the analysis of crystal formation problems from unseeded solutions.

Most of Miers' work was carried out on solutions that were initially unseeded. Since most industrial cases concern seeded solutions, it is important to determine whether or not seeded solutions can be maintained for appreciable lengths of time under such conditions that nuclei other than the seeds themselves do not form. Work carried out on the supersolubility relationships of seeded solutions has shown that, if a solution initially without seeds is cooled at a definite rate and if definite quantities of a definite size of seeds are added when the concentration of the solution reaches saturation, there is a definite degree of undercooling corresponding to the formation of new nuclei. Other reproducible supersaturation curves are found at which the rate formation of crystals becomes a maximum. Results of such experiments on $MgSO_4.7H_2O$ are shown in Fig. 6. The t_1 curves show where new nuclei first formed, and the t_2 curves show where the rate of formation reaches a maximum. Supersaturated solutions of KCl show the same type of behavior. Curves of the type shown in the figure are quite readily reproducible, whereas the supersaturation curves of the Miers type are very difficult to reproduce except under very carefully controlled conditions, as it is very difficult to prevent the effects of fortuitous seeding. The supersaturations obtained in seeded solutions are definitely considerably less than those found in crystallizing unseeded solutions.

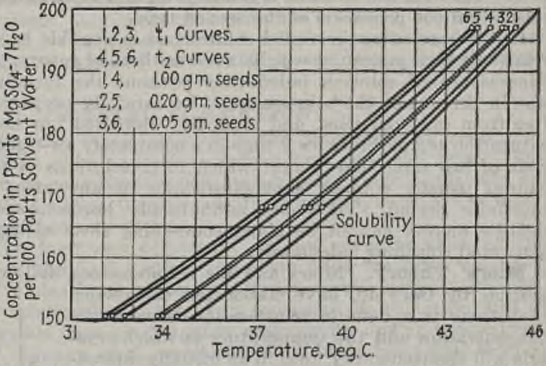


FIG. 6a.—Supersaturation curves, seeded solutions. Effect of weight of seed crystals. Grams seeds per 1000 g. solution.

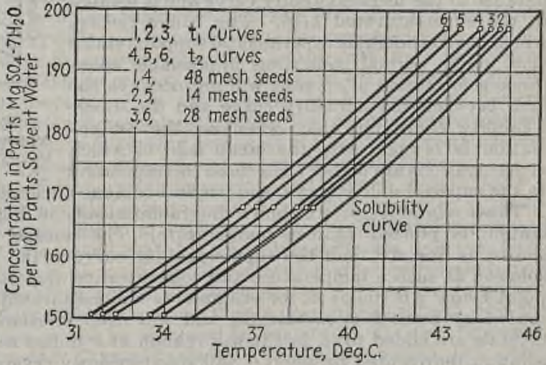


FIG. 6b.—Supersaturation curves, seeded solutions. Effect of size of seed crystals.

These results indicate that the ability of crystals to inoculate a solution and to cause the formation of new crystals is an important fundamental factor in crystallization.

Curves such as those of Fig. 6 suggest the question whether or not it is possible to maintain a supersaturated solution in contact with seed crystals for an indefinite period without the formation of new nuclei. Experimental results indicate that it is not possible, even at low supersaturations, to suppress crystal formation indefinitely in a seeded solution. The lower the supersaturation, the slower will be the formation of new nuclei, and the longer will be the time that will elapse before new nuclei are formed, but, if time enough is allowed, present knowledge indicates that sooner or later new nuclei will form. At very low supersaturations, where the material available for crystal formation is small, but few nuclei can be formed before the solution reaches equilibrium.

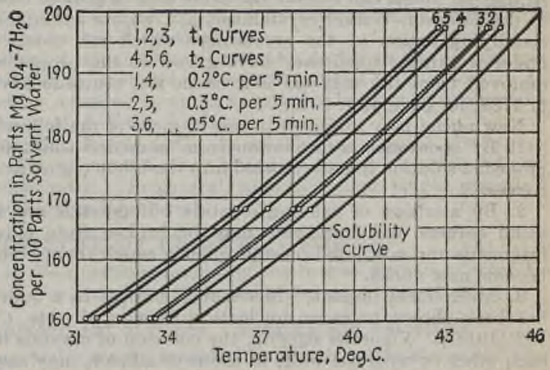


Fig. 6c.—Supersaturation curves, seeded solutions. Effect of cooling rate.

If the rate of nucleus formation is very low and if the rate of growth is reasonably high in comparison with the formation rate, it is possible to maintain a seeded supersaturated solution without nucleation long enough so that the original seeds will grow to the desired size before the new nuclei appear. The crystallization of sugar is an example of such a case. In sugar crystallization the nuclei are formed almost instantly by a sudden increase in supersaturation. Once the initial batch of nuclei is formed, supersaturation is maintained at a point low enough that the formation of additional nuclei does not take place before the original crystals have been allowed to grow to the desired size. Apparently, this method is not possible in the case where nucleation tends to be too rapid to allow time enough for the original seeds to grow to the desired size.

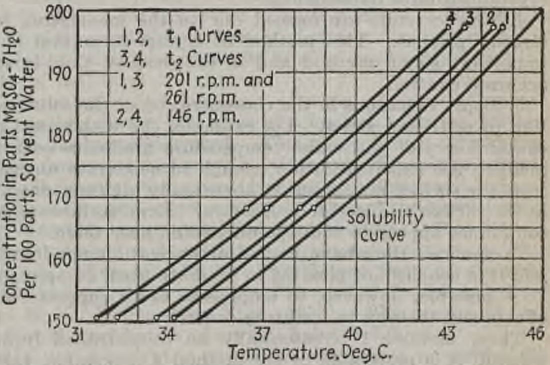


Fig. 6d.—Supersaturation curves, seeded solutions. Effect of stirring rate.

Methods of Forming Crystals in Solutions. In any crystallization process the nuclei formation should be under control. In a batch process, if uniform crystals are desired, it is advisable to form as large a proportion as possible of the crystals at the same time, even if additional nucleation cannot be altogether prevented. Otherwise, a non-uniform crop will be obtained. In a continuous process the number of nuclei formed per unit time will be continuous and uniform and must equal the number of crystals that are withdrawn per unit time from the crystallizer. In a continuous crystallizer either most of the nuclei should form within a narrow zone in the unit so that all nuclei can receive the same time of growth (the method applicable to the Swenson-Walker crystallizer, p. 1783, for example) or there must be a classifying action in the crystallizer which will retain the small crystals under treatment until they have grown to the proper size before they are removed from the unit (as is done in the continuous vacuum crystallizer, p. 1787, for example).

New nuclei may originate in one or more of the following ways:

1. By spontaneous nucleation from unseeded solutions. In this case an unseeded solution must be cooled into the labile region as shown by the Miers curves.
2. By attrition of existing crystals. If crystals are agitated vigorously, small corners and fragments may be broken from existing crystals; such fragments and mutilated crystals quickly repair themselves and the fragments become new nuclei.
3. Mechanical impact. Mechanical impact in a supersaturated solution has been shown to cause nucleation [Young, *J. Am. Chem. Soc.*, **33**, 148, 162 (1911)]. Vigorous stirring, the collision of crystals in the solution, with each other or with the walls of the crystallizers, may cause the formation of some new nuclei. This formation is over and above that resulting from the mechanical fracture of existing crystals. Its importance in industrial crystallization is questionable.
4. New crystals are formed due to the inoculating influence of crystals already present. This method of crystal formation is probably the most important single method and is the method that is subject to the most accurate control.
5. Local variations in the concentration of the solution may cause nucleation in restricted zones. For example, the withdrawal of heat through the containing wall will cause temperature gradients near the wall which can increase the supersaturation enough to accelerate nucleation. Evaporation from the surface may result in abnormally high concentrations in the solution at the surface and lead to nucleation. Even surfaces at solution temperature sometimes appear to catalyze nucleation near them.

In general, the above causes of nucleation are inextricably interwoven, and it is usually not possible to separate them completely in any given case. It is possible, however, to emphasize or to suppress individual nucleation effects and thereby to facilitate control.

Thus, method 1 is essentially an uncontrolled formation method. In general, it is preferable to use method 4 if possible, rather than method 1. Figure 16 [Seavoy and Caldwell, *Ind. Eng. Chem.*, **32**, 633 (1940)] shows the temperature-concentration history of a batch crystallizer operated two ways: first, with an unseeded solution and, second, with a seeded solution. In the second case a high supersaturation, characterized by uncontrolled nucleation, is prevented.

The formation of crystals by attrition (method 2) should be suppressed as much as possible. Such formation occurs at the expense of existing perfect crystals and is not subject to adequate control. For practical control purposes, methods 3 and 4 can be considered as equivalent. The mechanical impact of stirrers or of crystals on each other accounts for only a small proportion of the total nucleation and depends on stirring rate and the number and size of crystals existing in the equipment at any given time. The much more important inoculating effect is determined by these same variables. Increased stirring rate, for example, brings about more uniform distribution of crystals in the crystallizing solution, which will increase their inoculating influence. Such effects increase nucleation rate more than they do growth rate and are the logical explanation of the general observation that stirring tends to cause small crystals.

Nucleation by method 5 depends largely on the design of the crystallizer and the rate at which it is operated. For best control it is advisable to suppress this method by reducing as much as possible local variations in the temperature or concentration of the solution.

Effects of Impurities. Impurities in the solution may inhibit the formation of new nuclei. The effect of a given impurity cannot be predicted and must be found by experiment. In general, high molecular-weight materials seem to be the most effective inhibitors.

GEOMETRY OF CRYSTAL GROWTH

As a preliminary to the questions involving the rate of growth of crystals, certain facts regarding the geometry of crystal growth are important.

Parallel Displacement of Faces. Translation Velocities

Geometrically, a crystal is a solid bounded by planes. The shape and size of such a solid are a function of the interfacial angles and of the linear dimensions of the faces. As a result of the constancy of its interfacial angles, each face of a growing or dissolving crystal, as it moves away from or toward the center of the crystal, is always parallel to its original position. This fact is known as the *principle of the parallel displacement of faces*. The rate at which a face moves in a direction perpendicular to its original position is called the **translation velocity** of that face.

Crystal Habit. From an industrial point of view, the term "crystal habit" refers to the relative sizes of the faces of a crystal. No general law controlling crystal habit has been discovered. This property is easily affected by conditions of crystal formation and growth. It is very difficult to prepare perfect crystals with all faces of the same form equally developed. Small amounts of foreign substances will often completely change the crystal habit of a material. For example, sodium chloride crystallizes as cubes from a pure solution but forms octahedra if precipitated from a solution containing urea. The selective adsorption of dyes by the different faces of a crystal can greatly modify the habit of the crystal (see France, "Colloid Symposium Annual," vol. 7, pp. 59–87, Wiley, New York, 1930). Phenomena of this kind are so general that the prediction of crystal habit is difficult.

Invariant Crystals. Although it is impossible at present to predict the crystal habit of a definite material, it is reasonable to assume that under constant external conditions the relative translation velocities of the different faces of a crystal do not vary during the growth of the crystal. Such a crystal is known as an **invariant crystal.**

The most important property of an invariant crystal is the fact that such a crystal remains geometrically similar to its original shape as it grows or dissolves.

Overlapping Principle. Since the relative sizes of the individual faces of a crystal vary between wide limits, it follows that different faces must have different translation velocities. A geometric law of crystal growth known as the **overlapping principle** is based on these velocity differences and may be stated as follows: In the growing of a crystal, only those faces having the lowest translation velocities survive, and, in the dissolving of a crystal, only those faces having the highest translation velocities survive. For example, consider cross sections of a growing crystal, as in Fig. 7. The polygons shown in the figure represent varying stages in the growth of the crystal. The faces marked *A* are slow-growing faces (low translation velocities), and the faces marked *B* are fast growing (high translation velocities). It is apparent from Fig. 7 that the faster *B* faces tend to disappear, as they are "overlapped" by the slower *A* faces.

It has been shown [Valeton, *Z. Kryst.*, **59**, 135–169 (1923)] that the overlapping principle, if combined with the principle of the parallel displacement of faces, makes it possible to predict the final shape of a crystal when the initial shape and the relative translation velocities of the faces are known.

FIG. 7.—Overlapping principle.

Variation of Translation Velocities. The translation velocities of the faces of an invariant crystal are not all equal, unless the crystal is a regular geometric solid. The smaller faces have greater rates of growth, measured in weight per unit area per unit time, than have the larger faces. This difference in rate has been found experimentally.

CRYSTAL GROWTH

The earliest theory of crystal shape was the thermodynamic theory of Gibbs and Curie. This theory stated that a crystal, as it grows, chooses that form compatible with the symmetry of the crystal that gives the minimum surface energy. For commercial sizes, however, the surface-energy differences between a thermodynamically stable crystal and one that is unstable because of its shape are too small to be of importance.

Rate of Solution. The first theories of the rate of crystal growth consisted of attempts to consider crystallization as the reverse of solution. Earlier work by Noyes and Whitney [*Z. physik. Chem.*, **23**, 689 (1897)] and others led to the establishment of a theory of the rate of solution that has been applied to a number of other heterogeneous reaction-velocity problems. The theory assumes that such reactions are controlled by the rate of diffusion of the reactants and products to and from the solid-liquid interface. Any reactions taking place at the interface are very rapid in comparison with the diffusion rates. The diffusional resistance is confined to a comparatively thin film surrounding the solid, since convection currents equalize the concentrations in the bulk of the liquid. Low viscosity and vigorous stirring attenuate the film, decrease the diffusional resistance, and increase the reaction rate; while high viscosity and poor agitation decrease the reaction rate because the films are then relatively thick and the diffusion process is slow.

The Berthoud-Valeton Theory [*J. chim. phys.*, **10**, 625 (1912). *Z. Kryst.*, **59**, 335–365 (1923); **60**, 1–38 (1924)]. This theory assumes that the diffusional process is followed in series with a first-order interfacial reaction and the net rate of crystallization depends on both reactions. If the concentration of a saturated solution is C_0, that of the bulk of the solution is C, and that of the solution in contact with the crystal surface is C', the rate of material deposition is

$$\frac{dW}{d\theta} = k'S(C' - C_0) = \frac{kS}{L}(C - C') \qquad (2)$$

where W is the weight, θ is the time, k is the diffusion coefficient, k' is the rate of reaction constant of the interfacial reaction, S is the surface area, and L is the effective film thickness.

If C' is eliminated from Eq. (2), the result may be written:

$$\frac{dW}{d\theta} = \frac{S(C - C_0)}{\dfrac{1}{k'} + \dfrac{L}{k}} \qquad (3)$$

There are three corollaries of Eq. (3):

1. The over-all reaction is of the first order, since for any particular face at constant temperature $1/(1/k' + L/k)$ is a constant. (This will not be true, however, if the interface reaction is not first order.)

2. Although the reaction is one of first order, the value of k' varies from face to face, and so the rates of growth of the individual faces of the same crystal can vary, in spite of the fact that k (the diffusion film coefficient) is constant for all the faces.

3. The relative magnitudes of k' and k may vary in different cases. If k is large in comparison with k', diffusion is an unimportant factor and the surface reaction controls the process of crystallization. If k' is large in comparison with k, the surface reaction has but little influence on the rate, and the diffusion then is the controlling factor. If k' is very large in comparison with k, the theory becomes identical with the Noyes-Whitney theory.

Effect of Impurities on Crystal Growth. Small amounts of impurities may have an important effect on the rate of crystal growth, just as they do on the rate of crystal formation. Apparently, the inhibition of growth is due to the adsorption of the impurity on the crystal face. No general rule governing these phenomena has been discovered. The amount of impurity adsorbed depends not only on the material and the impurity but varies from face to face of the same crystal. The face that adsorbs the greatest amount of impurity will have the lowest translation velocities and hence will increase in size relative to the other faces, in accordance with the overlapping principle (France, "Colloid Symposium Annual," vol. 7, pp. 59–87, Wiley, New York, 1930).

The ΔL Law. It has been shown [McCabe, *Ind. Eng. Chem.*, **21**, 30, 112, (1929)] that all geometrically similar crystals of the same material suspended in the same solution grow at the same rate, if the growth is measured as the increase in length of geometrically corresponding distances on all of the crystals. If ΔL is the increase in linear dimension of one crystal, it is at the same time equal to the increase in the corresponding dimension of each of the other crystals and is independent of the initial size of any of the original crystals, provided all crystals in the suspension are treated exactly alike.

Calculation of Screen Analysis of Product from That of Seeds. The ΔL law gives a solution to the following problem:

Problem. Given a saturated solution in which is suspended a known weight of seed crystals of known screen analysis, and assuming this solution to be cooled under known conditions, what will be the weight and screen analysis of the crystals at the end of the process, if there is negligible formation of new nuclei?

The weight of material precipitated is calculated in the same manner as is the yield of a crystallization process (see p. 1760). The distribution of the precipitating material takes place in accordance with the ΔL law. If D is the size of the opening of a sieve that will just pass a given crystal, then D and L can be considered proportional, or

$$\alpha D = L \tag{4}$$

and

$$\alpha \, \Delta D = \Delta L \tag{5}$$

where α is a proportionality constant, that is identical for all crystals of the batch. If ΔL is the same for all crystals, ΔD is also the same for all crystals. It has been shown that

$$W_p = \int_0^{W_s} \left(1 + \frac{\Delta D}{D_s}\right)^3 dW_s \tag{6}$$

where W_p is the weight of product obtained from W_s g. seed crystals, D_s and W_s are the coordinates of the cumulative screen analysis curve (where W_s is total weight retained on the screen of opening size D_s), and ΔD is defined in Eq. (5).

The steps in the calculation of the yield and screen analysis of the product of a crystal-growth process are:

1. Calculate the theoretical yield from the ratio of seeds to solution and the solubility change of the material during the process.
2. Assume a value of ΔD, and calculate the weight of the product that corresponds to this assumed value by integrating Eq. (6) over the range $W_s = 0$ to $W_s = 100$.
3. If the value of W_p calculated in step 2 does not check that calculated in step 1, adjust ΔD by trial and error, until fair agreement is reached between the weights of product calculated by the two methods.
4. Using the correct value of ΔD as determined in step 3, plot the integral curve of Eq. (6).
5. Plot W_p against D_p. This plot can be constructed from the data at hand at this point, since the integral curve, plotted in step 4, gives the relationship between W_p and W_s; the screen-analysis curve of the seeds exhibits D_s as a function of W_s; and $D_p = D_s + \Delta D$.
6. From the curve of W_p against D_p that was plotted in step 5, read off the values of W_p that correspond to the sizes of the various screen openings, and convert the values to percentages of the entire product. The result is the cumulative screen analysis of the product. The differential analysis is easily derived from the cumulative analysis by subtraction.

Example 5. A Swenson-Walker (see p. 1783) crystallizer is cooling a potassium chloride solution. It discharges a saturated solution at 60°C. The discharge consists of 1750 lb. saturated solution and 165 lb. crystals per hour. The screen analysis of the crystals is as follows:

Table 1. Screen Analysis of Seeds

Meshes per inch	Size of screen opening, cm.	Screen analysis, %	
		Differential	Cumulative
On 12	0.1397	0.0	0.0
14	0.1168	0.1	0.1
16	0.0991	2.9	3.0
20	0.0833	12.7	15.7
24	0.0701	13.0	28.7
28	0.0589	25.8	54.5
32	0.0495	19.6	74.1
35	0.0417	13.3	87.4
42	0.0351	6.3	93.7
48	0.0295	3.6	97.3
60	0.0246	1.0	98.3
65	0.0208	1.2	99.5
Through 100	(0.0175)*	0.5	100.0

* Estimated.

What would be (*a*) the weight and (*b*) the screen analysis of the product if additional sections were added to the crystallizer to cool the above product from 60° to 30°C.?

The solubility of potassium chloride at 60°C. is 45.0 parts per 100 parts water, and the solubility at 30°C. is 37.35 parts per 100 parts water.

Solution. The answer to question *a*, namely the new crop weight, is computed from the solubility data. Since the potassium chloride that precipitates is in the anhydrous form, and since the solubilities given above are based on weight of water, the increased yield due to the cooling from 60° to 30°C. is $45.0 - 37.35 = 7.65$ lb. per 100 lb. water. Since the crystallizer handles $(100/145)(1750) = 1206$ lb. water, the increase in yield is $(12.06)(7.65) = 92.3$ lb. The product from the new section will then be $92.3 + 165 = 257.3$ lb. crystals per hour. This is the answer to question *a*.

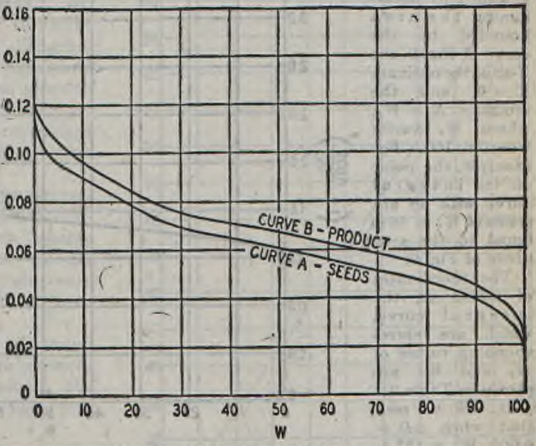


FIG. 8.—Example 5: Screen analyses.

In order to determine the screen analysis of the new product (question *b*), the procedure of pp. 1773 to 1774 is applied. The screen analysis of the seed crystals is given in the data of the problem. The second and fourth columns of Table 2 are D_s and W_s, respectively, based on 100 lb. seed crystals. The product is $(257.3/165)(100) = 156$ lb. on this same basis. Step 1 of the procedure has been completed.

The second step is the trial-and-error determination of the correct value of ΔD that corresponds to the required weight increase of 56 lb. per 100 lb. seeds. This is done by assigning a value to ΔD, evaluating the integral of Eq. (6) between the limits $W_s = 100$ and $W_s = 0$, and thereby calculating a corresponding value of W_p. The required value of ΔD is that value that gives a value of W_p of 156.

Without reproducing the preliminary trials, it is found that if ΔD is 0.009, the yield requirement is closely met. This value will be shown to be correct in the next step.

Table 2. Coordinates of W_s *vs.* $(1 + 0.009/D_s)^3$ Curve

W_s	D_s	$\left(1+\frac{0.009}{D_s}\right)^3$	W_s	D_s	$\left(1+\frac{0.009}{D_s}\right)^3$
0.0	0.1397	1.21	50	0.0611	1.51
0.1	.1168	1.25	60	.0563	1.56
3.0	.0991	1.30	70	.0514	1.62
5.0	.0960	1.31	80	.0466	1.70
10.0	.0901	1.33	90	.0391	1.86
20.0	.0789	1.38	95	.0330	2.06
30.0	.0695	1.44	98	.0270	2.37
40.0	.0653	1.47	100	.0175	3.47

The third step is to substitute $\Delta D = 0.009$ in Eq. (6) and to calculate the integral curve of the equation. This is done by plotting values of $(1 + 0.009/D_s)^3$ as ordinates against values of W_s as abscissas. The relationship between D_s and W_s is obtained by plotting the second column of the screen-analysis table (Table 1) against the fourth column. This curve is curve *A* of Fig. 8. The coordinates of the $(1 + 0.009/D_s)^3$ *vs.* W_s curve are as shown above and the curve is shown in Fig. 9.

The integral curve of Eq. (6) is obtained by determining the area bounded by the curve of Fig. 9, the *X*-axis, the ordinate $X = 0$, and the ordinate $X = W_s$, where W_s varies from 0 to 100. For example, the point on the integral curve with an abscissa of $W_s = 50$ is equal to the area *abcda* of Fig. 9.

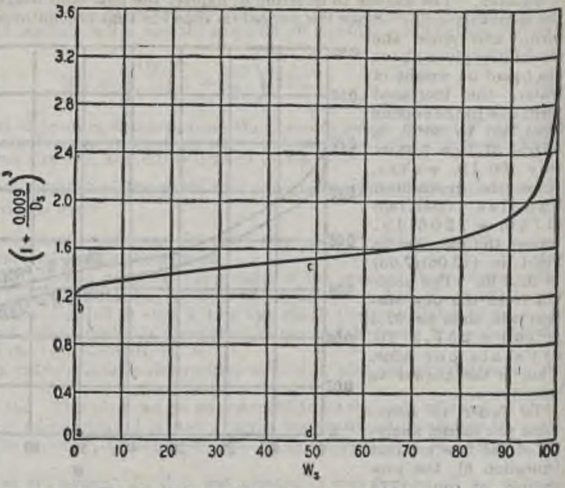


FIG. 9.—Example 5: Graphical integration.

The coordinates of points on the integral curve, which are corresponding values of W_p and W_s, are shown in Table 3.

It will be seen that when $\Delta D = 0.009$, $W_p = 157.4$, which is a satisfactory check with the theoretical yield of 156. The plot of W_p *vs.* W_s is shown in Fig. 10.

The next step, the determination of W_p *vs.* D_p (the size distribution of the product), can now be carried out. It is convenient to assign values of D_p equal to the actual sizes of openings of the screens, obtain the corresponding values of D_s by subtracting 0.009 from these D_p values, read the corresponding values of W_s from curve *A* of Fig. 8, and then read the values of W_p from Fig. 10. The cumulative and differential analyses are then readily calculated. The procedure is shown in Table 5.

Table 3. Relation between W_p and W_s

W_s	W_p	W_s	W_p
5	6.4	70	101.5
10	13.0	80	118.0
20	26.6	85	126.7
30	40.8	90	135.8
40	55.4	95	145.5
50	70.3	97.5	151.0
60	85.7	100	157.4

Limitations of ΔL Law. It must be understood that the above analysis fails completely in any case where crystals are given preferential treatment based on size. If larger crystals have a higher velocity relative to the solution than do smaller ones, the larger crystals will grow more and the smaller crystals less, than is called for by the ΔL law. If there is a classifying action in the crystallizer by which small crystals are retained longer than large crystals, the ΔL law does not apply. It is applicable only if all crystals, regardless of size, are treated exactly alike, both as to conditions of growth and time of growth. It also applies only to the growth of existing crystals and is not concerned with nucleation.

Summary of Work on Crystal Growth. In spite of the considerable disagreement among the published researches on crystal growth and crystal geometry, certain conclusions can be drawn that have much evidence in their favor:

1. A crystal that grows under constant external conditions remains very nearly geometrically similar to its original shape but relatively slight changes in conditions may alter the shape.
2. The addition of small amounts of foreign substances may profoundly affect the shape of a growing crystal and its rate of growth.
3. Different faces of the same crystal usually have different translation velocities.
4. Although the exact mechanism and the order of reaction of the growing process are not definitely known, the rate of growth of any particular face in grams per square centimeter is a function of the supersaturation of the bulk of the solution in contact with that face.

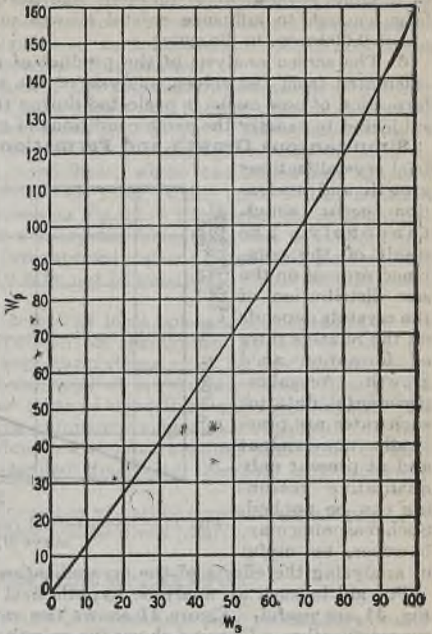


Fig. 10.—Example 5: Weight of product (W_p) vs. weight of seeds (W_s).

Table 4. Screen Analysis of Product

Meshes per inch	D_p	$D_s = D_p - 0.009$	W_s	W_p	Screen analysis of product	
					Cumulative	Differential
On 12	0.1397	0.1307	0.0	0.0	0.0	0.0
14	.1168	.1078	0.6	1.0	0.6	0.6
16	.0991	.0901	10.0	13.0	8.3	7.7
20	.0833	.0743	23.7	32.4	20.8	12.5
24	.0701	.0611	49.9	70.3	44.6	23.8
28	.0589	.0499	73.2	107.3	68.2	23.6
32	.0495	.0405	88.5	134.0	85.1	16.9
35	.0417	.0327	95.1	145.7	92.5	7.4
42	.0351	.0261	96.2	152.5	96.8	4.3
48	.0295	.0205	99.5	156.2	99.2	2.4
60	.0246	.0156	100.0	157.4	100.0	0.8

The cumulative analysis of the product is shown as curve *B*, Fig. 8.

5. It is very doubtful that the differences in solubility of the various faces of a single crystal, or of different size particles of the same material, are large enough to influence crystal growth unless the crystals are less than about 0.002 mm. in diameter.

6. The screen analysis of the product of a crystallization process can be calculated from the screen analysis of the seeds and the total yield if the formation of new nuclei is neglected during the process and if all crystals are subjected to exactly the same conditions of growth.

Simultaneous Growth and Formation of Crystals. In most industrial crystallizations growth and nucleation occur simultaneously. The result of the combined process on the size distribution of the crystals depends on the relative rates of formation and growth. Actual experimental data for such rates are practically non-existent and at present only qualitative reasoning can be applied. Such reasoning may, however, be useful in analyzing the effects of the crystallization factors in industrial processes.

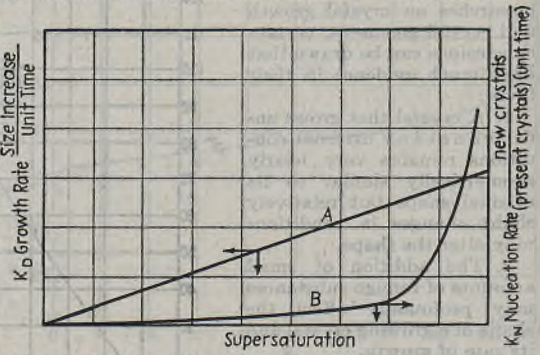


Fig. 11.—Hypothetical rate curves. Curve *A*, growth rate; curve *B*, nucleation rate.

For aid in such an analysis, hypothetical curves such as those shown in Fig. 11 are useful. Figure 11 shows two rate curves, both plotted against supersaturation. Curve *A* shows the growth rate k_D expressed as a coefficient giving the rate of increase of linear dimension per unit time. Curve *B* shows the nucleation rate expressed as a coefficient k_N defined as the number of new crystals produced per unit time per existing crystal. On the assumption that growth is approximately a first-order reaction, curve *A* is straight; on the other hand, the work of Miers and Ting and others indicates that the nucleation rate curve *B* will start low and remain low in the metastable region but will rise sharply at supersaturations corresponding roughly to the point where the supersaturation curves found by Ting are located. As mentioned above, however, the nucleation rate is not necessarily zero at low supersaturations.

Figure 11 shows that at low supersaturations growth will tend to predominate over nucleation, although both rates will be low. At high supersaturations, especially those in which the nucleation rate has passed the break point in the curve, nucleation will predominate strongly over growth.

The positions of curves such as those shown in Fig. 11 will undoubtedly vary considerably with changes in such operating variables as stirring rate, temperature, concentration, and size and number of crystals present in the solution. For example, the position of curve *A* can be expected to depend primarily on viscosity and stirring conditions and, perhaps to some extent, on concentration. The position of curve *B* will undoubtedly be influenced by the number of nuclei present in a unit volume of solution, probably by the area of the crystals present, perhaps by the relative motion of the crystals of the various sizes through the solution, and possibly by the rate of stirring. If nucleation at vapor-liquid interfaces or in other localities in a crystallizer occurs, curve *B* will also be affected by such nucleation. Both curves can be strongly influenced by the presence of impurities.

In a batch crystallization wherein the supersaturation varies with time, the final result of the crystallization process would depend upon an integrated effect along curves such as those shown in Fig. 11. Present knowledge of crystallization is not sufficient to provide a quantitative basis for such an integration. In a continuous crystallizer, where conditions are constant at any given point in the apparatus, the rates at this point will be constant and correspond to a definite abscissa on Fig. 11. The higher the supersaturation at a given point, the more will crystal formation tend to predominate over crystal growth, whereas at low supersaturations growth will predominate over formation, but, in general, it may not be possible to eliminate nucleation entirely.

In general, except for such factors as local concentration gradients, the rate curves of Fig. 11 are independent of the type of apparatus used. If large crystals are desired, low supersaturations must be used or too many nuclei will form, regardless of the type of crystallizer used. Low supersaturations necessarily mean low rates of deposition and large crystals can be made only at the expense of low volumetric crystallizer capacity. Certain types of equipment tend to produce low supersaturations and retain crystals under growth conditions for extended times and therefore give relatively large crystals at low capacity.

The interpretation of the performance of actual crystallization equipment in terms of hypothetical rate curves will be given later.

CRYSTALLIZATION APPARATUS

To force crystallization, it is necessary to maintain the crystallizing solution in a supersaturated condition. The means chosen for producing and maintaining supersaturation usually depend on the solubility-temperature relation of the substance to be crystallized [Svanoe, *Ind. Eng. Chem.*, **32**, 637 (1940)]. Some solutes, such as sodium chloride, have a very small positive temperature coefficient of solubility. Others, such as anhydrous sodium sulfate and sodium carbonate monohydrate, have negative coefficients and become less soluble as the temperature is increased. In such cases supersaturation must be developed by evaporation. Other solutes, *e.g.*, Glauber's salt, Epsom salt, copperas, and hypo, have large positive temperature coefficients of solubility, and cooling without evaporation can produce the required supersaturation. In intermediate cases some evaporation in addition to cooling may be advisable to build up supersaturation.

The application of cooling as compared with evaporation is also important with respect to the yield of the process. If the solubility curve is such that the concentration of the final mother liquor is low in comparison with that of the initial solution, cooling without substantial evaporation will give a satisfactory yield per pass, and the amount of mother liquor recycled will be small. If, however, there is but little solubility change with temperature, the yield per pass through the crystallizer will be small if there is no evaporation and a large quantity of mother liquor must be recycled per unit weight of product. In such a case evaporation should be used, either as such or in combination with cooling. If the water balance of the entire process requires an evaporation step somewhere in the cycle, any evaporation accomplished in the crystallizer also reduces the evaporator load.

A classification of crystallization equipment based on the means used to develop supersaturation and to control yield per pass is

1. Supersaturation produced by cooling without substantial evaporation:

a. Atmospheric cooling by natural convection. Examples: tank crystallizers, Wulff-Bock crystallizer.

b. Cooling by liquid cooling medium, absorbing heat through metal surface. Examples: agitated batch crystallizer, Howard crystallizer, double-pipe crystallizer, Swenson-Walker crystallizer, Krystal cooling crystallizer.

2. Supersaturation produced by evaporation without substantial cooling, where the heat for the evaporation is transferred to the solution through metal surfaces. Examples: crystallizing evaporators, Krystal evaporator crystallizer.

3. Supersaturation produced by adiabatic evaporation and cooling. Example: vacuum crystallizers.

Crystallizers may also be classified in the following manners: batch *vs.* continuous, agitated or non-agitated, classifying or non-classifying. Classifying crystallizers function in such a manner that crystals are retained in the crystallizer until they have reached a minimum size before discharge.

Tank Crystallization. Common practice in producing crystals has been to prepare hot, nearly saturated solutions and to cool them, by natural convection, in open rectangular tanks. Little or no attempt is made to seed these tanks, to provide agitation, or to control the crystallization during the process. Sometimes rods or strings are hung in the tanks to give the crystals additional surface on which to grow and to keep at least a part of the product out of the sediment that might collect in the bottom.

When the tanks have cooled sufficiently, which is usually a matter of several days, any remaining mother liquor is drained off and the crystals are removed by hand. This involves much labor and often results in the inclusion, with the crystals, of any impurities that have settled to the bottom of the tank. The floor space and labor required and the amount of material tied up in the process are large. For moderate capacities, ordinary bathtubs of cheap quality are often very satisfactory for batch crystallizers.

The following data from a typical batch crystallization are available (G. M. Darby, Dorr Co., New York, private communication). The material crystallized was copperas ($FeSO_4.7H_2O$). The concentrated liquor was placed in rectangular crystallization tanks about 21 ft. 6 in. long, 10 ft. 8 in. wide, and 1 ft. deep. The solution was allowed to cool for from 48 to 96 hr., and the crystals removed. About 6 hr. were required to drain a tank, remove the crystals, and refill. The following data pertain to this process:

Table 5. Tank Crystallization of Copperas

Total time of cycle, hr.*	Solution received, gal.	Sp. gr. of solution received	$FeSO_4.7H_2O$ in solution, lb.	Crop, lb.	Mother liquor, gal.	Sp. gr. of mother liquor
48	1,630	1.400 at 62°C.	11,171	7,275	835	1.240 at 15°C.
60	1,630	1.395 at 60°C.	11,085	7,365	945	1.247 at 19°C.
72	1,630	1.935 at 53°C.	10,994	8,550	815	1.210 at 10°C.
96	1,660	1.400 at 51°C.	11,254	8,787	950	1.201 at 9°C.

* Time of cooling = total time—6 hr. (approx.).

The mechanism of crystallization in a tank crystallizer can be visualized as follows: Ordinarily such crystallizers are open to the atmosphere. They lose a considerable proportion of their heat by evaporation and by convection at the surface. In all probability such crystallizers are seeded to some extent by the atmosphere which will contain seeds and dust from previous crystallizations of the same material. The evaporation at the surface develops local supersaturation at the surface and nucleation tends to occur at that point. Nucleation may also occur near the wall of the crystallizer because there will be some heat transfer through the walls to the surroundings. The loss of heat at the surface and boundaries of the crystallizer causes slow convection currents through the solution, and seed crystals are carried with such currents and grow slowly. Since there is no agitation, as soon as crystals are of any appreciable size, they settle in the bottom of the crystallizer and most of the growth occurs in the bottom of the vessel. The convection currents coming down from the top and sides of the vessel bring supersaturated solution in contact with the crystals at the bottom and provide a driving force for the growth. Since the contact between crystals and solution is poor and the cooling rate is low, the nucleation rate is relatively small, and growth predominates over formation. The crystals are usually large and tend to interlock. This interlocking may result in the occlusion of mother liquor and the introduction of impurities. The size of the individual crystals is variable because of the lack of control of the convection currents and the lack of agitation. The capacity of such a crystallizer is low because of the low rate of heat transfer obtainable by atmospheric cooling. The evaporation from the surface is sensitive to the relative humidity of the air, and at times in summer weather, when the relative humidity and air temperature are both high, production from such equipment may be very low.

In the operation of naturally cooled tank crystallizers, there is no way to control either nucleation nor growth except by using suitable lagging, or by varying the ratio of tank surface to tank volume. The size of the unit can be so chosen that the rate of heat loss corresponds roughly to the cooling time necessary to give the desired crystal size.

The Wulff-Bock Crystallizer. The Wulff-Bock [Griffiths, *J. Soc. Chem. Ind.*, **44**, 7*T* (1925)] type crystallizer has been widely used in Germany and in England but has not been used extensively in the United States. It consists of a shallow trough set at a slight inclination and mounted on rollers so that it can be rocked from side to side. At frequent intervals along its length are partitions extending part way across, so that the liquid, instead of flowing directly from one end to the other, flows in a zigzag path. Cooling is by natural convection, and the crystallizer is continuously operated.

The slow rate of cooling inherent in the Wulff-Bock crystallizer results in a relatively low capacity, but this crystallizer gives uniform crystals of unusually large size.

Agitated Batch Crystallizers. Figure 12 shows an agitated batch crystallizer. Water is circulated through the cooling coils and the solution is agitated by the propellers on the central shaft. This agitation performs two functions: (1) It increases the rate of heat transfer and keeps the temperature of the solution more nearly uniform, and (2) by keeping the fine crystals in suspension it gives them an opportunity to grow uniformly instead of forming large crystals or aggregates. Further, the agitation, combined with the more rapid cooling, results in the formation of a large number of nuclei as compared with the tank method, and therefore the product of this operation is not only more uniform but also very much finer than that from the older tanks. The difficulties with this apparatus are: first, that it is a batch or discontinuous method; and, second, that the solubility is least in the stagnant film on the surface of the cooling coils. Consequently crystal growth is most rapid at this point, and the coils rapidly build up with a mass of crystals which decreases the rate of heat transfer.

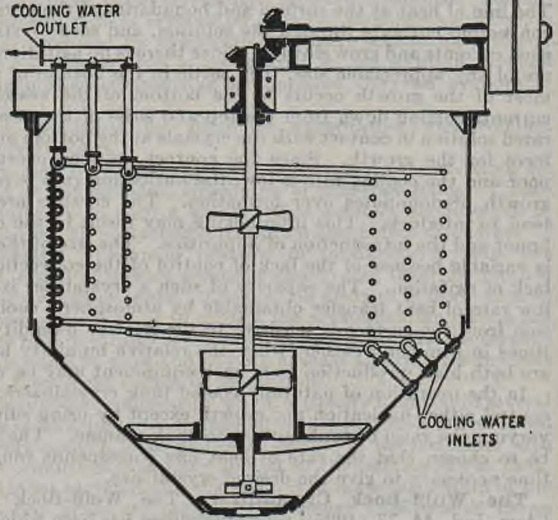


Fig. 12.—Agitated batch crystallizer.

A certain amount of control can be exercised on the crystallization process occurring in a batch, artificially cooled crystallizer by varying the rate of cooling. As long as the solution is undersaturated, it may be cooled as rapidly as the temperature and rate of the cooling water and the area of the cooling surface permit. When the solution becomes supersaturated, the rate of temperature decrease should be retarded in order that the labile region is not entered. It is advisable to seed the solution at this point to prevent the uncontrolled initial nucleation characteristic of unseeded solutions. Once there is available considerable crystal surface for growth, the rate of heat removal can be increased, provided the coils are not so badly salted that effective heat transmission is prevented. The rate of temperature decrease may be very small during the period of maximum crystallization rate, even if the rate of heat transfer is good because of the necessity of withdrawing from the solution the latent heat of crystallization.

The Howard Crystallizer. This crystallizer (Fig. 13) consists essentially of a vertical conical device through which solution flows in an upward direction. The upper end of the crystallizer is the wide part of the cone. A concentric outer conical chamber serves as a cooling water channel. Crystals that are suspended in the upward flowing stream of solution must grow to such a size that they will settle through the fastest part of the stream of solution at the apex of the cone (the bottom of the crystallizer) before they can escape. By regulating the velocity of flow at the bottom of the crystallizer, the size of the product is controlled. On the other hand, the cross section of the top of the crystallizer is large, the velocity of the solution is low, and the smaller crystals are not carried over the top. The apparatus functions both as a crystallizer and as a hydraulic classification device. The concentration of the solution is maintained by the inflow of strong solution from a storage tank, and the product is withdrawn continuously in a stream of mother liquor.

Since the Howard crystallizer is a continuous one, conditions in it can reach a steady state. As the solution flows up through the crystallizing cone its supersaturation increases due to the cooling brought about by the inner cooling cone *C*. Nucleation should start at a fairly definite point in the crystallizer. Nuclei will tend to drop through the incoming solution due to the action of gravity and are thereby continually contacted with fresh supersaturated solution. Growth therefore occurs, and, when the crystal is large enough, it will settle into vessel *G*. Nucleation due to the inoculating effect of existing crystals can accompany the growth in the same zone in which the growth is taking place.

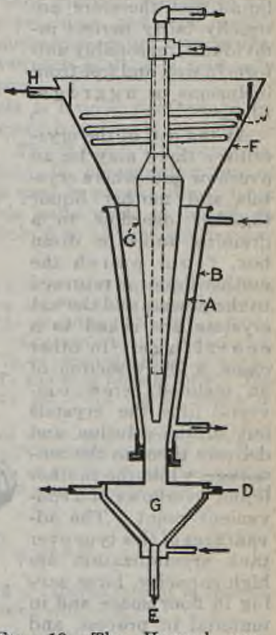


FIG. 13.—The Howard crystallizer.

The Double-pipe Crystallizer. [Seavoy and Caldwell, *Ind. Eng. Chem.*, **32**, 628 (1940)]. This crystallizer consists essentially of a double-pipe heat exchanger fitted with internal helical ribbons. The cooling liquid passes through the annular space between the two pipes, and the crystallizing solution is pumped through the inner tube countercurrent to the cooling liquid. The helical ribbons act as scrapers to keep crystals from building up on the cooling surface. The scrapers make contact with the inner wall of the inside pipe. This crystallizer is ordinarily used in a continuous batch manner. It is placed in series with a large tank containing the solution to be cooled, and the solution is pumped through the double-pipe unit at a rate sufficient to ensure an adequate heat-transfer rate.

The Swenson-Walker Crystallizer. In the United States the most successful continuous crystallizer using a liquid cooling medium is the Swenson-Walker crystallizer. This crystallizer is shown in Fig. 14 (Caldwell and Seavoy, *loc. cit.*). It consists of an open trough 24 in. wide, with a semicylindrical bottom, a water jacket welded to the outside of the trough, and a slow-speed, long-pitch, spiral agitator set close to the bottom of the trough, but not so close as to make contact with the trough. This apparatus is ordinarily built in units 10 ft. long, and a number of units may be joined together to give increased capacity. Forty feet is the maximum length usually driven from one shaft and, if lengths greater than this are desired, it is usual to arrange several such crystallizers, one above the other, and allow the solution to cascade from one bank to the other.

The hot concentrated solution to be crystallized is fed at one end of the trough, and cooling water usually flows through the jackets, countercurrent to the solution. In order to control crystal size, it is sometimes desirable to introduce an extra amount of water into certain sections. When conditions are properly adjusted, nuclei begin to form a short distance from the point where feed is introduced; these nuclei grow regularly as the solution passes down the length of the crystallizer. The function of the spiral stirrer is not especially that of either agitation or conveying the crystals. Its purposes are (1) to prevent an accumulation of crystals on the cooling surface and (2) to lift the crystals which have already been formed and shower them down through the solution. In this manner the crystals grow while they are freely suspended in the liquid and therefore are usually fairly perfect individuals, reasonably uniform in size, and free from inclusions or aggregations.

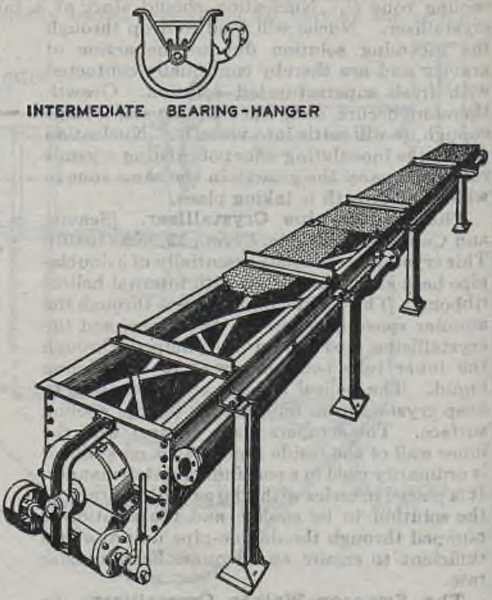


FIG. 14.—The Swenson-Walker crystallizer. [*Seavoy and Caldwell, Ind. Eng. Chem.*, **32**, 632 (1940).]

At the end of the crystallizer there may be an overflow gate where crystals and mother liquor together overflow to a draining table or drain box, from which the mother liquor is returned to the process and the wet crystals are raked to a centrifuge. In other cases, a short section of an inclined screw conveyor lifts the crystals out of the solution and delivers them to the centrifuge, while the mother liquor overflows at a convenient point. The advantages of this type over tank crystallization are high capacity, large saving in floor space and in material in process, and especially a saving in labor. To reach low final temperatures, the cooling can be obtained by the use of refrigerated brine instead of cooling water.

The mechanism of crystallization in the Swenson-Walker crystallizer is as follows: Preferably the incoming solution should be slightly superheated above its supersaturated temperature. It should be cooled uniformly, especially through the range of temperature where nucleation first occurs. It is possible to seed the solution artificially, but ordinarily seeding is restricted to the fortuitous seeding from the atmosphere. If the solution is not seeded and is also protected from atmospheric seeding, it will tend to supercool well into the supersaturated region and on reaching the Miers curve will tend to form in an uncontrolled fashion too many nuclei. Once the nucleation begins, either through fortuitous seeding or by entering the Miers labile region, crystals and solution flow together through the remaining crystallizer length.

It is important that the solution is not shocked at any point in the crystallizer by too rapid local cooling into producing a large crop of nuclei at that point. With uniform cooling, a steady growth of the crystals occurs. The presence of the crystals, however, and the action of the agitator, together with any superficial evaporation from the surface, cause a certain amount of continuous nucleation along with the growth. With slow cooling, uniform cooling, and a reduction of evaporation at the surface by covering the crystallizer, a fairly uniform product without too many fines is produced. There is no classifying action in the Swenson-Walker, and therefore complete uniformity is not possible in the product. Under ideal conditions the Swenson-Walker should be operated in such a fashion that most of the nuclei are formed in a comparatively short length of the crystallizer, and such cooling conditions are maintained that a minimum of nucleation occurs beyond this point. The supersaturation produced in the Swenson-Walker is primarily due to cooling and only incidentally to evaporation.

The capacity of a Swenson-Walker crystallizer is largely determined by heat-transfer considerations. An over-all coefficient based on a logarithmic average temperature difference of 10 to 25 B.t.u./(°F)(hr.)(sq.ft). is used and an effective heat-transfer area of 3 sq. ft. per running foot of crystallizer assumed. The number of units to be used in parallel depends on the total capacity desired. For most inorganic salts such as trisodium phosphate, Glauber's salt, etc., a production of from 5 to 15 tons per day per unit can be obtained.

Example 6. A Swenson-Walker crystallizer is to cool a 23 per cent solution of Na_3PO_4 from a temperature of 104° to 77°F. During the cooling, $Na_3PO_4.12H_2O$ is crystallized. It is desired to produce 500 lb. of product per hour. The solubility of Na_3PO_4 at 77°F. is 15.5 parts anhydrous salt per 100 parts total water. The specific heat of the solution can be taken as 0.77, and the heat of crystallization of 1 lb. of product is 63 B.t.u. per lb. Cooling water is to enter the crystallizer jacket at 60°F. and is to leave at 68°F. The over-all heat-transfer coefficient is 25 B.t.u./(sq. ft.) (hr.)(°F.). What length of crystallizer should be used?

Solution. The weight of solution per hour that will give 500 lb. of crystals is calculated with the aid of Eq. (1). The molecular weight of $Na_3PO_4.12H_2O$ is 380.2, and that of Na_3PO_4 is 164.0; $R = 380.2/164.0 = 2.32$. If a basis of 100 lb. original solution is chosen, $w_0 = 23.0$; $S = 15.5$; and $H = 100 - 23.0 = 77.0$ lb. The product obtained from 100 lb. solution is, by Eq. (1),

$$C = 2.32\frac{(100)(23.0) - (15.5)(77.0)}{100 - 15.5(2.32 - 1.0)} = 32.2 \text{ lb.}$$

In order that 500 lb. per hr. of crop be obtained, a feed of (100/32.2)(500) = 1550 lb. is necessary. The heat to be removed from the crystallizing solution is:

$$\begin{aligned} &\text{To cool solution } (1550)(0.77)(104 - 77) &&= 32{,}200 \text{ B.t.u. per hr.} \\ &\text{To crystallize: } (500)(63) &&= \underline{31{,}500 \text{ B.t.u. per hr.}} \\ &\text{Total} = q &&= 63{,}700 \text{ B.t.u. per hr.} \end{aligned}$$

The logarithmic mean temperature drop is

$$(\Delta t)_m = \frac{(104 - 68) - (77 - 60)}{2.303 \log \dfrac{104 - 68}{77 - 60}} = 25°\text{F.}$$

The length of crystallizer is

$$\frac{q}{3U(\Delta t)_m} = \frac{63{,}700}{(3)(25)(25)} = 34 \text{ ft.}$$

Four 10-ft. sections should be used.

Crystallizing Evaporators. The development of supersaturation by means of evaporation without substantial cooling is often carried out in equipment that has the physical characteristics of an evaporator and, in fact, is designed essentially as an evaporator largely because the essential engineering problem is one of heat transfer. Usually the equipment employed is so nearly like that used in ordinary evaporation that it is considered to be an evaporator, although the crystallization may be the more difficult of the two parts of the problem. In the evaporation of a salting liquor (*e.g.*, the precipitation of NaCl from brine in the common-salt industry) the crystallization is usually incidental to the evaporation, and no particular control of size is exercised. On the other hand, the crystallizing of sugar is carried out in a vacuum evaporator, but the control is based entirely on building a correct crystal. In this case the operator brings the syrup to a definite density, shocks out the desired number of nuclei, and grows them to the correct size without forming new crystallization centers. The control is exercised entirely by varying the vacuum and steam supply.

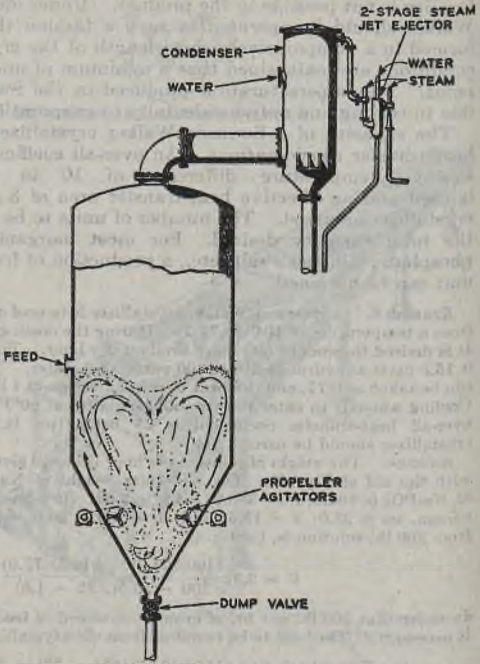


FIG. 15*a*.—Batch crystallizer without booster. [*Seavoy and Caldwell, Ind. Eng. Chem.*, **32**, 628 (1940).]

A special case of evaporative crystallization is the salt grainer (Badger and Baker, "Inorganic Chemical Technology," pp. 15–22). In the salt grainer the solution is kept hot and supersaturation is developed by evaporation rather than by cooling. Nucleation occurs at the surface of the brine, the nuclei tend to be retained at the surface by surface-tension effects and to form hopper-shaped crystal agglomerates, which, when large enough, break away, drop to the bottom of the grainer, and are raked out by slow-moving rakes.

Vacuum Crystallizer. Assume that a warm, saturated solution is fed to a lagged closed vessel that is maintained under a vacuum and that the solution is fed in such a way that it reaches the surface of the liquid in the crystallizer. The solution will have a definite boiling temperature under the vacuum existing in the vessel. If this temperature is *less* than that of the feed solution, the solution will spontaneously and adiabatically cool to the boiling temperature corresponding to the vacuum existing in the vessel and reach equilibrium with respect to the vapor in the crystallizer. If the solubility of the solute decreases as the temperature decreases, the cooling will result in crystallization, not only because of the cooling of the solution, but also because of the evaporation of some of the solvent, as the heat evolved in the cooling and crystallizing of the solution must appear as latent heat of evaporation. Such a crystallizer is a vacuum crystallizer.

In a vacuum crystallizer it is necessary to ensure that the incoming solution reaches the surface and can thereby flash to equilibrium with the vapor in the crystallizer. Artificial circulation is necessary to accomplish this; otherwise the feed will tend to short-circuit to the discharge, especially if the discharge is at the bottom of the vessel. Under the low absolute pressures commonly used in this equipment, a foot of hydrostatic head causes a decided increase in boiling point of the solution; if short-circuiting occurs, little or no supersaturation will be produced in the short-circuited solution and it will not deposit solid material.

The vacuum crystallizer has become very successful during the past few years and continues to replace many of the older types of the mechanical crystallizer. Four forms of the vacuum crystallizer are shown in Fig. 15 [Seavoy and Caldwell, *Ind. Eng. Chem.*, **32**, 627–636 (1940)]. Figure 15*a* represents a simple type of batch vacuum crystallizer. Propeller agitators are used to develop a swirling action in the crystallizer. In case the temperature in the crystallizer is high enough so that the vapor from it is condensable by the cooling water available, the vapors from the crystallizer pass directly to a condenser as shown in Fig. 15*a*. In most cases, however, the cooling water is too warm to condense the vapor leaving a vacuum crystallizer at the desired temperature of operation in the crystallizer. In this case a steam-jet booster is used to compress the vapors to a point where they can be condensed by the cooling water available. The air and non-condensable gases from the condenser are commonly ejected to the atmosphere by further steam-jet equipment. A second type of vacuum crystallizer, operated by such a booster, is shown in Fig. 15*b*. This type is operated continuously. The feed tube is so positioned that the feed solution is forced to the surface and flashes to equilibrium with the vapor. Propeller agitators aid in keeping the crystals in suspension and in preventing short circuiting.

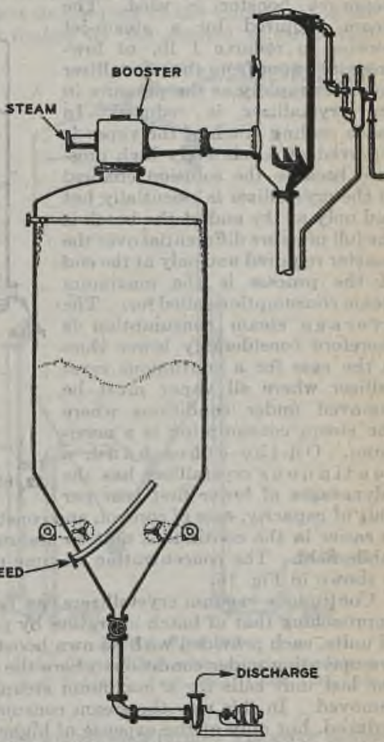


FIG. 15*b*.—Continuous crystallizer with booster. [*Seavoy and Caldwell, Ind. Eng. Chem.*, **32**, 628 (1940).]

The forms of crystallizer shown in Figs. 15*c* and *d* can be operated either as batch or as continuous units. The combination of propeller and draft tube shown in Fig. 15*c* is effective in preventing short circuiting of the feed.

The form shown in Fig. 15*d* has an external circulating pump which takes suction from the side of the crystallizer and discharges tangentially into the cone. The agitators can be omitted from this unit. The feed is introduced into the circulation stream, and the circulation stream must be large enough that the mixed stream is not so supersaturated that it is in the labile condition.

Batch operation has the advantage of low-steam consumption if a steam-jet booster is used. The steam required by a steam-jet booster to remove 1 lb. of low-pressure vapor from the crystallizer increases rapidly as the pressure in the crystallizer is reduced. In batch cooling much of the vapor is removed at a relatively high pressure because the solution charged to the crystallizer is essentially hot and only at the end of the batch is the full pressure differential over the booster required and only at the end of the process is the maximum steam consumption called for. The average steam consumption is therefore considerably lower than is the case for a continuous crystallizer where all vapor must be removed under conditions where the steam consumption is a maximum. On the other hand, a continuous crystallizer has the advantages of lower first cost per unit of capacity, ease of control, and constant mass of crystals in the unit. It is easier in the continuous unit to maintain supersaturations outside of the labile field. The concentration *vs.* time relationship for a typical batch case is shown in Fig. 16.

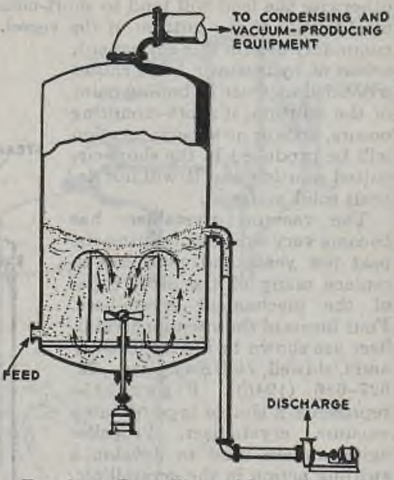


Fig. 15*c*.—Crystallizer with vertical agitator. [*Seavoy and Caldwell, Ind. Eng. Chem.*, **32**, 629 (1940).]

Continuous vacuum crystallizers can be operated to give a steam economy approaching that of batch operation by passing the solution through a series of units, each provided with its own booster. In the first stages the boosters are operating under conditions where the steam consumption is low and only the last unit calls for a maximum steam consumption per pound of vapor removed. In this way the steam consumption of a continuous unit may be reduced, but only at the expense of higher investment costs.

Analysis of Vacuum Crystallizer Operation. The action of a continuous vacuum crystallizer in forming and growing crystals may be visualized as follows:

Assume that the agitation and circulation in such a crystallizer are adequate to ensure homogenous conditions throughout the crystallizing liquid. Assume also that the crystallizer is fed constantly with solution of a given concentration and temperature and that the pressure in the crystallizer is constant.

Under steady-state conditions there must be a definite and constant number of crystals in the crystallizer at all times. These crystals will have a definite size distribution, and a steady state is reached if the rate of formation and growth does not change with time. Assume also that the operation of the crystallizer is such that a crystal will not be able to leave the crystallizer until it has reached a definite size. Under steady-state conditions the entering solution immediately flashes to equilibrium temperature. The flash results in a certain amount of evaporation and considerable cooling, which develops in the solution a definite supersolubility. The solution is mixed quickly with the bulk of the solution in the crystallizer and a definite supersaturation can be assumed to exist in the bulk of the liquid. It is the relatively high supersaturation contributed by the flashed solution that maintains the supersaturation at a definite average value in competition with the efforts of the crystals to destroy the supersaturation. If the flashed solution is blended rapidly enough with the liquor in the crystallizer, it will not form new nuclei previous to its incorporation with the bulk of the liquid. Since at all times there is a definite number of crystals of a definite size range, and also a definite supersaturation in the solution, there will be a definite rate of growth and a definite rate of formation corresponding to a given abscissa on a curve such as that of Fig. 11. Since there will be formed a constant number of crystals per unit time, the rate of formation of crystals must be equal to the rate at which crystals are withdrawn from the crystallizer. The rate of nucleation must adjust itself in such a manner that the number of crystals formed per unit time must equal the number of crystals withdrawn per unit time. Also, the rate of growth must adjust itself in such a manner that the crystals formed will grow to the desired size in the time available for their growth. If the rate curves are of the form shown in Fig. 11, there is only one supersaturation that will give both a correct rate of formation and a correct rate of growth. For larger crystals, growth rate rather than nucleation rate must be emphasized, and a low supersaturation will exist in the crystallizer. For smaller crystals, nucleation will increase relative to growth, and larger supersaturation will be necessary. Rate of withdrawal (assuming that adequate flashing and vapor liberation can be maintained) controls all these factors automatically. If the rate of withdrawal from the crystallizer is slow, the crystals withdrawn will be large. The capacity of the crystallizer will decrease, of course, as the size of the crystals is increased.

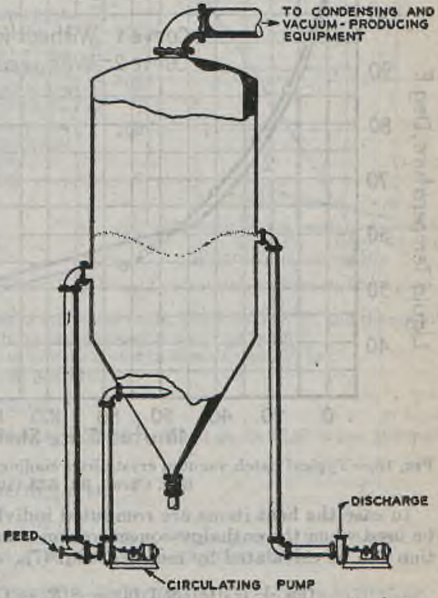


FIG. 15*d*.—Crystallizer with pump circulation. [*Seavoy and Caldwell*, *Ind. Eng. Chem.*, **32**, 629 (1940).]

Yield from a Vacuum Crystallizer. The calculation of the yield obtainable in a vacuum crystallizer depends upon the method used in the heat-balance calculations. A vacuum crystallizer operates essentially adiabatically. The heat liberated by the solution on cooling to the equilibrium temperature and the heat of crystallization are available for vaporizing water from the solution, and these thermal effects must balance. If the enthalpy-concentration chart is used, the total enthalpy of the vapors and magma leaving the crystallizer must equal the total enthalpy of the feed solution entering the unit.

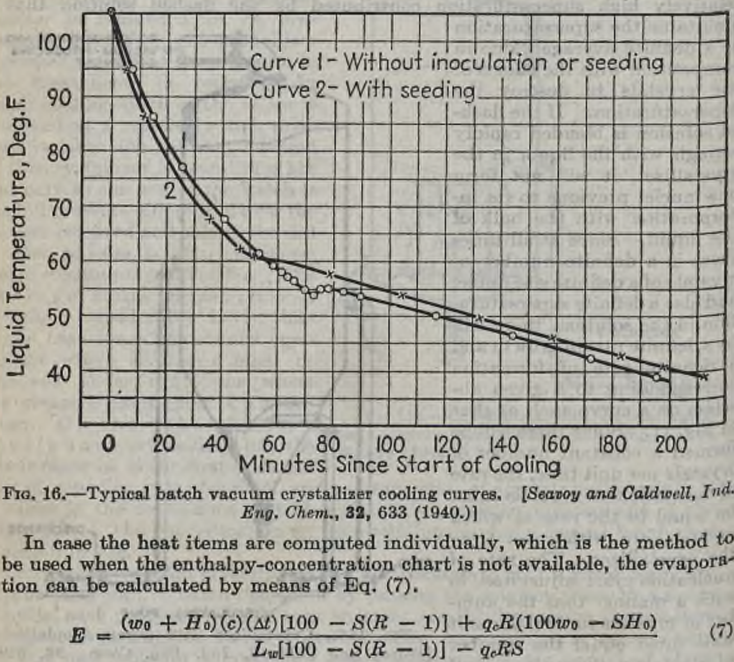


FIG. 16.—Typical batch vacuum crystallizer cooling curves. [*Seavoy and Caldwell, Ind. Eng. Chem.*, **32**, 633 (1940.)]

In case the heat items are computed individually, which is the method to be used when the enthalpy-concentration chart is not available, the evaporation can be calculated by means of Eq. (7).

$$E = \frac{(w_0 + H_0)(c)(\Delta t)[100 - S(R - 1)] + q_cR(100w_0 - SH_0)}{L_w[100 - S(R - 1)] - q_cRS} \tag{7}$$

where w_0 is the weight of anhydrous solute; H_0 is the total weight of solvent in the feed solution; c is the specific heat of the feed solution; Δt is the temperature range through which the solution is cooled (temperature of feed to temperature of discharge); q_c is the heat of crystallization per unit weight of crystal; L_w is the latent heat of evaporation from the solution; S is the anhydrous solubility in parts solute per 100 parts total solvent; and R is the ratio of the molecular weight of the crystals to that of the anhydrous salt. When the value of E is known, the yield is calculated by means of Eq. (1).

If the enthalpy-concentration chart is used, the simplest method of vacuum crystallizer calculation is one based on the "basic construction" applicable to the chart. Since the process is adiabatic, a single straight line on the chart must pass through the three points on the chart that represent feed solution, magma, and vapor, respectively. Furthermore, the point representing the magma must lie on the straight-line isotherm lying in the magma field and corresponding to the temperature of the vapor and magma. The point representing the magma is found by locating the intersection of these two lines as shown in Fig. 17. The ratio of crystals to mother liquor in the magma is found from the intersection by applying the lever-arm principle to the two line segments on the straight-line isotherm.

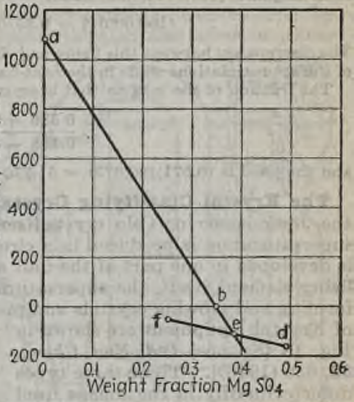


FIG. 17.—Solution to Part *b*, Example 7.

Example 7. A continuous vacuum crystallizer is fed with 100,000 lb. per hr. of a 35 per cent solution of $MgSO_4$ at a temperature of 183°F. An absolute pressure of 0.2 lb. per sq. in. is maintained in the crystallizer by the booster, and the solution has a 10°F. elevation in boiling point. Calculate the yield of $MgSO_4.7H_2O$, and the evaporation for this crystallizer:

a. By means of Eqs. (1) and (7).
b. By means of Fig. 1.

Solution, Part *a*: For this take

c = 0.77 B.t.u. per °F. per lb.
L_w = 1080 B.t.u. per lb.
q_c = 23 B.t.u. per lb. $MgSO_4.7H_2O$

The temperature of boiling water at 0.2 lb. per sq. in. absolute is 53°F., and the equilibrium temperature of the solution in the crystallizer is 53 + 10 = 63°F.

The remaining numerical values to be substituted in Eqs. (1) and (7) are:

$w_0 = (0.35)(100{,}000) = 35{,}000$ lb. per hr.

$$R = \frac{\text{mol. wt. } MgSO_4.7H_2O}{\text{mol. wt. } MgSO_4} = 2.045$$

S = solubility of anhydrous $MgSO_4$ at 63°F. = 33.33 parts $MgSO_4$ per 100 parts total water (Fig. 2)

$\Delta t = 183 - 63 = 120$°F.

$H_0 = 100{,}000 - 35{,}000 = 65{,}000$ lb. per hr.

then

$$(w_0 + H_0)(c)(\Delta t)[100 - S(R-1)] = (100{,}000)(0.77)(120)[100 - 33.33(2.045 - 1)] = 6.022 \times 10^8$$

$$(q_c)(R)(100w_0 - SH_0) = (23)(2.045)[(100)(35{,}000) - (33.33)(65{,}000)] = 0.62 \times 10^8$$

$$L_w[(100 - S(R-1)] - q_cRS = 1080[100 - 33.33(2.045 - 1)] - (23)(2.045)(33.33) = 68{,}793$$

By Eq. (7) the evaporation is:

$$E = \frac{(6.022)(10^8) - 0.627 \times 10^8}{68{,}793} = 7842 \text{ lb. per hr.}$$

By Eq. (1) the yield is:

$$C = (2.045)\frac{(100)(35{,}000) - (33.33)(65{,}000 - 7842)}{100 - (33.33)(2.045 - 1)} = 50{,}050 \text{ lb. per hr.}$$

Solution, Part *b*: Figure 17 shows the graphical solution of this problem using values from the enthalpy-concentration chart of Fig. 1. From steam tables, the enthalpy of 1 lb. of steam at a pressure of 0.2 per sq. in. absolute and a temperature of 63°F. is 1089.5 B.t.u. Point *a* has the coordinates $H = 1089.5$, $c = 0$. The enthalpy of the feed solution, which has a concentration of 0.35 and a temperature of 183°F., is 0. The coordinates of point *b* are $H = 0$, $c = 0.35$, and straight line *ab* can be drawn. The straight line *fd* is the 63°F. isotherm in the area *bcih* of Fig. 1. The coordinates of its terminals are: point f, $H = -49.5$, $c = 0.25$; point d, $H = -157.5$, $c = 0.488$. Lines fd and ab intersect at point e, which has the coordinates $H = -111.2$, $c = 0.386$. The average concentration of the final magma is, therefore, 38.6 per cent $MgSO_4$. The total magma is $(100{,}000)(0.35/0.386) = 90{,}673$ lb. per hr. The evaporation is, therefore,

$$(100{,}000)(1 - 0.35/0.386) = 9327 \text{ lb. per hr.}$$

The discrepancy between this figure and that found in part *a* (7842 lb. per hr.) is a result of the approximations made in the heat-balance items used in part *a*.

The fraction of the magma that is crystalline is

$$\frac{0.386 - 0.25}{0.488 - 0.25} = 0.57.$$

and the yield is $(0.571)(90{,}673) = 51{,}770$ lb. per hr.

The Krystal Classifying Crystallizer. This equipment, also known as the Jeremiassen or Oslo crystallizer, is characterized by the fact that the supersaturation is produced in a circulating stream, and the supersaturation is developed in one part of the unit and released in another. In the crystallizing element itself, the supersaturated solution flows up through a bed of forming and growing crystals and provides a classifying action. Three types of Krystal equipment are shown in Fig. 18 [Svanoe, *Ind. Eng. Chem.*, **32**, 636 (1940)]. These three types differ primarily in the means used for developing supersaturation. In Fig. 18*a* the supersaturation is obtained by heating the circulating stream while under a static head great enough to prevent its vaporization and flashing of the heated solution in vessel *A*. The vapor released by the flash is removed through pipe *U*. The solution, supersaturated with respect to the temperature existing in crystallizing vessel *E*, leaves the flash vessel *A*, passes up through a screen in the bottom of vessel *E*, contacts crystals above the screen and loses its supersaturation while in contact with them. The overflow stream, leaving vessel *A* and passing to heater *H*, should be practically saturated. Feed solution is mixed with this solution at *T*. Crystals are drawn off continuously or periodically through discharge *M*. Pump *F*, driven by the motor forces the circulating stream through its circuit. Heater *H* is steam heated. This type of crystallizer is used in cases where the supersaturation must be developed entirely by evaporation and not by cooling. Two or more of these units can be connected in multiple effect just as in the case of an ordinary evaporator. Figure 18*b* represents a vacuum type of Krystal unit. This unit is a true vacuum crystallizer, in that the supersaturation is developed by adiabatic pressure reduction on the hot, concentrated, feed solution. The feed enters at *T*, is incorporated in a circulating cycle stream, and the combined stream flashed in flash vessel *A*. Supersaturated solution passing through pipe *B* contacts growing crystals in crystallizer *E*, and the flow of liquid in vessel *E* performs a classifying action. Mother liquor is drawn off at *N* and the magma drawn

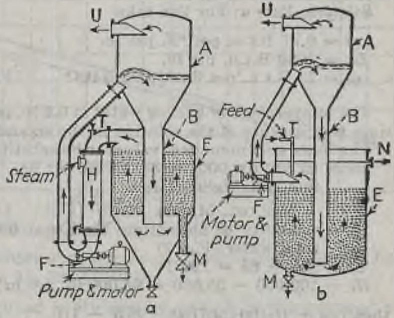


FIG. 18*a* and *b*. Krystal crystallizer. *a*. Evaporator crystallizer. *b*. Vacuum crystallizer. [*Svanoe, Ind. Eng. Chem.*, **32**, 636, 637 (1940).]

off at *M*. Nucleation can occur in vessel *E* by the inoculation of the solution by the existing crystals and by any impact of the crystals on each other and on the wall of the vessel. In continuous operation the rate of nucleation must equal the number of finished crystals withdrawn as product.

The modification shown in Fig. 18*c* develops supersaturation entirely by liquid cooling. The circulating stream passes through the tubes of cooler *H*, and supersaturated solution flows through pipe *B* to the bottom of crystallizer *E*. The feed, which should be warm and concentrated, enters at *T*. It is incorporated immediately in the circulating stream, the combined stream is cooled in *H*, and supersaturation is thereby obtained. The diluting of the incoming feed with a comparatively large circulating stream of mother liquor allows the solution to be cooled in cooler *H* without entering the labile region and thereby allows this cooling to be accomplished without nucleation until the solution comes in contact with the crystals in *E*. Vessel *G* can be used to remove very small nuclei that reach the upper layers of the crystallizer *E*. If small nuclei are continually removed, the average size of the crystal crop is increased. The action of vessel *G* has substantially the same effect as a reduction in the rate of nucleation curve shown in Fig. 11. Mother liquor leaves the crystallizer at *M*.

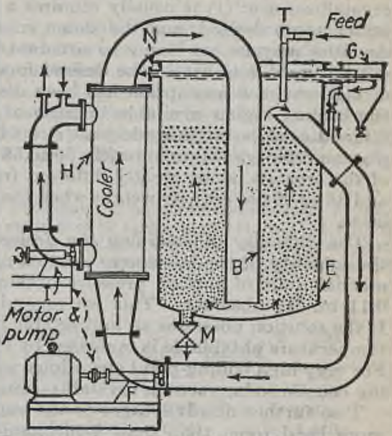


Fig. 18*c*.—Cooling crystallizer. [*Svanoe, Ind. Eng. Chem.*, **32**, 638 (1940).]

In all forms of the Krystal evaporator the theory of the metastable region of a supersaturated solution is applied; nucleation is not obtained in the solution while developing its supersaturation in flash vessel *A* or cooler *H* because the supersaturation developed in these parts is kept low enough so that the labile region of the unseeded solution is not entered. This can be done because of the large amount of recirculating solution. A small supersaturation developed in a large recirculating stream makes available to the crystallizer enough potential precipitating material to force crystallization in vessel *E*. In vessel *E*, however, the presence of the nuclei results, not only in the growth, but also in nucleation due to inoculation, impact, surface, and attrition to supply the nuclei required.

Comparison of Vacuum and Liquid-cooled Crystallizers. In cases where the necessary supersaturation can be developed by cooling, the choice of a crystallizer usually lies between a mechanical unit, cooled by transmitting heat through a metal wall, and a vacuum unit that has no heat-transfer surface in the crystallizer itself. In such a comparison the Swenson-Walker may be taken as representative of the mechanical type, and the vacuum crystallizers shown in Fig. 15, as representative of the vacuum type [Seavoy and Caldwell, *Ind. Eng. Chem.*, **32**, 631 (1940)].

The main difference between the vacuum and mechanical crystallizers is that the heat is removed in the vacuum crystallizer without passing it through a heating surface. This gives the vacuum crystallizer several important advantages. The absence of large heat-transfer surfaces results in a lower first cost for the vacuum crystallizer and also allows the crystallizer to be built of corrosion-resisting materials. The absence of the cooling surface also eliminates the growth of crystals on a metal surface from which they must be removed mechanically during operation. The limitations of the vacuum crystallizer are: (1) it usually requires a steam-jet booster to obtain the low temperature desired, and the steam consumption may be large; (2) magma densities may be too heavy to circulate freely in the crystallizer; (3) it may not be possible to attain the desired final temperature. The first limitation of high steam consumption has been discussed. Magma density may limit the cooling region available because of the difficulty of discharge and the difficulties of developing adequate circulation in the crystallizer. The critical magma density is approximately 50 to 55 per cent by weight when the density of the crystals is not greatly different from the density of the mother liquor and 35 to 40 per cent by weight when the differences in densities are relatively large.

The difficulty of attaining the desired final temperature may be due to the inability of the booster to exhaust at a low enough pressure. The commercial limit of suction pressure obtainable with such equipment is about 0.11 in. Hg absolute. This corresponds to a water boiling point of 20°F. If the solution possesses an appreciable boiling-point elevation, the minimum temperature obtainable is increased by the amount of boiling-point elevation. For very high boiling-point elevations, such as those encountered in crystallizing caustic soda, vacuum crystallization is out of the question.

Two further disadvantages of the vacuum crystallizer are: (1) it requires more head room than does a mechanical-type crystallizer and (2) rubber linings, which are commonly used for corrosive solutions, are unsatisfactory for hot acid solutions.

The mechanical crystallizer has the advantage in not requiring vacuum-producing equipment, of being independent of the vapor pressure of the solution, and in requiring no steam. It can also handle stiffer magmas than can the vacuum crystallizer because free circulation is not required.

The mechanical crystallizer is frequently limited by the materials of construction. The materials used must be such that they allow the transfer of heat and must be structurally strong enough to resist the erosion by the crystals and the action of the agitators. The first cost of mechanical crystallizers for large capacities is relatively high even when they can be constructed of steel. They are subject also to the difficulties arising from the fouling of the cooling surface on the water side if the water is hard. The water side of a mechanical crystallizer cannot usually be cleaned readily.

In most cases where either type can be used, costs favor the vacuum crystallizer.

Cost of Crystallization Equipment. The cost of a crystallizing tank or an agitated batch crystallizer will approach that of an ordinary tank with or without cooling coils and stirrers. The cost of the crystallizing equipment in such cases is a small item in the total cost of the operation, due to the heavy labor charges necessary to handle the product of such crystallizers.

The cost of a Swenson-Walker crystallizer on the basis of steel construction with heavy drives when decks are 40 ft. long and standard drives when decks are 30 ft. long will vary from $70 to $40 per linear foot. The $70 cost is for a length of 20 ft., and the lower figure is for a length of 1000 to 1200 ft.

The cost of a vacuum crystallizer is largely affected by the cost of the vacuum-producing equipment since the balance of the construction is ordinary tank and piping practice. If, however, special metals or rubber-lined equipment is necessary, the fabrication costs of the material used must, of course, be used. A detailed discussion of other factors that influence the costs of crystallizing in a vacuum crystallizer is given by Seavoy and Caldwell (*loc. cit.*).

Crystallization Costs. It is hardly possible to give general cost curves showing the costs of crystallizing the materials that are prepared commercially in the crystalline state. The materials, the conditions of crystallization, and the methods of operation are too varied to give general costs. Each individual case should be investigated as it arises. All comparative costs should be made on the basis of a completed crystalline product, properly screened, dried, and ready for packaging. The following specific examples may be of interest:

Case 1. Crystallization of trisodium phosphate, in naturally cooled vats. Production, 31 tons per 24 hr. The equipment consists of 80 tanks, each 6 ft. by 12 ft. by 2 ft. The time of cooling is 5 days. The first cost of each tank was \$150, and accessory costs were \$2000. The total investment was (80)(150) + 2000 = \$14,000. Operation, 300 days per year.

Annual Costs:

Depreciation at 5% per year = (14,000)(0.05)	\$ 700
Average interest at 5%, allowing for interest earned by depreciation reserve = $(\frac{21}{20})$(14,000)(0.05/2)	368
Repairs, maintenance, and lubrication	200
Floor rent, 10,080 sq. ft. at \$0.20 per sq. ft. = (10,800)(0.20)	2,016
Power, 20 kw. at \$0.01 per kw.-hr. = (20)(24)(300)(0.01)	1,440
Interest on material in process, at 5% interest and \$0.025 per lb. = (5)(2000)(31)(0.05)(0.025)	387
Labor, 188 man-hr. per day at \$0.50 per hr. = (188)(300)(0.50)	28,200
Total annual costs	\$33,311

$$\text{Cost per ton of product (not including centrifuging, drying, screening, etc.)} = \frac{33,311}{(300)(31)} = \$3.58$$

Case 2. Trisodium phosphate, in Swenson-Walker crystallizer. Production, 31 tons per 24 hr. The equipment consists of two units, each 40 ft. long. The investment was \$19,000. Operation is 300 days per year.

Annual Costs:

Depreciation at 12½% per year = (0.125)(19,000)	\$ 2,375
Average interest at 5%, allowing for interest earned by depreciation reserve = $(\frac{9}{8})$(19,000)(0.05/2)	535
Repairs, maintenance, and lubrication	500
Floor rent, 1500 sq. ft. at \$0.20 per sq. ft. = (1500)(0.20)	300
Interest on material in process, at \$0.025 per lb. and 5% interest = (62,000)(0.025)(0.06)	77
Power, 15 kw. at \$0.01 per kw.-hr. = (15)(24)(300)(0.01)	1,080
Water	1,000
Labor, 24 man-hr. per day at \$0.60 and 14 man-hr. per day at \$0.50 per hr. = (24 × 0.60 + 14 × 0.50)(300)	6,420
Total annual costs	\$12,287

Cost per ton of product (not including process costs subsequent to crystallizer) =

$$\frac{12,287}{(31)(300)} = \$1.32$$

It should be noted that the crystals produced in Case 2 are small (but more uniform and regular) than those obtained in Case 1. Costs of centrifuging, drying, screening, reworking of off-size crystals, etc., will be less for Case 2 than for Case 1. If large crystals are demanded, the more expensive vat system would be necessary.

Case 3. Glauber's salt, in a Swenson-Walker crystallizer. Production, 8 tons in 24 hr. The equipment consists of one 24-ft. and one 30-ft. crystallizer. The investment was $2500. Operation is 300 days per year.

Annual Costs:

Depreciation at 10% per year = (0.10)(2500)	$ 250.00
Average interest, allowing for interest earned on depreciation reserve = $({}^{11}/_{10})(2500)(0.05/2)$	68.80
Repairs and maintenance	120.00
Power, 2.25 kw. at $0.0125 per kw.-hr. = (2.25)(24)(300)(0.0125)	203.00
Water, including fixed and operating costs on pumps	600.00
Labor (casual inspection only) 1 hr. per day at $0.60 per hr. = (0.60)(300)	180.00
Total annual cost	$1,421.80

$$\text{Cost per ton of product (at crystallizer outlet)} = \frac{1421.80}{(8)(300)} = \$0.59$$

The following three cases are given by Seavoy and Caldwell (*loc. cit.*). In each case the design factors and costs are given for two methods: first, using a Swenson-Walker crystallizer (Fig. 14) and, second, using a liquid-cooled crystallizer. In all cases the costs include the crystallizer and its accessories only and do not include costs of processing steps subsequent to the crystallization itself.

Case 4. Disodium phosphate, $Na_2HPO_4.12H_2O$.

Case 5. Viscose spin-bath liquor, containing by weight 8 per cent H_2SO_4, 14 per cent Na_2SO_4, 5 per cent glucose, 1 per cent $ZnSO_4$, and 72 per cent H_2O.

Case 6. Sulfuric acid solution containing ferrous sulfate to be crystallized as $FeSO_4.7H_2O$.

The pertinent quantities involved in these cases are shown in the following tables:

Engineering and Cost Data

	4		5		6	
	Liquid cooled	Vacuum	Liquid cooled	Vacuum	Liquid cooled	Vacuum
Type of crystallizer	Swenson-Walker	Three stage	Swenson-Walker	Single stage	Double pipe	Single stage
Type of operation	Cont.	Cont.	Cont.	Batch	Cont.	Batch
Cooling water, gal. per min	360	300	None*	2,900	180†	675
Power for agitation, driver, etc., including pumping, condenser water, h.p.	70	52	50	80	15	30
Power for refrigeration and brine pumping, h.p.	....	....	630	....	160	
Steam consumption, lb. per hr.	....	650	....	8,000	....	2,000
Refrigerated brine, gal. per min.	....	....	535	....	100	
Brine refrigeration, tons per day	....	....	335	....	85	
Water evaporated, lb. per day	....	84,000	....	94,000	....	35,000
Materials of construction	Steel and cast iron	Steel	Nickel and nickel-clad steel	Rubber-lined steel	Steel and lead	Rubber-lined steel
Approx. space (length, width, height).	100′ × 50′ × 10′	30′ × 10′ × 40′	100′ × 50′ × 10′	45′ × 20′ × 55′	30′ × 10′ × 10′	20′ × 15′ × 55′
Installation cost‡	$52,000	$28,000	$140,000	$65,000	$30,000	$18,000
Operating costs, cost per year.						
Interest and depreciation 15 per cent	$7,800	$4,200	$21,000	$9,750	$4,500	$2,700
Cooling water at 1 ct. per 1000 gal.	1,868	1,556	....	15,040	933	3,500
Power at 1 ct. per kw.-hr	4,510	3,350	4,840	5,150	1,450	2,160
Refrigeration at 1.4 kw.-hr. per ton refrigeration	....	....	30,250	....	8,225	
Steam at 30 cts. per 1000 lb.	....	1,685	....	20,750	....	5,184
Labor at 70 cts. per man-hr.	5,328	2,664	10,656	5,328	2,664	2,664
Maintenance	3,120	700	8,400	1,950	3,000	540
Gross operating cost	22,626	14,155	75,146	57,968	20,772	16,748
Credit for water evaporated	....	3,629§	....	12,700‖	....	4,730‖
Net operating cost	$22,626	$10,526	$75,146	$45,268	$20,772	$12,018

* Not including cooling water for ammonia condenser.

† Cool part way with water. Not including cooling for ammonia condenser.

‡ Exclusive of building and foundations.

§ Triple effect economy: 0.4 lb. steam per lb. evaporation.

‖ Single effect economy: 1.25 lb. steam per lb. evaporation.

Operating Conditions and Thermal Data

Item	Case		
	4	5	6
Feed to crystallizer, gal. per 24 hr	80,000	150,000	42,000
Crystals, lb. per 24 hr	300,000	225,000	125,000
Feed density, °Bé	32	26	50
Feed temperature, °F	190	105	140
Final temperature, °F	100	41	50
Initial crystallization temperature, °F	142	58	90
Boiling-point elevation of feed solution, °F	10	8	11
Boiling-point elevation of mother liquor, °F	8	9	15
Available cooling water temp. (max.), °F	80	85	80
Available steam pressure, lb. per sq. in	125	125	125
Specific heat of solution and mother liquor (assumed)	0.85	0.85	0.85
Specific heat of crystals	0.40	0.35	0.30
Heat of crystallization, B.t.u. per lb. crystal	70	105	29
Available refrigerated brine, °F	....	10	20

CAKING OF CRYSTALS

A problem that is often met in handling crystalline products is their tendency to cake or bind together. This is often troublesome in bulk storage or in barreled products, but is most serious in those cases where crystals are sold,in small packages. The difficulty may exist in degrees, varying from loose aggregates that fall apart between the fingers, to solid lumps that can be crushed only by considerable force. The demand of the average consumer that the material shall flow freely from the package makes the prevention of caking a serious problem for the manufacturer.

Critical Humidity. Just as the vapor pressure of water is fixed by its temperature, so the vapor pressure of any solution is fixed by its temperature at an amount somewhat lower than the vapor pressure of water at that temperature. If a saturated solution is brought into contact with air in which the partial pressure of water is less than the vapor pressure of the solution, the solution will evaporate. On the other hand, if the air contains more moisture than this limiting amount, the solution will absorb water until it is so dilute that its vapor pressure is equal to the partial pressure of the moisture of the air with which it is in contact. If a crystal of a soluble salt is in contact with air that contains less water than would be in equilibrium with the saturated solution, the crystal must stay dry, because if it were surrounded with a film of solution, that solution would necessarily evaporate. On the other hand, if the crystal is brought into contact with air containing more moisture than would be in equilibrium with its saturated solution, then the crystal will become damp and in time will absorb water until it is completely dissolved, and the solution is so dilute that it is in equilibrium with the air.

In the range of temperatures around ordinary room temperature, the vapor pressure of a given solution varies with temperature in such a way that it is nearly a constant percentage of the vapor pressure of water at the same temperature. Saturated sodium chloride, for instance, has a vapor pressure approximately 80 per cent of that of water at the same temperature. If sodium chloride, therefore, is brought into contact with air of more than 80 per cent relative humidity it will absorb moisture, while if it is brought into contact with air of less than 80 per cent relative humidity it will stay dry.

From this follows the conception of **critical humidity** of a solid salt. This is the humidity above which it will always become damp and below which it will always stay dry. If the crystal should be coated with impurities derived from the mother liquor from which it was separated (in the case of sodium chloride such impurities would be calcium and magnesium chlorides), this may result in a critical humidity higher or lower than that of the pure salt, according to whether the impurities give solutions having greater or less vapor pressures than that of the salt in question. Consequently, the critical humidity of a commercial grade of a crystalline material may differ appreciably from the critical humidity of the pure substance.

Prevention of Caking. Suppose a sample of sodium chloride be exposed for a short time to an atmosphere more moist than its critical humidity and then that it be removed to an atmosphere less moist than its critical humidity. During the first period it will absorb more moisture, and during the second period it will lose this moisture. If the crystals are large, so that there are relatively few points of contact and there is a large free volume between the crystals, there will probably be no appreciable bonding of the crystals due to this solution and reevaporation, if the time of exposure is not too great. If, on the other hand, the crystals are fine, or have a small percentage of voids, or are in contact with a moist atmosphere for a long time, sufficient moisture may be absorbed to fill the voids entirely with saturated solution; and when this has been reevaporated the crystals will lock into a solid mass. Consequently, to prevent the caking of such salts, the following conditions are desirable: (1) the highest possible critical humidity; (2) a product containing uniform grains with the maximum percentage of voids and the fewest possible points of contact; (3) a coating of powdery inert material that can absorb reasonable amounts of moisture.

The first condition (maximum critical humidity) is often met by removing impurities, such as calcium chloride in the case of common salt, free acid where a salt is formed in acid solution, etc. It often happens that the impurities have a lower critical humidity than the product desired, although this is entirely accidental. To increase the per cent of voids, it is not necessary to produce larger crystals but to produce a more uniform mixture. For a given crystal form, and for absolutely uniform crystals as to size, the per cent of voids is the same no matter what the size of the crystals. A variation in particle size, however, rapidly decreases the per cent of voids. On the other hand, a fine product has more points of contact per unit volume than a coarse one and, hence, a greater tendency to cake. The third remedy is not always applicable. Illustrations of its use are dusting of table salt with magnesia or tricalcium phosphate and the dusting of flake calcium chloride (25 per cent H_2O) with anhydrous calcium chloride.

Some hydrated salts have a melting point so near room temperature that they may sometimes be stored under conditions where fusion begins. Here again the same considerations hold, for, if the percentage of voids is large or the points of contact between adjacent crystals few, the amount of fused material may not be sufficient to lock crystals together on resolidification. If, due to extremely fine crystal size or a mixture of sizes, the percentage of voids is too far reduced or the number of points of contact too greatly increased, the crystals may be firmly locked on resolidification. In this case, also, caking may be partly prevented by dusting the crystals with powdered material. In the case of hydrated salts, this powdery material may be produced from the salt itself by drying under such conditions that a very thin surface layer is dehydrated.

HEATS OF SOLUTION

Inorganic Substances

Heat evolved, in kilogram calories per gram formula weight, on solution in water at 18°C. Computed from data in Bichowsky and Rossini "Thermochemistry of Chemical Substances," Reinhold, 1936

Substance	Dilution*	Formula	Heat, $\frac{\text{kg.-cal.}}{\text{g.-mol}}$
Aluminum bromide	*aq*	$AlBr_3$	+85.3
chloride	600	$AlCl_3$	+77.9
	600	$AlCl_3.6H_2O$	+13.2
fluoride	*aq*	AlF_3	+31
	aq	$AlF_3.\frac{1}{2}H_2O$	+19.0
	aq	$AlF_3.3\frac{1}{2}H_2O$	−1.7
iodide	*aq*	AlI_3	+89.0
sulfate	*aq*	$Al_2(SO_4)_3$	+126
	aq	$Al_2(SO_4)_3.6H_2O$	+56.2
	aq	$Al_2(SO_4)_3.18H_2O$	+6.7
Ammonium bromide	*aq*	NH_4Br	−4.45
chloride	∞	NH_4Cl	−3.82
chromate	*aq*	$(NH_4)_2CrO_4$	−5.82
dichromate	600	$(NH_4)_2Cr_2O_7$	−12.9
iodide	*aq*	NH_4I	−3.56
nitrate	∞	NH_4NO_3	−6.47
perborate	*aq*	$NH_4BO_3.H_2O$	−9.0
sulfate	∞	$(NH_4)_2SO_4$	−2.75
sulfate, acid	800	NH_4HSO_4	+0.56
sulfite	*aq*	$(NH_4)_2SO_3$	−1.2
	aq	$(NH_4)_2SO_3.H_2O$	−4.13
Antimony fluoride	*aq*	SbF_3	−1.7
iodide	*aq*	SbI_3	−0.8
Arsenic acid	*aq*	H_3AsO_4	−0.4
Barium bromate	∞	$Ba(BrO_3)_2.H_2O$	−15.9
bromide	∞	$BaBr_2$	+5.3
	∞	$BaBr_2.H_2O$	−0.8
	∞	$BaBr_2.2H_2O$	−3.87
chlorate	∞	$Ba(ClO_3)_2$	−6.7
	∞	$Ba(ClO_3)_2.H_2O$	−10.6
chloride	∞	$BaCl_2$	+2.4
	∞	$BaCl_2.H_2O$	−2.17
	∞	$BaCl_2.2H_2O$	−4.5
cyanide	*aq*	$Ba(CN)_2$	+1.5
	aq	$Ba(CN)_2.H_2O$	−2.4
	aq	$Ba(CN)_2.2H_2O$	−4.9
iodate	∞	$Ba(IO_3)_2$	−9.1
	∞	$Ba(IO_3)_2.H_2O$	−11.3
iodide	∞	BaI_2	+10.5
	∞	$BaI_2.H_2O$	+2.7
	∞	$BaI_2.2H_2O$	+0.14
	∞	$BaI_2.2\frac{1}{2}H_2O$	−0.58
	∞	$BaI_2.7H_2O$	−6.61
nitrate	∞	$Ba(NO_3)_2$	−10.2
perchlorate	∞	$Ba(ClO_4)_2$	−2.8
	∞	$Ba(ClO_4)_2.3H_2O$	−10.5
sulfide	∞	BaS	+7.2
Beryllium bromide	*aq*	$BeBr_2$	+62.6
chloride	*aq*	$BeCl_2$	+51.1
iodide	*aq*	BeI_2	+72.6
sulfate	*aq*	$BeSO_4$	+18.1
	aq	$BeSO_4.H_2O$	+13.5
	aq	$BeSO_4.2H_2O$	+7.9
	aq	$BeSO_4.4H_2O$	+1.1

---

* The numbers represent mols of water used to dissolve 1 g. formula weight of substance; ∞ means "infinite dilution"; and *aq* means "aqueous solution of unspecified dilution."

Inorganic Substances.—(*Continued*)

Substance	Dilution	Formula	Heat, $\frac{\text{kg.-cal.}}{\text{g.-mol}}$
Bismuth iodide	*aq*	BiI_3	+3
Boric acid	*aq*	H_3BO_3	-5.4
Cadmium bromide	400	$CdBr_2$	+0.4
	400	$CdBr_2.4H_2O$	-7.3
chloride	400	$CdCl_2$	+3.1
	400	$CdCl_2.H_2O$	+0.6
	400	$CdCl_2.2\frac{1}{2}H_2O$	-3.00
	400	$CdCl_2.4H_2O$	-93.7
nitrate	400	$Cd(NO_3)_2.H_2O$	+4.17
	400	$Cd(NO_3)_2.4H_2O$	-5.08
sulfate	400	$CdSO_4$	+10.69
	400	$CdSO_4.H_2O$	+6.05
	400	$CdSO_4.2\frac{2}{3}H_2O$	+2.51
Calcium acetate	∞	$Ca(C_2H_3O_2)_2$	+7.6
	∞	$Ca(C_2H_3O_2)_2.H_2O$	+6.5
bromide	∞	$CaBr_2$	+24.86
	∞	$CaBr_2.6H_2O$	-0.9
chloride	∞	$CaCl_2$	+4.9
	∞	$CaCl_2.H_2O$	+12.3
	∞	$CaCl_2.2H_2O$	+12.5
	∞	$CaCl_2.4H_2O$	+2.4
	∞	$CaCl_2.6H_2O$	-4.11
formate	400	$Ca(CHO_2)_2$	+0.7
iodide	∞	CaI_2	+28.0
	∞	$CaI_2.8H_2O$	+1.8
nitrate	∞	$Ca(NO_3)_2$	+4.1
	∞	$Ca(NO_3)_2.H_2O$	+0.7
	∞	$Ca(NO_3)_2.2H_2O$	-3.2
	∞	$Ca(NO_3)_2.3H_2O$	-4.2
	∞	$Ca(NO_3)_2.4H_2O$	-7.99
phosphate, mono-	*aq*	$Ca(H_2PO_4)_2.H_2O$	-0.6
dibasic	*aq*	$CaHPO_4.2H_2O$	-1
sulfate	∞	$CaSO_4$	+5.1
	∞	$CaSO_4.\frac{1}{2}H_2O$	+3.6
	∞	$CaSO_4.2H_2O$	-0.18
Chromous chloride	*aq*	$CrCl_2$	+18.6
		$CrCl_2.3H_2O$	+5.3
		$CrCl_2.4H_2O$	+3.0
iodide	*aq*	CrI_2	+6.3
Cobaltous bromide	*aq*	$CoBr_2$	+18.4
	aq	$CoBr_2.6H_2O$	-1.25
chloride	400	$CoCl_2$	+18.5
	400	$CoCl_2.2H_2O$	+9.8
	400	$CoCl_2.6H_2O$	-2.9
iodide	*aq*	CoI_2	+18.8
sulfate	400	$CoSO_4$	+15.0
	400	$CoSO_4.6H_2O$	-1.4
	400	$CoSO_4.7H_2O$	-3.6
Cupric acetate	*aq*	$Cu(C_2H_3O_2)_2$	+2.4
formate	*aq*	$Cu(CHO_2)_2$	+0.5
nitrate	200	$Cu(NO_3)_2$	+10.3
	200	$Cu(NO_3)_2.3H_2O$	-2.6
	200	$Cu(NO_3)_2.6H_2O$	-10.7
sulfate	800	$CuSO_4$	+15.9
		$CuSO_4.H_2O$	+9.3
		$CuSO_4.3H_2O$	+3.65
		$CuSO_4.5H_2O$	-2.85
Cuprous sulfate	*aq*	Cu_2SO_4	+11.6
Ferric chloride	1000	$FeCl_3$	+31.7
	1000	$FeCl_3.2\frac{1}{2}H_2O$	+21.0
	1000	$FeCl_3.6H_2O$	+5.6
nitrate	800	$Fe(NO_3)_3.9H_2O$	-9.1

Inorganic Substances.—(*Continued*)

Substance	Dilution	Formula	Heat, $\frac{\text{kg-cal.}}{\text{g.-mol}}$
Ferrous bromide	*aq*	$FeBr_2$	+18.0
chloride	400	$FeCl_2$	+17.9
	400	$FeCl_2.2H_2O$	+8.7
	400	$FeCl_2.4H_2O$	+2.7
iodide	*aq*	FeI_2	+23.3
sulfate	400	$FeSO_4$	+14.7
	400	$FeSO_4.H_2O$	+7.35
	400	$FeSO_4.4H_2O$	+1.4
	400	$FeSO_4.7H_2O$	−4.4
Lead acetate	400	$Pb(C_2H_3O_2)_2$	+1.4
	400	$Pb(C_2H_3O_2)_2.3H_2O$	−5.9
bromide	*aq*	$PbBr_2$	−10.1
chloride	*aq*	$PbCl_2$	−3.4
formate	*aq*	$Pb(CHO_2)_2$	−6.9
nitrate	400	$Pb(NO_3)_2$	−7.61
Lithium bromide	∞	$LiBr$	+11.54
	∞	$LiBr.H_2O$	+5.30
	∞	$LiBr.2H_2O$	+2.05
	∞	$LiBr.3H_2O$	−1.59
chloride	∞	$LiCl$	+8.66
	∞	$LiCl.H_2O$	+4.45
	∞	$LiCl.2H_2O$	+1.07
	∞	$LiCl.3H_2O$	−1.98
fluoride	∞	LiF	−0.74
hydroxide	∞	$LiOH$	+4.74
	∞	$LiOH.\frac{1}{8}H_2O$	+4.39
	∞	$LiOH.H_2O$	+9.6
iodide	∞	LiI	+14.92
	∞	$LiI.\frac{1}{2}H_2O$	+10.08
	∞	$LiI.H_2O$	+6.93
	∞	$LiI.2H_2O$	+3.43
	∞	$LiI.3H_2O$	−0.17
nitrate	∞	$LiNO_3$	+0.466
	∞	$LiNO_3.3H_2O$	−7.87
sulfate	∞	$LiSO_4$	+6.71
	∞	$LiSO_4.H_2O$	+3.77
Magnesium bromide	∞	$MgBr_2$	+43.7
	∞	$MgBr_2.H_2O$	+35.9
	∞	$MgBr_2.6H_2O$	+19.8
chloride	∞	$MgCl_2$	+36.3
	∞	$MgCl_2.2H_2O$	+20.8
	∞	$MgCl_2.4H_2O$	+10.5
	∞	$MgCl_2.6H_2O$	+3.4
iodide	∞	MgI_2	+50.2
nitrate	∞	$Mg(NO_3)_2.6H_2O$	−3.7
phosphate	*aq*	$Mg_3(PO_4)_2$	+10.2
sulfate	∞	$MgSO_4$	+21.1
	∞	$MgSO_4.H_2O$	+14.0
	∞	$MgSO_4.2H_2O$	+11.7
	∞	$MgSO_4.4H_2O$	+4.9
	∞	$MgSO_4.6H_2O$	+0.55
	∞	$MgSO_4.7H_2O$	−3.18
sulfide	*aq*	MgS	+25.8
Manganic nitrate	400	$Mn(NO_3)_2$	+12.9
	400	$Mn(NO_3)_2.3H_2O$	−3.9
	400	$Mn(NO_3)_2.6H_2O$	−6.2
sulfate	*aq*	$Mn_2(SO_4)_3$	+22
Manganous acetate	*aq*	$Mn(C_2H_3O_2)_2$	+12.2
	aq	$Mn(C_2H_3O_2)_2.4H_2O$	+1.6

Inorganic Substances.—(*Continued*)

Substance	Dilution	Formula	Heat, $\frac{\text{kg-cal.}}{\text{g.-mol}}$
Manganous, bromide	*aq*	$MnBr_2$	+15
	aq	$MnBr_2.H_2O$	+14.4
	aq	$MnBr_2.4H_2O$	+16.1
chloride	400	$MnCl_2$	+16.0
	400	$MnCl_2.2H_2O$	+8.2
	400	$MnCl_2.4H_2O$	+1.5
formate	*aq*	$Mn(CHO_2)_2$	+4.3
	aq	$Mn(CHO_2)_2.2H_2O$	−2.9
iodide	*aq*	MnI_2	+26.2
	aq	$MnI_2.H_2O$	+24.1
	aq	$MnI_2.2H_2O$	+22.7
	aq	$MnI_2.4H_2O$	+19.9
	aq	$MnI_2.6H_2O$	+21.2
sulfate	400	$MnSO_4$	+13.8
	400	$MnSO_4.H_2O$	+11.9
	400	$MnSO_4.7H_2O$	−1.7
Mercuric acetate	*aq*	$Hg(C_2H_3O_2)_2$	−4.0
bromide	*aq*	$HgBr_2$	−2.4
chloride	*aq*	$HgCl_2$	−3.3
nitrate	*aq*	$Hg(NO_3)_2.\frac{1}{2}H_2O$	−0.7
Mercurous nitrate	*aq*	$Hg_2(NO_3)_2.2H_2O$	−11.5
Nickel bromide	*aq*	$NiBr_2$	+19.0
	aq	$NiBr_2.3H_2O$	+0.2
chloride	800	$NiCl_2$	+19.23
	800	$NiCl_2.2H_2O$	+10.4
	800	$NiCl_2.4H_2O$	+4.2
	800	$NiCl_2.6H_2O$	−1.15
iodide	*aq*	NiI_2	+19.4
nitrate	200	$Ni(NO_3)_2$	+11.8
	200	$Ni(NO_3)_2.6H_2O$	−7.5
sulfate	200	$NiSO_4$	+15.1
	200	$NiSO_4.7H_2O$	−4.2
Phosphoric acid, ortho-	400	H_3PO_4	+2.79
	400	$H_3PO_4.\frac{1}{2}H_2O$	−0.1
pyro-	*aq*	$H_4P_2O_7$	+25.9
	aq	$H_4P_2O_7.1\frac{1}{2}H_2O$	+4.65
Potassium acetate	∞	$KC_2H_3O_2$	+3.55
aluminum sulfate	600	$KAl(SO_4)_2$	+48.5
	600	$KAl(SO_4)_2.3H_2O$	+26.6
		$KAl(SO_4)_2.12H_2O$	−10.1
bicarbonate	2000	$KHCO_3$	−5.1
bromate	∞	$KBrO_3$	−10.13
bromide	∞	KBr	−5.13
carbonate	∞	K_2CO_3	+6.58
		$K_2CO_3.\frac{1}{2}H_2O$	+4.25
		$K_2CO_3.1\frac{1}{2}H_2O$	−0.43
chlorate	∞	$KClO_3$	−10.31
chloride	∞	KCl	−4.404
chromate	2185	K_2CrO_4	−4.9
chrome sulfate	600	$KCr(SO_4)_2$	+55
		$KCr(SO_4)_2.H_2O$	+42
		$KCr(SO_4)_2.2H_2O$	+33
		$KCr(SO_4)_2.6H_2O$	+7
		$KCr(SO_4)_2.12H_2O$	−9.5
cyanide	200	KCN	−3.0
dichromate	1600	$K_2Cr_2O_7$	−17.8
fluoride	∞	KF	+3.96
	∞	$KF.2H_2O$	−1.85
	∞	$KF.4H_2O$	−6.05
hydrosulfide	∞	KHS	+0.86
	∞	$KHS.\frac{1}{4}H_2O$	+1.21

Inorganic Substances.—*(Continued)*

Substance	Dilution	Formula	Heat, $\frac{\text{kg.-cal.}}{\text{g.-mol}}$
Potassium, hydroxide	∞	KOH	+12.91
	∞	$KOH.\frac{3}{4}H_2O$	+4.27
	∞	$KOH.H_2O$	+3.48
	∞	$KOH.7H_2O$	+0.86
iodate	∞	KIO_3	−6.93
iodide	∞	KI	−5.23
nitrate	∞	KNO_3	−8.633
oxalate	400	$K_2C_2O_4$	−4.6
		$K_2C_2O_4.H_2O$	−7.5
perchlorate	∞	$KClO_4$	−12.94
permanganate	400	$KMnO_4$	−10.4
phosphate, dihydrogen	*aq*	KH_2PO_4	+4.7
pyrosulfite	*aq*	$K_2S_2O_5$	−11.0
	aq	$K_2S_2O_5.\frac{1}{2}H_2O$	−10.22
sulfate	∞	K_2SO_4	−6.32
sulfate, acid	800	$KHSO_4$	−3.10
sulfide	∞	K_2S	−11.0
sulfite	*aq*	K_2SO_3	+1.8
	aq	$K_3SO_3.H_2O$	+1.37
thiocyanate	∞	$KCNS$	−6.08
thionate, di-	*aq*	$K_2S_2O_6$	−13.0
thiosulfate	∞	$K_2S_2O_3$	−4.5
Silver acetate	*aq*	$AgC_2H_3O_2$	−5.4
nitrate	200	$AgNO_3$	−4.4
Sodium acetate	∞	$NaC_2H_3O_2$	+4.085
	∞	$NaC_2H_3O_2.3H_2O$	−4.665
arsenate	500	Na_3AsO_4	+15.6
	500	$Na_3AsO_4.12H_2O$	−12.61
bicarbonate	1800	$NaHCO_3$	−4.1
borate, tetra-	900	$Na_2B_4O_7$	+10.0
	900	$Na_2B_4O_7.10H_2O$	−16.8
bromide	∞	$NaBr$	−0.58
	∞	$NaBr.2H_2O$	−4.57
carbonate	∞	Na_2CO_3	+5.57
	∞	$Na_2CO_3.H_2O$	+2.19
	∞	$Na_2CO_3.7H_2O$	−10.81
	∞	$Na_2CO_3.10H_2O$	−16.22
chlorate	∞	$NaClO_3$	−5.37
chloride	∞	$NaCl$	−1.164
chromate	800	Na_2CrO_4	+2.50
	800	$Na_2CrO_4.4H_2O$	−7.52
	800	$Na_2CrO_4.10H_2O$	−16.0
cyanide	200	$NaCN$	−0.37
	200	$NaCN.\frac{1}{2}H_2O$	−0.92
	200	$NaCN.2H_2O$	−4.41
fluoride	∞	NaF	−0.27
hydrosulfide	∞	$NaHS$	+4.62
	∞	$NaHS.2H_2O$	−1.49
hydroxide	∞	$NaOH$	+10.18
	∞	$NaOH.\frac{1}{2}H_2O$	+8.17
	∞	$NaOH.\frac{2}{3}H_2O$	+7.08
	∞	$NaOH.\frac{3}{4}H_2O$	+6.48
	∞	$NaOH.H_2O$	+5.17
iodide	∞	NaI	+1.57
	∞	$NaI\ 2H_2O$	−3.89
metaphosphate	600	$NaPO_3$	+3.97
nitrate	∞	$NaNO_3$	−5.05
nitrite	*aq*	$NaNO_2$	−3.6
perchlorate	∞	$NaClO_4$	−4.15
phosphate, di-	1600	Na_2HPO_4	+5.21
tri-	1600	Na_3PO_4	+13

Inorganic Substances.—(*Concluded*)

Substance	Dilution	Formula	Heat, $\frac{\text{kg.-cal.}}{\text{g.-mol}}$
Sodium, phosphate	1600	$Na_3PO_4.12H_2O$	−15.3
di-	1600	$Na_2HPO_4.2H_2O$	−0.82
	1600	$Na_2HPO_4.7H_2O$	−12.04
	1600	$Na_2HPO_4.12H_2O$	−23.18
phosphite, mono-	600	NaH_2PO_3	+0.90
	600	$NaH_2PO_3.2\frac{1}{2}H_2O$	−5.29
di-	800	Na_2HPO_3	+9.30
	800	$Na_2HPO_3.5H_2O$	−4.54
pyrophosphate	1600	$Na_4P_2O_7$	+11.9
	1600	$Na_4P_2O_7.10H_2O$	−11.7
di-	1200	$Na_2H_2P_2O_7$	−2.2
	1200	$Na_2H_2P_2O_7.6H_2O$	−14.0
sulfate	∞	Na_2SO_4	+0.28
	∞	$Na_2SO_4.10H_2O$	−18.74
sulfate, acid	800	$NaHSO_4$	+1.74
	800	$NaHSO_4.H_2O$	+0.15
sulfide	∞	Na_2S	+15.2
	∞	$Na_2S.4\frac{1}{2}H_2O$	+0.09
	∞	$Na_2S.5H_2O$	−6.54
	∞	$Na_2S.9H_2O$	−16.65
sulfite	∞	Na_2SO_3	+2.8
	∞	$Na_2SO_3.7H_2O$	−11.1
thiocyanate	∞	$NaCNS$	−1.83
thionate, di-	*aq*	$Na_2S_2O_6$	−5.80
	aq	$Na_2S_2O_6.2H_2O$	−11.86
thiosulfate	*aq*	$Na_2S_2O_3$	+2.0
	aq	$Na_2S_2O_3.5H_2O$	−11.30
Stannic bromide	*aq*	$SnBr_4$	+15.5
Stannous bromide	*aq*	$SnBr_2$	−1.6
iodide	*aq*	SnI_2	−5.8
Strontium acetate	∞	$Sr(C_2H_3O_2)_2$	+6.2
	∞	$Sr(C_2H_3O_2)_2.\frac{1}{2}H_2O$	+5.9
bromide	∞	$SrBr_2$	+16.4
	∞	$SrBr_2.H_2O$	+9.25
	∞	$SrBr_2.2H_2O$	+6.5
	∞	$SrBr_2.4H_2O$	+0.4
	∞	$SrBr_2.6H_2O$	−6.1
chloride	∞	$SrCl_2$	+11.54
	∞	$SrCl_2.H_2O$	+6.4
	∞	$SrCl_2.2H_2O$	+2.95
	∞	$SrCl_2.2H_2O$	−7.1
iodide	∞	SrI_2	+20.7
	∞	$SrI_2.H_2O$	+12.65
	∞	$SrI_2.2H_2O$	+10.4
	∞	$SrI_2.6H_2O$	−4.5
nitrate	∞	$Sr(NO_3)_2$	−4.8
	∞	$Sr(NO_3)_2.4H_2O$	−12.4
sulfate	∞	$SrSO_4$	+0.5
Sulfuric acid, pyro-	∞	$H_2S_2O_7$	−18.08
Zinc acetate	400	$Zn(C_2H_3O_2)_2$	+9.8
	400	$Zn(C_2H_3O_2)_2.H_2O$	+7.0
	400	$Zn(C_2H_3O_2)_2.2H_2O$	+3.9
bromide	400	$ZnBr_2$	+15.0
chloride	400	$ZnCl_2$	+15.72
iodide	*aq*	ZnI_2	+11.6
nitrate	400	$Zn(NO_3)_2.3H_2O$	−5
	400	$Zn(NO_3)_2.6H_2O$	−6.0
sulfate	400	$ZnSO_4$	+18.5
	400	$ZnSO_4.H_2O$	+10.0
	400	$ZnSO_4.6H_2O$	−0.8
	400	$ZnSO_4.7H_2O$	−4.3

Organic Compounds in Water (at Infinite Dilution and Approximately Room Temperature)

(Recalculated and rearranged from "International Critical Tables," vol. 5, pp. 148–150)

Solute	Heat of Solution, G.-cal. per G.-mol Solute*
Acetic acid (solid), $C_2H_4O_2$	−2,251
Acetylacetone, $C_5H_8O_2$	−641
Acetylurea, $C_3H_6N_2O_2$	−6,812
Aconitic acid, $C_6H_6O_6$	−4,206
Ammonium benzoate, $C_7H_9NO_2$	−2,700
picrate	−8,700
succinate (n-)	−3,489
Aniline, hydrochloride, C_6H_8ClN	−2,732
Barium picrate	−4,708
Benzoic acid, $C_7H_6O_2$	−6,501
Camphoric acid, $C_{10}H_{16}O_4$	−502
Citric acid, $C_6H_8O_7$	−5,401
Dextrin, $C_{12}H_{20}O_{10}$	268
Fumaric acid, $C_4H_4O_4$	−5,903
Hexamethylenetetramine, $C_6H_{12}N_4$	4,780
Hydroxybenzamide (m-), $C_7H_7NO_2$	−4,161
(m-) (HCl)	−7,003
(o-) $C_7H_7NO_2$	−4,340
(p-)	−5,392
Hydroxybenzoic acid (o-), $C_7H_6O_3$	−6,350
(p-), $C_7H_6O_3$	−5,781
Hydroxybenzyl alcohol (o-), $C_7H_8O_2$	−3,203
Inulin, $C_{36}H_{62}O_{31}$	−96
Itaconic acid, $C_5H_6O_4$	−5,922
Isosuccinic acid, $C_4H_6O_4$	−3,420
Lactose, $C_{12}H_{22}O_{11}.H_2O$	−3,705
Lead picrate	−7,098
$(2H_2O)$	−13,193
Magnesium picrate	14,699
$(8H_2O)$	−15,894
Maleic acid, $C_4H_4O_4$	−4,441
Malic acid, $C_4H_6O_5$	−3,150
Malonic acid, $C_3H_4O_4$	−4,493
Mandelic acid, $C_8H_8O_3$	−3,090
Mannitol, $C_6H_{14}O_6$	−5,260
Menthol, $C_{10}H_{20}O$	0
Nicotine dihydrochloride, $C_{10}H_{16}Cl_2N_2$	6,561
Nitrobenzoic acid (m-) $C_7H_5NO_4$	−5,593
(o-), $C_7H_5NO_4$	−5,306
(p-) $C_7H_5NO_4$	−8,891

* + denotes heat evolved, and − denotes heat absorbed. All values are positive unless otherwise noted. The data in the "International Critical Tables" were calculated by E. Anderson.

Organic Compounds in Water (at Infinite Dilution and Approximately Room Temperature)—(*Concluded*)

Solute	Heat of Solution, G.-cal. per G.-mol Solute
Nitrophenol (m-) $C_6H_5NO_2$	−5,210
(p-) $C_6H_5NO_2$	−6,310
(p-) $C_6H_5NO_2$	−4,493
Oxalic acid, $C_2H_2O_4$	−2,290
($2H_2O$)	−8,485
Phenol (solid), C_6H_6O	−2,605
Phthalic acid, $C_8H_6O_4$	−4,871
Picric acid, $C_6H_3N_3O_7$	−7,098
Piperic acid, $C_{12}H_{10}O_4$	−10,492
Piperonylic acid, $C_8H_6O_4$	−9,106
Potassium benzoate	−1,506
citrate	2,820
tartrate (n-) (0.5 H_2O)	−5,562
Pyrogallol, $C_6H_6O_3$	−3,705
Pyrotartaric acid	−5,019
Quinone	−3,991
Raffinose, $C_{18}H_{32}O_{16}$ ($5H_2O$)	−9,703
Resorcinol, $C_6H_6O_2$	−3,960
Silver malonate (n-)	−9,799
Sodium citrate (tri-)	5,270
picrate	−6,441
potassium tartrate	−1,817
($4H_2O$)	−12,342
succinate (n-)	2,390
($6H_2O$)	−10,994
tartrate (n-)	−1,121
($2H_2O$)	−5,882
Strontium picrate	7,887
($6H_2O$)	−14,412
Succinic acid, $C_4H_6O_4$	−6,405
Succinimide, $C_4H_5NO_2$	−4,302
Sucrose, $C_{12}H_{22}O_{11}$	−1,319
Tartaric acid (d-)	−3,451
Thiourea, CH_4N_2S	−5,330
Urea, CH_4N_2O	−3,609
acetate	−8,795
formate	−7,194
nitrate	−10,803
oxalate	−17,806
Vanillic acid	−5,160
Vanillin	−5,210
Zinc picrate	−11,496
($8H_2O$)	−15,894

CENTRIFUGES

BY A. E. FLOWERS

Definitions. A **centrifugal force** is produced by any moving mass that is compelled to depart from the rectilinear path which it tends to follow, the force being exerted in the direction away from the center of curvature of its path. A **centrifuge** is a machine designed to subject material, held in it or being passed through it, to centrifugal force. A **batch centrifuge** holds material so that it may be subjected to centrifugal force. A **continuous centrifuge** passes material in a steady stream through a part of the centrifugal apparatus, where it is subjected to centrifugal force and continuously discharges the separated components. A **centrifugal clarifier** is a machine designed to subject a mass or stream of liquid to centrifugal force, thereby clearing it by removing any small particles that were in it. A **centrifugal separator** is a machine designed to subject a mass or continuous stream of mixed liquids to centrifugal force, thereby effecting their separation. A **centrifugal purifier** is a machine designed to remove, centrifugally, foreign materials (such as water or dirt) from a liquid passed continuously through it, thereby purifying it. A **test-tube centrifuge** carries cups or graduated test tubes to hold material to be subjected to centrifugal force and is usually employed for the purpose of measuring the quantities of the separable components present and of estimating their separability. A **basket centrifuge** is a machine designed to hold a mass of material (such as clothes, masses of crystals, etc.) and subject them to centrifugal force for the purpose of removing water or other liquids. Such machines usually have fairly large "baskets" or "bowls" and are operated at moderate speeds. (The term *centrifugal drier* is often used for such machines but is less appropriate since complete drying is not usually carried out by such machines alone.) A basket centrifuge of the overflow type discharges the liquid over the basket lip, retaining the solids in the basket. A **centrifugal filter** is a centrifuge carrying a filtering medium (cloth, paper, or metal screen) to catch, or strain out, the solids to be retained in the centrifuge while allowing the liquids to pass through and be freely discharged.

THEORY

Force

The force on a particle, compelled to move in a circular path, is determined by the "acceleration" toward the center, or rate of change of velocity direction away from the linear path any rotating particle tends to follow.

The centrifugal force may, therefore, be expressed as:

$$F = \frac{W}{g} r\omega^2 \qquad (1)$$

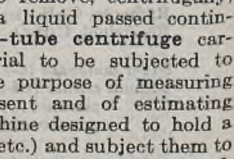


FIG. 1.—Derivation of centrifugal force.

where F is centrifugal force in grams; W is weight of particle in grams; r is radius of curvature of path in centimeters; ω is angular velocity in radians per second; g is acceleration of gravity, usually taken as 981 in centimeters per second per second.

Also

$$F = \frac{WV^2}{gr} \tag{2}$$

where V is peripheral velocity in centimeters per second or

$$F = \frac{\pi^2}{900}\frac{W}{g}r\ (\text{r.p.m.})^2 \tag{3}$$

$$F = 1.118Wr\ (\text{r.p.m.})^2\ 10^{-5}$$

where the angular velocity is expressed in revolutions per minute (r.p.m.).

The force on a ring section, or cylindrical element, of the bowl, or vessel, required to hold the material being processed, due to its own mass, may be expressed approximately as follows:

$$T_r = \frac{\delta_r}{g}r^2\omega^2 \tag{4}$$

or

$$T_r = \frac{\delta_r}{g}V^2 \tag{5}$$

$$T_r = \frac{\pi^2}{900}\frac{\delta_r}{g}r^2\ (\text{r.p.m.})^2 \tag{6}$$

FIG. 2.—Derivation of centrifugal forces on a ring section.

where T_r is the unit stress in the ring section in grams per square centimeter; δ_r is the specific gravity of the wall material in grams per cubic centimeter; r is the mean radius of the ring in centimeters.

The force on a liquid at various depths results in a unit liquid pressure which may be expressed as follows:

$$P_l = \frac{\delta_l}{2g}\omega^2(r_2^2 - r_1^2) \tag{7}$$

where r_2 is the radius at any point in the liquid bed for which the pressure is to be calculated; r_1 is the radius of the inner liquid surface.

FIG. 3.—Derivation of centrifugal force on a liquid.

If the radial thickness of the bowl-shell wall is expressed by the symbol s then the average unit tension produced by the liquid pressure P_l at the inner wall of the bowl shell is:

$$T_l = \frac{\delta_l r_2}{2gs}\omega^2(r_2^2 - r_1^2) \tag{8}$$

or

$$T_l = \frac{\delta_l V^2}{2gr_2s}(r_2^2 - r_1^2) \tag{9}$$

or

$$T_l = \frac{\pi^2\delta_l}{1800gs}r_2(r_2^2 - r_1^2)(\text{r.p.m.})^2 \tag{10}$$

The total stress in the bowl-shell wall is the sum of that due to the wall material and that due to the liquid bed, assuming s small in comparison to r_2, so that r_2 may be substituted for r in Eq. (4).

$$T_b = T_r + T_l = \frac{r_2\omega^2}{g}\left[r_2\delta_r + \frac{\delta_l}{2s}(r_2^2 - r_1^2) \right] \tag{11}$$

The force on balanced columns of different liquids of different specific gravities which are being separated centrifugally determines the radial position of the line of division between the liquids, since the heavier liquid takes the outer position in the bowl and, owing to its greater specific gravity, will discharge over a lip or through a non-restricted opening farther from the axis of rotation than the opening for the lighter component. If the velocities of the liquid through the bowl passages are not so great as to set up considerable viscous and turbulent flow resistances, and particularly if the flow lines are *radial*, out and back (so that slippage is avoided), then the two liquids will balance each other about a cylindrical zone, with the lighter component filling the inner space to its small diameter discharge lip and the heavier component the outer part of the bowl, past the partition wall (or top disk) and back to its larger discharge lip.

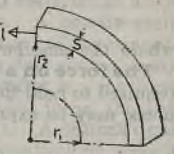


FIG. 4.—Derivation of centrifugal stresses on a shell due to a liquid.

The position of this neutral zone and that of the partition wall are important factors in the performance of a centrifuge.

The partition wall is usually given as large a diameter as the inside wall of the bowl shell will allow in order to allow a wide range of positions for the neutral zone. The position of the neutral zone governs the proportion of the total centrifugal effect applied to each component. Obviously when the neutral zone is near the center the lighter component gets but a small amount of exposure to the centrifugal separation or purification, the major effects being exerted on the heavier component. The cream separator is an example of this kind in which the performance is judged by the ability to remove the last hundredths of a per cent of butter fat from the heavier skim milk.

When the neutral zone is near the bowl wall, or rather the partition wall, the major centrifugal effects are exerted on the lighter component. The oil purifier for removing moisture from electrical insulating oil is an example of this kind where the last thousandth of 1 per cent of moisture must be removed from the oil in order to get a high dielectric-strength test.

Since the neutral zone is at the line between the two liquids, it is desirable that the feed should be brought in or through the bowl as near to the neutral zone as is feasible.

The exact position of the neutral zone can be adjusted by adjusting the distance from the center, of *either* the inner or the outer of the two discharge lips.

It may also vary, or be varied, if either of the two components on their way from the neutral zone to their respective discharge lips are not guided radially, so that some slippage occurs. If slippage occurs in a component moving toward the center then it will retain a portion of the peripheral velocity it possessed at the larger diameter and will "run ahead" or have a greater rotational velocity than the corresponding radius of the bowl part. This will result in a greater back pressure which will in turn restrict the flow of this component and displace the neutral zone.

If slippage occurs on a component moving to the outer parts of the bowl it will "run behind" the bowl and so having a lower rotational velocity will exert less pressure on the bowl shell and in the radial space between the neutral zone and the partition wall. Slippage is useful as a means of control of the ratio of the two discharges to each other in constant through-put types of machines such as cream separators.

The expression for the relation of the position of the neutral zone or balance circle for radial flow (no slippage) and negligible fluid flow friction may be developed as follows:

The static pressure on the bed of light liquid L between the inner discharge lip and the neutral zone must equal that of the bed of heavy liquid H, where the two lips allow a free discharge into receiving covers or vessels under equal pressures.

From Eq. (7) the liquid pressures may, therefore, be stated as

$$P_L \text{ (at the neutral zone } r_n) = P_H$$

Therefore

$$\frac{\delta_{li}\omega^2}{2g}(r_n^2 - r_i^2) = \frac{\delta_{lo}\omega^2}{2g}(r_n^2 - r_o^2)$$

$$\delta_{li}(r_n^2 - r_i^2) = \delta_{lo}(r_n^2 - r_o^2)$$

$$r_n = \sqrt{\frac{\delta_{lo}r_o^2 - \delta_{li}r_i^2}{\delta_{lo} - \delta_{li}}} = \sqrt{\frac{r_o^2 - r_i^2(\delta_{li}/\delta_{lo})}{1 - \delta_{li}/\delta_{lo}}} \qquad (12)$$

The difference in the densities of the two liquids exerts the major effect on the position and stability of the balance circle. If this difference is too small, separation becomes difficult through sensitiveness to other, though small, disturbing influences; or even impossible, though separations may be effected easily with differences in densities as small as 3 per cent, and a fraction of 1 per cent by special constructions.

The expression in Eq. (12) shows that the position of the neutral zone radius r_n may be adjusted by increasing or decreasing the radius of either the inner lip r_i or the outer lip r_o to compensate for differences in the specific gravity ratio of the lighter to that of the heavier liquid. Both are made use of in different types of bowls, though more often the adjustment of the radius of the outer lip for the heavier component is the one employed.

The forces producing the through-put or pumping capacity have no necessary connection with the separating or the clarifying capacity (though often confused with them), as it may be seen from Fig. 5 that a bowl must function as a centrifugal pump with a pumping head determined by the radial columns or heads A and B between the check ring, whose radius r_c determines the overflow capacity, in relation to the radii of the inner and outer lips r_i and r_o. Thus the head for the lighter liquid is $A = r_i - r_c$, and the head for the heavier liquid is $B = r_o - r_c$.

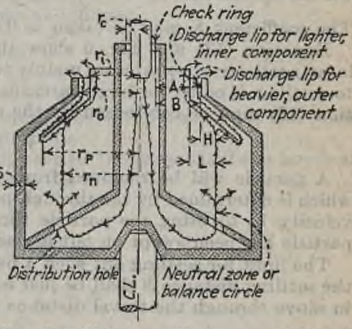


FIG. 5.—Diagrammatic representation of the principle of a separating bowl.

The pumping capacity is finally determined by these forces, and by the fluid friction-resisting effects, mainly turbulent friction. The volumetric pumping capacity varies but little, if at all, with the specific gravity of the liquid, as the pumping force increases along with the turbulent resistance to flow.

It is obvious, however, that the passages into and through the bowl should be ample in cross section and free from unnecessary sharp angles, turns, constrictions, and pockets.

It is undesirable to have the pumping capacity very greatly in excess of the separating or clarifying capacity, as this may result in only partially filling the bowl or the introduction of air drawn in with the feed.

Liquid-solid Separation (Clarification)

Settling Rate for Large Particles—Turbulent Motion. The centrifugal force acting on a large particle moving through a liquid of different net specific gravity brings the particle to a terminal velocity at which the resisting force, due to the hurling of the liquid out of the path of the particle, equals the centrifugal force. The conditions assume that the size and velocity of settling of the particle are sufficiently great to render the viscous resistance of the liquid negligible compared with the turbulent resistance.

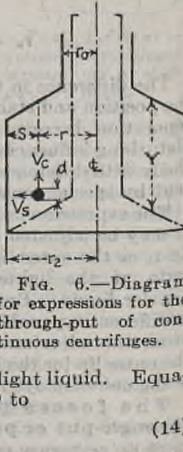


FIG. 6.—Diagram for expressions for the through-put of continuous centrifuges.

Thus we have:

$$F = R_l$$

$$\frac{(W_p - W_l)}{g} r\omega^2 = K\frac{\pi}{4}\delta_l d^2 V_s^2$$

$$\frac{\left(\delta_p \frac{\pi}{6} d^3 - \delta_l \frac{\pi}{6} d^3\right)}{g} r\omega^2 = K\frac{\pi}{4}\delta_l d^2 V_s^2$$

$$d(\delta_p - \delta_l) r\omega^2 = \frac{3}{2} K g \delta_l V_s^2$$

$$V_s = \sqrt{\frac{2}{3}\frac{d(\delta_p - \delta_l) r\omega^2}{K g \delta_l}} \qquad (13)$$

The coefficient K may be taken as 0.5.

Equation (13) and Fig. 6 show that clarification of large particles, encountering mainly turbulent resistance to settling, is easier for large particles of high density in a light liquid. Equation (13) may be condensed by the use of the symbol D to

$$V_s = D r^{1/2} \omega \qquad (14)$$

A particle will be removed from the stream of liquid if the velocity V_c, which is determined by the through-put rate, is sufficiently low for the settling velocity V_s to bring the particle through the settling distance S before the particle has been swept on out in passing through the travel distance Y.

The limit for settling out such a particle is that its settling time t_s through the settling distance S shall be just equal to, or less than, the time t_c required to move through the travel distance Y, or

$$T_s = \frac{S}{V_s} \leqq T_c = \frac{Y}{V_c}$$

From this, the limiting velocity V_c, which determines the maximum clarifying through-put, may be expressed as

$$V_c = \frac{Y}{S} V_s$$

This shows that the clarifying through-put capacity is directly proportional to the travel distance and inversely as the settling distance. The value of the settling velocity V_s is determined by the centrifugal forces, the excess of the particle density above that of the liquid and the **turbulent** resistance of the liquid to motion through it. The value for the average settling velocity may be obtained by integrating Eq. (14) and substituted above for V_s. This gives

$$V_c = \frac{Y}{S} \times \frac{2}{3} \frac{(r_2^{3/2} - r_0^{3/2})}{(r_2 - r_0)} \times \sqrt{\frac{2}{3} \frac{d\omega^2(\delta_p - \delta_l)}{kg\delta_l}} \qquad (15)$$

The capacity or volume of liquid passed through a bowl is the product of the through-put velocity and the section area for flow. In this simplified case

$$C = V_c \pi (r_2^2 - r_0^2)$$

Substituting from Eq. (15) and noting that $S = (r_2 - r_0)$ in this simplified case,

$$C = \frac{2}{3} \frac{Y\pi(r_2^2 - r_0^2)(r_2^{3/2} - r_0^{3/2})}{(r_2 - r_0)^2} \sqrt{\frac{2}{3} \frac{d\omega^2(\delta_p - \delta_l)}{kg\delta_l}} \qquad (16)$$

If $r_0/r_2 \doteq 0$ may be considered negligible compared to r_2, then the capacity for complete separation of particles of diameter equal to or greater than d is

$$C = \frac{2}{3} \pi Y r_2^{3/2} \omega \sqrt{\frac{2}{3} \frac{d(\delta_p - \delta_l)}{kg\delta_l}} \qquad (17)$$

Settling Rate for Small Particles—Viscous Resistance. The settling rate for small particles is determined by the viscous resistance of the liquid medium to the motion of particles settling through it. Stokes has shown that this resistance may be expressed in grams as follows:

$$f_z = \frac{3\pi z d V_s}{g}$$

where z is the absolute viscosity of the liquid medium in poises, and the other symbols have their previously given definitions.

The velocity of settling of such a small particle increases until the forces of viscous resistance equal the centrifugal settling force.

$$F = f_z$$

$$\frac{\pi}{6g} d^3(\delta_p - \delta_l) r\omega^2 = \frac{(3\pi z d V_s)}{g}$$

and from this

$$V_s = \frac{d^2}{18z}(\delta_p - \delta_l) r\omega^2 \qquad (18)$$

The clarifying through-put for these very small particles is here also determined by the limiting velocity V_c which would just permit settling out through the settling distance S before passing through the travel distance Y.

Therefore,

$$V_c \lesseqgtr \frac{Y}{S} V_s$$

Integration to get the average velocity of settling V_s, caused by the centrifugal force at different radial positions for the settling particle, gives

$$V_c = \frac{Y}{36sz} d^2(\delta_p - \delta_l)\left[\frac{(r_2^2 - r_0^2)}{(r_2 - r_0)}\right]\omega^2 \tag{19}$$

The capacity of the bowl, or volume passed through is as follows:

$$C = V_c \times \text{(section area for flow)} = V_c\pi(r_2^2 - r_0^2)$$

and since in this simple case $S = (r_2 - r_0)$

$$C = \frac{\pi Y d^2\omega^2(\delta_p - \delta_l)(r_2^2 - r_0^2)^2}{36z(r_3 - r_0)^2} \tag{20}$$

If $\frac{r_0}{r_2} \doteq 0$

$$C = \frac{\pi Y d^2\omega^2(\delta_p - \delta_l)r_2^2}{36z} \tag{21}$$

These equations are not directly applicable to conditions where the settling is controlled partly by turbulent and partly by viscous resistance. Actual capacities cannot exceed, but must be somewhat less than, the lower value of either Eq. (18) or (20). Both show that decrease of particle diameter makes complete separation increasingly difficult, particularly for separation of very small particles from a viscous medium.

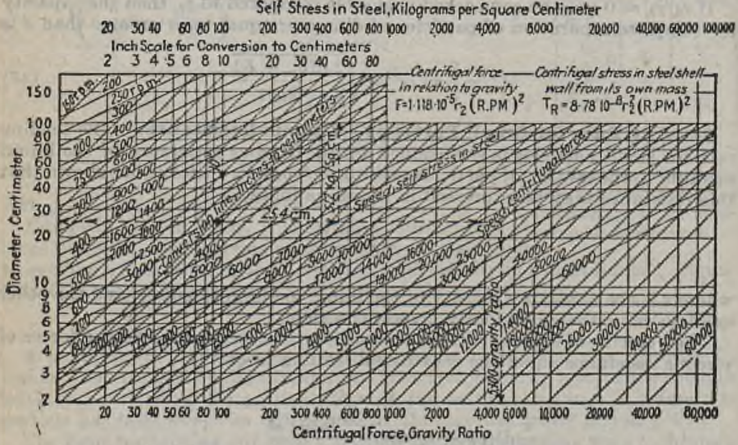


Fig. 7.—Centrifugal force-gravity ratio and self stresses in rotating rings.

Settling Rate for Microscopic Particles—Hindered Motion. When the particles become of such small sizes, relative to the size of the molecules of the liquid through which they must settle (or rise), or the temperatures so high, that Brownian movements appreciably disturb the settling, it may become practically impossible to effect separation without pretreatment for agglomeration of the particles or their adsorption on larger particles of some

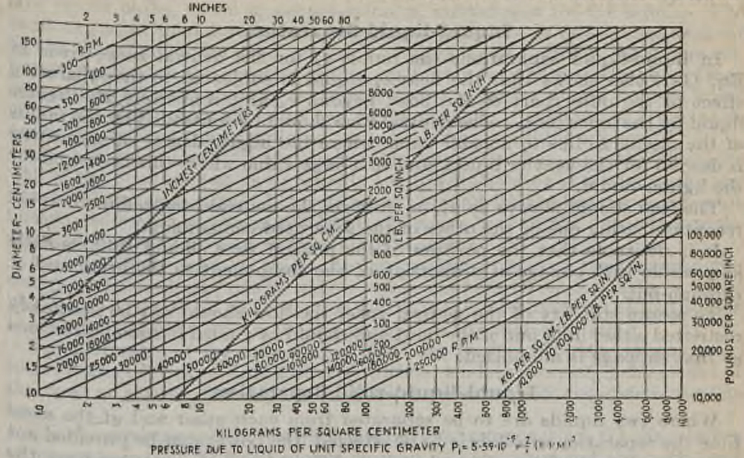


FIG. 8.—Centrifugally produced liquid pressure for water.

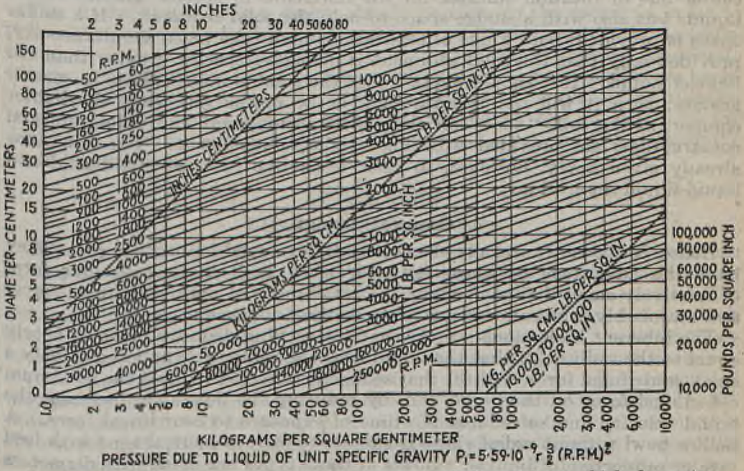


FIG. 9.—Centrifugally produced liquid stresses in shell wall 1 cm. thick, for water.

more readily settled material, or their transfer into another liquid which may carry them with it. Settling is hindered also by crowding and ceases at high concentrations.

Each case of this kind requires individual study by the aid of the microscope, ultramicroscope, and by laboratory tests on the mechanical, chemical, and physical factors of the colloidal conditions.

Liquid-liquid Separation

In liquid-liquid separations the radius r_n for the neutral zone, given in Eq. (12), also shows that the heavier liquid is subjected to the separating effect of the outer part of the bowl beyond r_n and conversely the lighter liquid by the inner part. The neutral zone should be controlled by the radius of the discharge lips to a *small* value where the higher degree of separation is desired for the heavier liquid, and to a *large* value where this is desired for the lighter liquid.

The feed should always be at, or close to, the neutral zone radius to avoid recontaminating one or the other of the separated components.

It is desirable that the position of the neutral zone should shift as little as possible with change of proportion of one component to the other and of through-put.

To secure stability of the neutral zone, the flow should not be needlessly restricted either in or out of the bowl and should be guided in radial directions so that slippage is minimized.

Liquid-liquid-solid Separation

Where two liquids are to be separated from each other and at the same time the separation of solids is to be effected, the bowl must be provided not only with two discharge lips placed at different radial distances from the center line of rotation suitable for the difference in specific gravity of the liquids but also with a sludge space to hold the solid sediment. If a sludge space is provided, both kinds of separations may take place simultaneously, provided only that the solid sediment is of higher specific gravity than the heavier liquid. If it should happen that the solid is intermediate in specific gravity, then it will not rise through, or be discharged with, the lighter component but will "float" on the heavier component and unless unusual constructions are provided will clog at the neutral zone. The equations already given apply, however, to their respective cases of liquid-solid and liquid-liquid separation.

General Design

Hollow bowls were naturally the first type developed for centrifuges. They have the advantage of simplicity and, when built in long tubular form of relatively small diameter, allow the attainment of high centrifugal forces and reasonably high ratios of travel to settling distance.

The inherent limitations are (1) that the settling distance must be nearly equal to the radius, (2) that the liquid near the center gets exposed to only a small centrifugal force, and (3) that sediment collected reduces the maximum centrifugal force on the liquid and, by reducing the flow area, increases the liquid velocity and so reduces the time of exposure to centrifugal force. A hollow bowl without radial vanes is subject to shift of neutral zone with feed rate or proportion of liquids. Bowls of these types are made with diameters up to 12 cm. and for speeds of 15,000 r.p.m.

Bowls with shallow settling spaces have been devised using a number of constructions such as concentric cylinders, spiral leaves, and cone-shaped disks. For settling large heavy particles out of much lighter liquids, bowls with generally one (though sometimes more) concentric cylindrical inner shell still find a considerable field of usefulness due to their simplicity of construction and, therefore, cleaning. The vast majority of separators use cone-shaped disks with thin spacer caulks to get very shallow settling spaces, long travel paths, and a moderate-velocity, guided flow for the liquid. The caulk or spacer thicknesses are selected so as to be two or more times as great as the diameter of the solid particles or globules of liquid to be separated. Larger values for the spacing caulks are better for those cases where large proportions of solids must be removed, and thin caulks should be used when only very small proportions of one component, *i.e.*, fractions of 1 per cent, are to be removed from the liquid and where the viscosities are low.

The cones are usually made appreciably smaller in diameter than the inner wall of the bowl shell so as to provide a sediment-holding space for such very heavy sediments as cannot be swept on out along with the heavier component liquid. Sediment may completely fill the space beyond the disk edges without altering the clarifying or separating capacity.

It is relatively easy to guide the flow radially out and back toward the center to the discharge lips by spacer strips or ribs between the disks in radial planes and so maintain a position for the balance circle dividing the two liquids that shifts but little with ratio of components, through-put, etc.

The half angle of the cone-shaped disk may be 30 to 50 deg., chosen so as to get a short radial settling distance, without unduly increasing the height of the bowl or getting the angle so small that sediment will not slide along the cone.

The disadvantages of bowls with shallow settling spaces are (1) the number of internal parts to be handled and cleaned, (2) the weight of the parts, and (3) the cost.

The *discharge lips* in any design should be close to the center in order to reduce the power, the frothing of the discharged liquids, and the air entrainment. A certain proportion of power, up to about 40 per cent of the kinetic energy of the discharge, may be conserved by directing the discharges backward (as in a reaction turbine). This also reduces the froth and air entrainment and, by reducing the air pressures induced by the jet action of the discharges, reduces entrainment by windage, of spray from one receiving cover into the other.

APPARATUS

Liquid-solid Separators

Dryers, so called, of the centrifugal type have found almost universal use in laundries and in dry-cleaning plants for the removal of the major part (down to about 20 per cent) of the water or dry-cleaning fluid from cloth. The remainder cannot be removed centrifugally and must be evaporated. Such dryers are made in a large variety of sizes from about 18-in. diameter "baskets" up to 60 in. The cloths are packed against the perforated walls and require about 5 to 10 min. centrifuging from start to stop. It is important that the cloths be laid smoothly and packed evenly from the inside wall to the width of the curb. If carelessly packed in lumpy wads the liquid removal is obstructed, and serious unbalance and shaking of the centrifuge, or even of the foundations, may occur. The holding volume per foot of vertical depth of basket is given by one of the curves of Fig. 18.

Cloth and similar materials normally carry 6 to 14 per cent moisture so that the removal of water is nearly complete, only a small proportion being left for removal by evaporation. Where dry-cleaning solvents such as petroleum spirits are to be removed, the centrifugal separation should be made as complete as possible, because the residues left by heating or by hot air blowing over the clothes may leave undesirable odors.

Basket centrifuges with overflow over the inner lip of the curb, or through the perforated walls of the basket, or through screens, find a considerable use in chemical plants for recovery of crystals precipitated from their mother liquor. The screening medium may be laid over a grid work or corrugated backing to allow a free flow-discharge path for the liquid after passing through the screen. Monel or other corrosion-resisting metal wires may also be woven in close mesh for screening out fine crystals. Such wire cloth may be "rolled" to flatten the wires and to reduce the mesh opening. The centrifugal dryer is desirable where a very complete removal of mother liquor is desirable and where washing of the crystals with water or other liquids is to be resorted to for further purification. For very coarse non-porous particles, 5 to 10 mm. minimum dimension in any direction, only a fraction of 1 per cent by weight of a free-flowing liquid may be left after centrifuging. This is representative of what may be done in removing cutting oil from machine tool chips even with flat or curled long-strip chips. Finer particles necessarily carry more liquid with them in proportion to the surface-volume relation which increases linearly with decrease of dimension of particles as shown below for the calculated values for spheres, whether assumed stacked with centers in line with each other, or nested, with the centers of the spheres of alternate layers displaced a distance equal to the radius of a sphere.

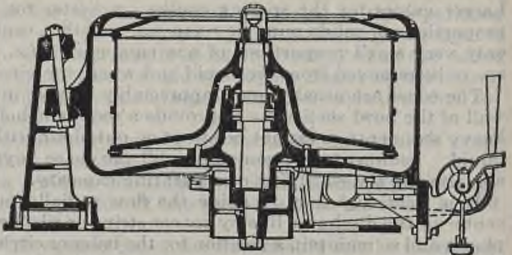


FIG. 10.—Center slung centrifugal.

High viscosity liquids may give thicker films and retain more liquid, unless the time or centrifugal force is increased. Fine, slimy precipitates may retain 70 per cent or more liquid, even when centrifuged in the smaller diameter, high-speed baskets which give the greatest centrifugal effect for a given stress on the wall material.

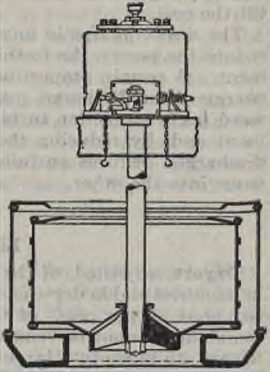


FIG. 11.—Suspended centrifugal.

Baskets up to 60 in. in diameter, at speeds up to 500 r.p.m., are used to get the advantage of large load capacity and reduce the time and cost of loading and labor for operating. Sprays for washing the material in the basket are frequently employed with the spray nozzles set in positions to discharge against the inner wall of the "cake."

It is usually unsafe to feed a slurry to the bowl while running, unless a bowl is supported from below or a very sturdy spindle is employed and a

Table 1. Surface-volume Relations

Diameter of sphere, cm.	Number of spheres per cc. centers in line	Number of spheres per cc. nested fully	Volume of spheres per cc. centers in line	Volume of spheres per cc. nested fully	Surface area per sphere	Surface area per cc. of spheres with centers in line	Surface area per cc. of spheres nested fully
1.0	1.	1.41	0.5236	0.740	3.14	3.14	4.45
0.1	10^3	1.41×10^3	0.5236	0.740	3.14×10^{-2}	3.14×10	4.45×10
0.01	10^6	1.41×10^6	0.5236	0.740	3.14×10^{-4}	3.14×10^2	4.45×10^2
0.001	10^9	1.41×10^9	0.5236	0.740	3.14×10^{-6}	3.14×10^3	4.45×10^3
0.000,1	10^{12}	1.41×10^{12}	0.5236	0.740	3.14×10^{-8}	3.14×10^4	4.45×10^4
0.000,01	10^{15}	1.41×10^{15}	0.5236	0.740	3.14×10^{-10}	3.14×10^5	4.45×10^5
0.000,001	10^{18}	1.41×10^{18}	0.5236	0.740	3.14×10^{-12}	3.14×10^6	4.45×10^6

spring mounting for the stationary frame provided, such as will allow a considerable unbalance of the bowl, without undue strains on the foundation. Loading and unloading are usually done with the bowl standing still. Bottom plates are often provided which can be lifted or tilted out of the way so that the load may drop or be pushed down and out of the bowl into chutes or boxes on wheels.

Mechanical unloaders, having knives or scoops which are swung into place and pick up the solids while the bowl is still *slowly* moving, can be used with some materials. If health or other risks exist, these offer advantages, but some materials cannot be so handled. The scrapers may leave too much of the solids on the walls, and, for hard or tightly packed solids, it is often difficult to get sufficient rigidity in the scraper or in the bowl support to allow the use of scrapers without "digging in" or jamming, particularly if the bowl has not been slowed down sufficiently or the operator is careless. This is also a difficulty with sticky or hard slimy deposits.

Horizontally mounted basket centrifuges have recently been developed with two bearing supports of great rigidity, which allow automatic unloading while at full speed by scraper knives. These may have fully automatic controls that regulate the feed, rinse, drying time with feed shutoff, and unloading.

Centrifuges for continuous discharge of dried solids as well as for the discharge of the separated liquid, while running at full speed, have been built and will operate for a few easily handled materials, such as the dewatering of washed coal or coal reclaimed from river beds by hydraulic pump-suction dredges, and of some mine slurries.

The solid particles must be fairly large and must be rather hard and of the type that roll or slide easily, without sticking or packing, such as hard coal, stone, sand, etc. Furthermore, the liquid must be quite fluid, with little tendency to adhere to the surfaces of the solids.

A vast amount of time and money has been spent in an attempt to build successful centrifuges with continuous discharge at full speed of sludges, slimes, sticky materials such as sugar crystals, sewage residues, and slurry from mining operations. Attempts to scrape off such materials from the walls or surfaces upon which they have been deposited or to employ some kind of a conveyor mechanism have met with limited success because of the great frictional forces that must be overcome to slide the materials along after they have once been deposited, thus allowing the full centrifugal force—many times that of gravity—to press them against the receiving surfaces. Also, most of these materials tend to gum, pack, or jam ahead of the scrapers.

Finally, there is the difficulty of balancing the slime and in balancing and in driving the conveyor parts whose several elements are also subjected to centrifugal forces, holding them together with great force so that the frictional resistances to movement are tremendously greater than when the bowl is standing still.

The type of construction using two concentric cones run at slightly different rotational speeds by means of a differential positive gear drive has had a certain amount of success. This type feeds a granular material that does not stick together or stick to the perforated walls of the outer cone and dumps the solids at the lower end (large-diameter part) of the cones, while the liquid is discharged radially through the perforations of the outer cone. The materials must be fed in at the smaller cone end and discharged at the larger end, which is usually at the bottom, employing a vertical shaft machine. Spiral scrapers are rigidly mounted on the inner cone and nearly touch the inner-surface walls of the outer cone from which the granular solids are to be scraped, and the direction of the spiral and the differential of speed must be so chosen as to cause the solids to move to the larger, or discharge, end of the cones. If the axial cone angle is large, centrifugal forces will help roll the solids along, and the scraper spiral may serve to hold the particles back, until dewatered, rather than to push them ahead.

Attempts to dewater sewage slimes, remove molasses from sugar, reclaim yeast from wort, and similar separations by mechanically operated scrapers for solid removal have had a limited success. The principal recent improvement in these types of machines has been the development of differential gear drives that would maintain the differential speed of the scrapers by reaction of gear trains between the bowl and scraper running at the bowl speed. When the torque between the two full-speed elements, required to maintain the small differential speed, is supplied to each part through a train of gears, each train is subjected to the scraping forces *multiplied* by the speed-reduction ratio.

Hollow-bowl clarifiers, having a low centrifugal force with no interior parts except the three to four wings which force the liquid to rotate with the bowl and the feed passages to bring the incoming materials to the outer parts of the bowl, serve sufficiently well for easily removed solids. A useful and simple improvement consists of employing an inner cylindrical shell, with a diameter about 70 per cent of that of the outer shell, so as to give equal holding volumes and through-put velocities in the two shells. The incoming material is fed first to the inner shell, preferably through a feed tube with a side or radially directed outlet so that each side or quarter of the bowl is fed, in turn, equal amounts, so as to ensure equal distribution, maximum purification, and uniform loading of sediment. The inner shell picks up the coarser and more readily separated sediment and then allows the liquid to pass to the outer shell for the removal of the finer sediment. Such bowls serve to remove coarse, unground pigment particles in paints, varnishes, and lacquers. The degree of clarification of pigments, etc., may be controlled either by change of through-put or by adjusting the bowl to a lower speed, so as to avoid throwing out too much of the pigment or requiring too frequent stoppage for bowl cleaning.

Hollow high-speed tubular bowls with high centrifugal forces over 10,000 g. may be used for difficult separations such as clear varnish and other cases of semicolloidal particles.

Disk-bowl clarifiers have usually cone-shaped separating disks of somewhat smaller diameter than the bowl shell so that sediment may collect without packing between, and thus impairing, the clarifying effectiveness of the disks. The feed channels must bring the incoming materials near to the outer edges of the disks and evenly distribute them to and around them.

Disk bowls, may be used for difficult clarification problems such as clear varnish and the removal of semicolloidal particles. They are also used for clarifying whole milk to remove particles of dirt picked up during milking or in handling and transhipping the milk to the milk plant. Clarification also removes such undesirable materials as scarf-skin particles from the udder of the cow and occasional pus or blood cells. All of these materials are collected as slime in the sediment space, out of the main path of the milk, and so successive contamination of further quantities of milk, which must necessarily occur with filters, is avoided.

Concentrator bowls with nozzle discharge of the solids afford a means of recovering the solids along with a much reduced proportion of the liquid. The nozzle passages lead from the outer part of the bowl, where the solids tend to collect but are swept out along with the portion of the liquid which discharges through the nozzles. The inner surfaces of the bowl shell are usually shaped somewhat like a beehive in order to provide a slope to the nozzle-passage entrances for the purpose of facilitating the sliding along of the solids. The nozzles sometimes discharge directly at the bowl diameter. However, as this may require very small diameter nozzles, which easily clog, and consume much power, the passages often lead to nozzles placed near the center of rotation, and the nozzles may be placed to discharge nearly tangentially, and backward, to recover energy, reduce frothing, and reduce spatter.

Bowls for yeast are built in sizes up to 440-mm. diameter and are usually provided with cone-shaped separating disks. They are generally run at lower speeds than similar separators.

The degree of concentration, or ratio of the (constant) volume discharged by the nozzles to the total volume fed to the bowl, may be regulated by decreasing the nozzle opening, provided, however, that the quantity of solids in the feed liquor and the ratio of solids in the concentrate are not so high as to decrease unduly the fluidity of the concentrate.

The solids in the concentrate may be raised to 50 or 65 per cent without too great flow resistance. The percentage concentration is, therefore, further limited by the ratio of solids in the feed, since all the solids appear in the concentrate. For instance, if 50 per cent solids is taken as a reasonable limit for free flow, a feed containing 5 per cent solids may be concentrated into 10 per cent of the feed, but if the feed contains 10 per cent solids it would not be possible to concentrate to less than 20 per cent of the feed volume.

Such types of concentrators are almost universally used to concentrate yeast cells into a small portion of the wort in which they were grown, recovering the yeast.

Valve bowls, are provided with nozzles or orifices closed by valves mounted on the inside wall. These valves may be opened automatically by means of springs, actuated by a lowered speed, or by liquid-filled floats, which empty directly through the nozzle when liquid feed to the inner end is decreased by accumulation of sludge. Instead of separate valves and nozzles, the whole bowl may be opened by mechanical devices, such as levers, or by feeding a control liquid to a pocket mounted on the bowl and letting the centrifugal force on the liquid in the pocket open the bowl at its periphery. The separate valves may also be opened by a rod concentric with the shaft, connected by appropriate rods and levers to the valve element. If the valve or the whole bowl is opened while running, a considerable amount of energy is dissipated in the discharge, and such machines require many times as much power as the usual types. Provision must also be made for replaceable wearing surfaces in the covers to take the impact of the jets and for steep sliding slopes or mechanical conveyors for removal of the sludges.

Development of various kinds of centrifugal apparatus for the continuous discharge of solids has been particularly active in recent years.

Liquid-liquid Separators

Concentrators, or ratio separators, are built with two liquid discharges at different radii suitable for the usually small difference in effective specific gravities, and with non-radially guided flow of either or both components in their respective paths toward the center and toward the outer wall of the bowl, so that a certain but controllable proportion of the feed will pass out by each of the two discharge paths. Practically all such concentrators are now built with cone-shaped separating disks and find their greatest field of application as "cream separators," in which field centrifugal separators have superseded all other methods throughout the world, a million or more centrifugal cream-separator machines being produced annually, which is perhaps one hundred or more times the number produced for all other purposes combined. The reasons are not only that the centrifugal separator can produce a cream with a butterfat content of any desired value from about 15 to 60 per cent (or even 80 per cent if adapted for the purpose) with a negligible loss in the skimmed milk (whereas allowing cream to rise by gravity on standing will seldom produce more than about 20 per cent butterfat cream and incur losses of ¼ per cent or more), but will also do so instantly, without the delay and consequent souring that may occur on "standing" cream.

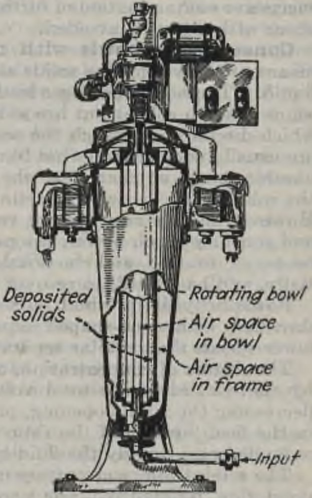


FIG. 12.—Tubular-bowl high-speed centrifuge.

Cream is an emulsion of butter oil in milk and the "cream" is merely a higher concentration of this emulsion than that in fresh whole milk. Whole milk may contain from 3 to 6 per cent of butterfat; "ordinary cream," as obtained for table use by pouring off the cream that rises on standing may have 18 per cent (or less) butterfat. "Heavy cream" is produced by centrifugal separation with 40 per cent fat, which is cut back with milk for "light cream." The name **separator** is firmly established by usage although the term "concentrator" much more truly describes the function performed, of concentrating **all** of the emulsion, *i.e.*, the cream in a part of the milk. The cream separator or concentrator is so built that if fed a homogeneous liquid, such as water, a portion will be discharged through **each** of the two spouts. The proportion of the total passing through the cream discharge may be regulated by making the cream discharge through the hole in a hollow screw, set in a radial direction, so that by screwing it toward the center the inner end picks up the cream at a point nearer the center where there is a lessened centrifugal pumping head, and therefore a smaller quantity discharged, resulting in a "richer" cream, *i.e.*, carrying less milk. This method of control is quite widely used on small separators. The control may also be put on the outer discharge, either by narrowing the opening through which the skimmed milk passes, or by arranging one or more screws so that they can be set either to allow or to stop slippage. Slippage of the heavier component in its progress toward the center to reach its discharge opening allows the liquid to retain some of the peripheral velocity it had at the larger diameter and so "run ahead" of the inner parts of the bowl thereby building up a centrifugal back pressure which tends to limit the flow in this path. Thus adjustable or permanent radial baffles or ribs in the flow to the discharge lips, by reducing slippage, increase rather than decrease flow.

The distribution holes in the disks are placed near the center so as to apply the major separating effect to the skimmed milk which constitutes the heavier outer component.

Concentrator-type bowls are quite suitable for constant through-put concentration of a liquid with a practically constant ratio of a lighter component or of an emulsion into a portion of the heavier liquid. They are not suited to separations where the proportion of one component, or the feed rate, varies during a run.

True separator bowls are provided with two discharge diameters, an inner and outer, with radial guide ribs throughout, between the cone-shaped separating disks or in the body of the hollow bowl and also in the return path of the heavier component past the top-disk skirt back to its discharge lip. It should be possible to vary the proportion of one component from 0 to 100 per cent of the total at any through-put rate without break-over of either component into the discharge passage intended for the other. The adjustment of the balance circle and the setting for different ratios of specific gravities are usually accomplished by changing the lip for the outer discharge. A number of rings or disks with center holes of different diameters are provided which may be readily installed by unscrewing a nut which grips the outer rim of the discharge ring firmly against a rubber ring or other gasket, to prevent leakage. Both small-diameter tubular bowls at high rotational speeds and cone-shaped disk bowls of larger diameter at lower speeds are used for separators; each has certain inherent characteristics, merits, and limitations, which have been discussed under the heading, General Design (p. 1816).

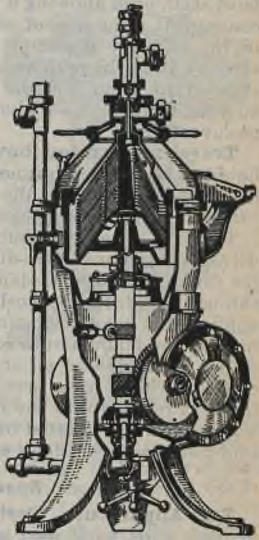


FIG. 13.—Closed feed and discharge disk bowl centrifuge.

Liquid-liquid-solid Separators

Concentrating separators with nozzles discharge the solids along with a portion of the heavier of two liquids being simultaneously separated from a lighter liquid. The nozzles are fed by passages from the outer part of the bowl. Two liquid-discharge lips and means to adjust the discharge diameter of one or the other to suit the specific gravity difference of the two liquids are provided. The heavier liquid component must be present in the feed, in proportions greater than that required for the nozzles so that an excess remains to fill up to the outer lip. This may be provided either by recirculating a portion of the heavier liquid or by its presence in excess in the feed. Commercial applications of this type of separation appear to be increasing.

Separator bowls with sludge pockets or sludge-holding spaces require a little modification of the liquid-liquid separators, namely, that of making the cone-shaped separating disks appreciably smaller in diameter than the bowl shell, or in allowing a portion of the hollow bowl to be filled up, with the consequent reduction of separating effect due to the progressive reduction of the effective diameter as filling occurs and the increased through-put velocity V_c as the path area for liquid flow is constricted. The bowl must be stopped to clean out the sludge, consequently this method is peculiarly adapted to those cases where only relatively small amounts of solids must be removed.

Traveling-cushion bowls have fed to them a liquid heavier than the liquid to be clarified but near enough to that of the solids or emulsions which are to be swept out of the bowl to cause their movement through the bowl rather than pocketing them in the sludge spaces next to the bowl-shell wall.

This is an attempt to substitute a sweeping-out liquid for the mechanically driven continuous slime-discharge bowls. The difficulties of proportioning the densities and of keeping the slimes from either floating upon or sinking through the traveling cushion of liquid are very great so that commercial applications have been rare. An exception is the method that was widely used in the manufacture of bright stock, where a carrier liquid, usually hot water, immiscible in oil or wax, was injected into the wax space just ahead of the discharge ring, serving the double purpose of a traveling cushion, as far as it reached into the bowl, and of melting the chilled wax. Melting of wax was further assisted by spraying hot water on the bowl top to be thrown into the wax cover along with the wax.

Special Types of Centrifuges

Test Apparatus. Test-tube centrifuges with heads carrying two, four, or more cups, so shaped as to hold graduated test tubes containing two or more liquids, or liquids and sediments, are employed to effect a separation or sedimentation and to give easily read volumetric proportions of the different components.

A wide variety of forms and shapes of test tubes are in use (with corresponding shapes for the holding cups), such as cylindrical glass bottles, *i.e.*, the 4-oz. oil sample bottle, small and large laboratory test tubes, pear-shaped vessels with a small cylindrical extension for reading small proportions, and cylindrical vessels with a cone-shaped lower end to give readable values for very small volumes of sediment or separated liquids.

The standard speed prescribed by the latest revision of A.S.T.M. standard method of test, Designation D 96 for water and sediment, is 1500 r.p.m. with 100-cc. test tubes supported in cups so that the tips swing in a 15- to 17-in. circle. This gives a centrifugal force at the extreme tip of the tube of 500 to 550 times gravity, and less for the other parts of the test tube. These forces are not sufficient to effect some separations or to break some emulsions; therefore, materials that tend to break emulsions and diluents, solvents, or precipitants are often added (see the latest revision of A.S.T.M. Method for Precipitation Number of Lubricating Oils, Designation D 91).

Specially designed and sturdily proportioned machines can be built for speeds up to 6000 r.p.m., using the standard A.S.T.M. 100-cc. cone-shaped test tube.

The test-tube centrifuge gives a means of observing the relative ease of centrifugal separation and a means of measuring the performance of commercial separators, for separation of water from insulating oils, etc., and for clarification of sediments.

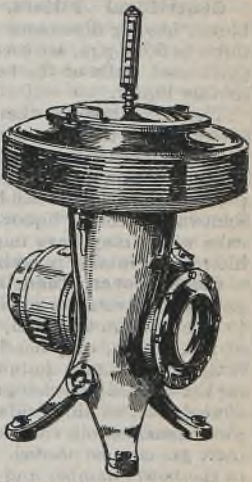


Fig. 14.—Test-tube centrifuge.

It must be observed that, as it is the *volume* which is measured, it becomes necessary to consider the extent to which sediments are "compacted" by long-continued centrifuging. It is quite possible to observe, with repeated centrifuging of a single given sample, a quick accumulation of sediment, then no further increment for several repeated centrifuging periods, followed by the accumulation of quite a large volume of sediment (presumably much more difficult to throw down than the first sediment observed) and, finally, the compacting of this last sediment into a smaller volume.

The final volume is, however, larger than the value for the previously observed quantity of quickly settled, coarse or heavy, easily precipitated sediment. When testing an unknown material, the centrifuging should be prolonged beyond the time for the first observed, "apparently" constant volume, unless complete clarification of the liquid, or other observations, confirm that all sediment has been removed from the sample and the sediment compacted down to its final volume. The final compacted volume is generally some function of the speed and time of centrifuging. Sediments differ markedly in compacting, and speed and time may alter the volume by 2 to 1 or more.

In the precipitation test, applied to lubricating oils, the settling of the precipitate is quickly accomplished in the centrifuge test tube and the **volume** of the precipitate read as prescribed by A.S.T.M. Method D 91. In addition the solvent and oil may be decanted and the precipitate collected, dissolved, and **weighed,** which obviates difficulties and inaccuracies in the calibration of the smaller volumes of the test-tube cone-shaped tip and of the compacting of the precipitate, as well as eliminating any possible effect from solids, etc., accidentally already present in the oil sample and therefore not actually precipitated by the precipitation naphtha.

The Babcock butterfat-milk tester is a special form of test-tube centrifuge in which the cylindrical cups are constructed to hold a special form of test bottle with a small diameter neck attached to the upper part of the test bottle and a filling tube passing through the shoulder alongside the neck and leading to the bottom of the test bottle.

This small neck catches the butter oil rising to the surface after sulfuric acid has destroyed the cream-globule film and allows reading values from 0.01 to 0.25 per cent. This method is still almost universally employed throughout the Americas, although various methods of extracting the butter oil by ether or other solvents, then evaporating the solvent and weighing the residue, are coming into use here and are almost universally used abroad, since they reveal small residues of butterfat, not measurable by the Babcock method. The Babcock centrifuge head may hold four or more, up to 32 test bottles. For the 16-in. diameter swing circle, a speed of about 850 r.p.m. is employed.

Centrifugal Filters. Basket-type centrifuges, with large-diameter bowls, having diameters of 18 in. up to 60 in. and speeds of 1500 r.p.m. down to 500 r.p.m. are used with screens or filtering mediums laid on the inside perforated walls of the bowl shell to hold precipitates and to throw off the mother liquor.

The advantage of the centrifugal filter over plate-and-frame and other filters lies in the very complete removal of mother liquor from the precipitate which centrifuging gives, the ease and completeness of washing the precipitate obtainable by spraying water or other wash liquids on the bed of precipitate while it is still held in the running bowl, and the final quite complete removal of mother liquor and wash water. This gives a quite clean "dry" cake which may have important chemical advantages over plate-and-frame filters, and rotary, pressure, and vacuum filters for certain processes.

Closed-cover, Feed, and Discharge Machines. Gas-tight, or nearly gas-tight, covers are provided where spray or fog from the discharges needs to be kept from the room, or where vapor losses must be prevented on account of fire, health, or other hazards. The joints in the covers may be snugly fitted or even made quite tight by gaskets. Flexible metal tubes are provided for the feed and discharge pipes. Inert gases may be fed to the inside of the covers; but, as all liquids tend to dissolve more or less gas, as well as carry away considerable volumes of gas entrapped as froth, large quantities of the inert gas may be needed. Special designs make it possible to hold a vacuum in the bowl chamber and in the covers which receive the discharge.

Liquid-sealed Discharges. It is possible to discharge one or both the liquids from a centrifuge bowl by the equivalent of a small centrifugal pump in which the housing rotates with the bowl and what would be the impeller is held against rotation, so that the liquid under the centrifugal pressure of the "pump" housing receives the liquid at the housing diameter and flows *toward* the center and out through the hollow center of the "shaft" or through a concentric tube surrounding it. Clarification of a single liquid is readily carried out with a single closed discharge. Separation of two liquids with both discharges closed may be effected wherever the two components have a constant proportion to each other, as in cream separators.

There is, however, the drawback that enclosure is not entirely complete, owing to running clearances, and some air froth is still produced. In skim milk this may be several per cent by volume, but it is much less than the 30 to 40 per cent produced by open-discharge separators. A considerable amount of power is consumed in liquid outlets of this type.

Bearing-type sealed passages may be provided by mounting the discharge chamber on the bowl neck as if it were a bearing, but flexibly supporting the chamber on a corrugated metal bellows or on a flexible diaphragm to allow it to adjust itself easily to the position of the high-speed rotating bowl. This construction is feasible for the clarification of liquids that will provide a lubricating effect, suitable for the light side pressure on the bearing-type seal and its high speed and large diameter.

Clarifiers with both inlet and outlet passages sealed by bearing-type connections are usually made with the bowl mounted on a hollow shaft with a bearing at each end so as to allow the feed to come in at one end of the hollow-shaft end and pass out through the other. The bearings may be combined with the feed or discharge passages or may be separate therefrom. If an oil or other material that will lubricate the bearings is to be clarified, it is much simpler, less costly, and more economical of power to combine the bearings with the inlet and outlet passages. For certain conditions of service, where the clarifier is subjected to severe mechanical shocks and vibrations, the double-bearing support offers material advantages over the more common type in which the bowl is mounted on the end of the spindle beyond its bearing support. The bearing is slipped, cap fashion, over the end of the hollow shaft and the liquid connection is made through a flexible hose screwed on the end of the bearing. The bearing is supported on a spring cushion to allow the bowl to rotate freely about its center of gravity, without undue side pressure on the bearing. The leakage of liquid past the inner end of the bearing along the shaft is controlled by reverse-helical pumping grooves. At the speeds prevailing in centrifugal clarifiers of 6000 to 9000 r.p.m., such helical pumping grooves will prevent leakage with internal pressures on the liquid of about 2 to 3 atmospheres above atmospheric pressure.

Clarifiers of this type are suitable for use on locomotives and in marine service on Diesel engines. Little or no water is likely to be present in the lubricating oil in Diesel engines, or, if present, would be held with the sludge. The clarification removes worn-metal particles, dust from the air, and carbon and sludge produced by oxidation, polymerization, or cracking of the oil.

Hermetic Separators and Clarifiers. Recently, completely closed-feed and closed-discharge cream separators and milk or other clarifiers have been developed having a hollow spindle feed to a bowl bolted tightly to the spindle and with closed-feed and discharge gaskets made of elastic compounds, shaped like the cup seals of hydraulic presses.

ACCESSORY APPARATUS

The pipes (see pp. 803–842), used for the feed and discharge connections, must be ample in size, not only because of the power lost in pumping through them against undue friction, but also because turbulence may cause emulsification and thus greatly hamper the separations.

Pumps (see pp. 2244–2260) for handling liquids to be centrifuged should preferably be of the positive displacement rotary type and run at moderate speeds. Centrifugal pumps are usually undesirable, not only because of the variation of delivery with back pressure, but also because of the violent agitation and tendency to produce emulsification, which they cause. Where, however, a mixing of two liquids is needed to promote a rapid chemical reaction, or a washing process prior to centrifugal separation, a centrifugal-type pump may advantageously be used to accomplish, or assist in accomplishing, the mixing. Piston pumps have the disadvantage of intermittent or varying rates of flow with each stroke (unless built with several cylinders), as a pulsating flow decreases the separating ability of the centrifugal bowl.

Regulation of pump delivery of constant-displacement pumps is best accomplished by gate-type valves, or still better by cocks, on the **suction side** of the pumps, since gate valves and cocks give a more definite and reproducible area of opening for a given setting of the handle than can be obtained with globe valves. The control of liquid fed to the pump by restricting the inlet valve opening, thereby producing pressure drop at the valve and vacuum on the pump, is usually better than choking the outlet valve and recirculating through a by-pass relief valve. The latter method builds up back pressure on the pump and increases the power needed to drive it. Also a recirculation of the liquid is not always desirable as it may promote emulsification.

The pump displacement for removing the centrifuged liquids from free discharge types of bowls should be from 25 to 50 per cent in excess of the liquid volume to allow for froth, air, etc., carried along with the centrifuge discharge.

Presettling tanks may perform an important function in settling out large amounts of coarse or easily separated materials prior to centrifuging, leaving only the smaller quantities of the more difficult part of the separation to be effected by the centrifuge. If water in large quantities is to be separated from an oil, a decanting tank may be used with a partition wall reaching to a point near the bottom, but leaving a passage under the baffle for water to pass into a water compartment from which it may flow out over a weir, slightly lower in height than the weir for the oil overflow on the other side of the partition wall or baffle. The residues of moisture left by a centrifugal separator in a purified oil are smaller when a smaller ratio of water to oil is present in the feed. In all cases the centrifugal separator should be relieved of coarse work and employed for the finer or more difficult separations.

Heaters or coolers (see pp. 948–1001) are frequently necessary to give a suitable fluidity or more suitable surface tension to the liquids to be processed, or to gain a differential in specific gravities. It is important that the heater mass and temperatures be kept low to avoid overrun in controlling (or overheating on shutting off) liquid flow, and that ample heat-transfer surfaces be provided in order to lower the maximum surface temperatures. **Radiant electric heaters** are undesirable because the heater units must be brought to a high temperature (*i.e.*, a red heat or higher) to get adequate radiating effects and upon shutting off the liquid flow, even with simultaneous disconnection of the electric supply, the temperature equilibrium reached involves excessive temperatures in the liquid.

Strip resistance heaters (see p. 2699) are more suitable, either immersed in the liquid or attached to the walls of the vessels or pipes. Steam or hot water taken directly from a boiler or from an auxiliary, electrically heated vessel and passed through heating coils, double-pipe heaters, or jackets offers a degree of safety from excessive temperatures not obtainable even with electric resistance heaters.

The rating of the heater required depends directly upon the liquid through-put, the temperature rise desired, and the specific heat of the liquid, so that for a through-put of 1000 gal. per hr. each degree Fahrenheit rise requires a net heat input of 2440 watts for water and about 1220 watts for oil. For 1000 l. per hr., 1160 watts is needed for each degree rise centigrade for water, and about 580 watts for oil.

The unit heat-transfer rate from pipe walls may be 5 to 6 watts per sq. cm. for heating water, but should be only 1.5 to 2 watts per sq. cm. for oil, to avoid overheating the oil film next to the wall. In any case, the average flow velocities through the pipes should be about 200 ft. per min. (1 m. per sec.).

Rates of heat transfer from the surface of the pipe wall (in contact with oil) to oil may be 15 to 20 B.t.u. per sq. ft. per hr. per °F. difference. For hot water to oil with coil or jacket heating, the coefficient of heat transfer may be 30 to 35, postulated, however, not on wall temperatures but on the temperatures of water and oil, and steam will transmit to oil in pipe coils about 60 to 70 B.t.u. per sq. ft. per hr. per °F., while steam in pipe coils heating oil in tanks will transmit only about 50.

Air-separating traps are often desirable in the centrifugal discharge connections to relieve the discharge pipes and pumps of air and so increase their carrying capacity. No fixed rules can be laid down, as the section area for air relief depends not only on the quantity of discharge and the ratio of air froth to be eliminated but upon the viscosity and, therefore, the temperature of the liquid.

As illustrations it may be said that for a flow of 20 gal. per min. a dry cleaner's naphtha may need only an 8-in. diameter vessel, while electric insulating oil may need a 30-in. diameter vessel.

Feed regulators (see pp. 2286–2292) are generally of the float type and operate either a balanced valve of the butterfly or of the piston type to reduce or even to stop the flow from the feed-pipe line as the float reaches a predetermined level in a supply cup or tank.

One point of advantage of such a method is that the rate of feed automatically decreases with increase of viscosity, which may therefore compensate to some extent for the increased difficulty of separation in the bowl on account of the greater viscosity.

When the supply is taken from a low head tank, the float may be made to control the rate of feed quite adequately by the simple expedient of allowing the top of the float to rise against the open end of the faucet or the end of the supply pipe.

A very satisfactory method of feed regulation is the employment of positive displacement pumps whose speed can be set at any desired value to give a corresponding through-put to the centrifugal. This avoids all probability of emulsification and allows the scheduling of runs as well as standardizing the performance of the centrifuge.

PROCESSING AND PREPARATION

Settling (and Settling Combined with Decanting). This is a simple and often useful process for the preparation of materials to be centrifuged. If a heavier liquid, such as water, is present in large quantities with oil, a settling process may be very advantageous in reducing the load on the centrifugal and also in improving its performance. A float carrying an intake port leading into a flexible hose or into a pipe with flexible couplings, so that only that part is drawn off which is at the top surface and therefore most completely clarified, is better than multiple draw-off cocks set at different levels.

A sight glass in the pipe line should be provided so that the character and amount of liquid being drawn off may be observed.

Where settling of hot liquids is to be accomplished, convection currents should be avoided. Heating, or even maintenance of temperature, by coils, promotes convection currents and is therefore less suitable than jacket heating. All surfaces should be well heat insulated, as the ideal condition for settling would be the maintenance of an absolutely unchanged temperature throughout the settling tank.

Where sediments or slimes are to be settled, the settling tanks should either have flat or plane-slanted bottoms to allow the use of mechanical scrapers, or should be cone-bottom tanks with cone slopes **greater** than the angle of repose of the sediment, so that opening the draw-off valve will give complete sediment drainage. Draw-offs should be closed by cocks set close to the cone of the tank or by gate valves, so that clots may be readily cleared by push rods.

Coagulating may be a very useful or even an essential step prior to centrifuging, to enhance the possible through-put rate and to improve the separation. If the particles also coalesce, the gain in differential densities usually more than counterbalances the slightly smaller dimensions of the new size of the coalesced particle. Whether coalesced or merely agglomerated, the settling rate and centrifugal separation rate are increased. In many cases, heating alone will coagulate. In others, washing or cooking, or digesting under pressure, with water or other liquids will bring about coagulation. The addition of some kinds of chemicals that do not injuriously affect the materials may be employed to reduce or to neutralize the electrical charge on the dispersed colloidal particles, thus allowing them to unite. Coagulation may also be carried out so as to form a kind of mat which, in settling, entangles and carries down with it all other particles. Disturbances such as convection currents must be avoided.

Partial centrifuging, makes possible the separation of three liquid mixtures by carrying out first, a partial separation which removes one liquid from the mixture and, second, by recentrifuging the mixture to separate the two remaining constituents. Generally it is somewhat easier to carry out the first step so as to discharge the mixture at the inner lip and one, only, at the outer lip. The proportion and gravity differentials of each of the three constituents determine the methods that may be used, for if two are closely alike in gravity they may be made to discharge together more readily at one lip, and if one of the three is present in quite large proportions it may be made to sweep along with it small portions of a second constituent that would otherwise lag and clog.

Partial centrifuging may also be applied by centrifuging, first, at a very high through-put rate for a partial separation or clarification, followed by a rerun at a low rate for a high degree of centrifugal purification. Another method is to pass the material through a large-diameter basket-type centrifugal for separation of large proportions of the more readily removed materials, followed by processing in a small high-speed centrifugal for a final purification.

Adsorbent Treatments (see pp. 1269 to 1336). Adsorbent treatments with carbon or other adsorbents may be used to pick up colloidal particles which may then be removed by settling, filtering, or centrifuging. If percolating beds are used, relatively large-mesh particles must be used (usually not finer than 30 to 60 mesh) to get reasonably high-liquid flow rates through them, thereby entailing a lower surface/volume relation and lower effectiveness of the adsorbent but somewhat smaller losses of liquid. If stirred in, "fines" may be used, which is more effective, but which usually entails some loss of the liquid, which either is thrown away with the spent adsorbent or, if extracted, is of low grade. Adsorbents may remove antioxidants and corrosion inhibitors from oils, and this effect must be borne in mind if they are used in oil treatment. A wide variety of natural, processed, and tradenamed materials are used for such purposes, such as diatomaceous and siliceous earths, fuller's earth, bentonite, silica gel, adsorbent carbons, Sil-o-Cel, Filtrol, Darco, etc. (see p. 1277).

Washing and Chemical Treatments. These are often important and sometimes essential processing aids prior to centrifuging. Washing may be partly a purely mechanical action for removal of impurities but often has chemical or physical chemical effects. For instance, water may extract water-soluble acidic deterioration products from oils, particularly when freshly produced, by preferential, or partial, solubility in water. The subsequent centrifugal separation of the wash water thereupon removes these deterioration products which had been in solution in the oil and so are not otherwise removable by centrifuging, since centrifuging does not separate materials in solution from their solvents. Washing may also carry forward a chemical reaction to a point where the intermediate is converted into an end product, water combination, emulsion, or other condition, where centrifuging can become effective. Deterioration products in oils may thus be quickly carried on over into the sludge or the sludge-and-emulsion stage and then thus removed by centrifuging. Fuller's earth and some other such materials which have an affinity for (or adsorbent action upon) water or other wash liquid may, upon the diffusing of such liquids through the liquid to be treated, have their mass or their gravity so increased as to promote greatly their ease of separation.

Temperature effects on viscosity and specific gravity differentials are closely related. The differences in rates of volumetric expansion with temperature form a means of obtaining, or increasing, the differential of specific gravity up to a value which will permit of gravity or centrifugal separation by heating (or cooling) to an appropriate temperature. It is even possible in some cases to reverse the relative gravities. The temperature coefficient of viscosity is very large, and heating of viscous liquids to a temperature which will reduce the viscosity to a point which will give ready centrifugal separation is very widely used.

Increase of temperature, however, increases the Brownian movement and so renders more difficult the settling or centrifuging out of small particles of colloidal or semicolloidal size.

Another temperature effect present in separation is the temperature-solubility coefficient. Even with such nominally mutually insoluble materials as oil and water there is a small solubility of water in oil, which increases with rise of temperature. Consequently the separation of water from oil at high temperature will leave some water in solution with the oil, which precipitates and appears as a fog or a cloud in the oil on cooling. This may be of importance in electrical insulating oil and makes desirable its centrifuging at moderate temperatures such as 30° to 40°C. or just sufficiently high to obtain a suitably low viscosity. In more viscous oils, it may be desirable to increase the temperature till the viscosity is reduced to about 300 sec. Saybolt Universal (or about 65 centistokes).

Emulsion breaking can often be effected by centrifuging without other treatment than heating to obtain a suitably low viscosity. Heating also brings about a number of other effects which facilitate breaking the emulsion.

A carefully conducted heating, just up to the point where the inner phase of the emulsion is partially evaporated, thereby breaking the surface film, may be employed, provided the operation is so conducted as not to cause excessive evaporation and frothing. The heating must be thermostatically controlled and may be applied in either a pipe line or a vessel. In addition, a cold liquid or one similar to the inner phase may be fed in, just after the heating, to condense vapor and to control frothing.

Emulsions may be dissociated by vacuum instead of by heating, to get a partial evaporation of the inner phase, although this method seems to be slightly more difficult to control as to frothing and loses the advantages of heating. Freezing may break some emulsions, just as long standing and long settling may cause dissociation, but these methods are less applicable for production operations.

Chemical methods, such as the addition of salts or acids, may be employed to dissociate emulsions.

MACHINE-OPERATING FEATURES

Starting devices and driving mechanisms must take into account the large moment of inertia of the bowl and the high speed to which any centrifugal bowl must be brought and be sturdy in design with ample capacity to start the bowl smoothly and to bring it quickly up to speed without jerk or shock.

Light bowls, such as the tubular type, are usually driven by a flexible fabric belt, and, as their inertia is small, they can easily be brought up to full speed by letting the belt slip during this short acceleration period.

Belt-driven machines with bowls of great mass are often started by allowing belt slip to occur at first to a greater, and later to a less, extent while the machine is being brought up to speed. This is the simplest method possible, requiring no special mechanism and giving very good results for infrequent starts, but it usually needs the personal attention of an operator, for the several minutes required to attain full speed and may entail undue wear on the belt.

Centrifugal friction clutches with the weights driven positively give an extremely smooth start, and the wearing pieces may be of brake lining or other readily renewable materials. Springs may be used to hold back the friction weights until the motor or the pulley gets up to a predetermined fraction of full speed, which is a considerable advantage for induction motors, particularly single-phase motors with split-phase starting coils.

Automatic electric controllers, which regulate the motor speed so as to bring the motor and centrifuge smoothly and gradually up to full speed together in a predeterminable time cycle with a regulated power demand on the supply lines, are quite advantageous for basket-type centrifuges on large production schedules requiring definitely laid-out schedules for time for loading, time to bring up to speed, time at full speed, time for slowing down, and time at standstill for unloading. Usually in such a cycle the time for accelerating to full speed is greater than any of the others and so is the most important to control. This method is the most economical of power, as none need be wasted in friction or in attendant's time and so, for some uses, well justifies its high first cost.

Driving mechanisms may be direct belt from a motor or line shaft to the bowl spindle pulley or by worm wheel and worm with either a true worm tooth or a helical worm and straight-face worm-wheel tooth, or, more rarely, by bevel gears. Basket centrifuges use the belt drive almost exclusively, but the smaller, high-speed centrifuges may use either, both giving quite adequate service results. For either type of drive the principal requirement is that the size for belts or gear teeth must be amply proportioned in order to give adequate wear resistance and to offset the handicap of the high speeds involved.

Lubrication of vertical spindles carrying bowls mounted on the upper end requires special provisions for the feed and maintenance of a lubricating oil film. A bearing whose lower end can be immersed in oil may have helical oil-pumping grooves on either the journal or the bearing to raise the oil to the top end. Such grooves may give high pumping pressures, for instance a 7 m. per sec. surface velocity can be made to develop a pressure of 2 atm. The edges of oil grooves must be chamfered to spread the oil peripherally. Such grooves serve also to wash grit out without scoring the surfaces. The top bearing of such a machine is subjected to severe operating conditions.

If, as is usually the case, full speed is above the critical speed, the bowl which rotates about its form center at low speed changes over as it passes through the critical speed to rotate about its center of gravity. This changeover is the usual cause for noise and rough running while coming up to speed. At full speeds, above the critical speed, the spindle will weave slightly and a flexible bearing support must be used if excessive bearing stresses are to be avoided.

The flexible support should be well damped to reduce the amplitude and continuance of vibrations.

The attachment of tightly closed lubricating-oil tubes to provide feed of lubricant under pressure presents certain difficulties because of the vibration of the bearing as it follows the spindle positions. On this account, oil is usually fed to a recess in the upper end of the bearing by an oil-feed tube which is not fastened to the bearing mechanically but is freely supported from the frame in a position to drop the oil into the recess. Either vertical or helical grooves may be used to conduct and to distribute oil to all parts of the bearing. A cap or cover should be fastened to the upper end of the bearing with a hole just large enough to clear the spindle surface to prevent windage from blowing the oil feed off the top end of the bearing. The surface speeds are so high that oils of low viscosity would be satisfactory for continued operation at full speed, but, owing to the need for a lubricating film at the lower speeds during starting and stopping and the need for maintaining a supporting film even after some wear has taken place, the "medium" or "heavy" oils are more suitable considering the all-round conditions of service, particularly considering that vertical centrifuge bearings seldom have a complete oil film.

Balance and vibration have a major effect on wear and performance. Bowls are dynamically balanced when built so that not only the bowl runs smoothly but also the spindle, in order that the frame may not suffer severe vibrating forces. The frame is usually supported upon rubber cushions or held by heavy springs to relieve stress upon the foundation bolts and foundation. Machines with bowls running at speeds of 6000 r.p.m. or more are balanced so that frames and other parts do not have amplitudes of vibration greater than fractions of 0.001 in. Basket centrifuges with speeds of 500 to 1500 r.p.m. may have higher vibration amplitudes. In such machines, which are often loaded and unloaded every few minutes and where the load masses cannot readily be disposed so as to eliminate unbalance, the suspension of the frame on springs is of considerable advantage.

Hollow tubular bowls, having a small mass relative to the frame and suspended on the drive shaft, may be run in frames solidly bolted to the foundation. Guide bushings and semiflexible means for damping side swing are usually provided.

Wear and corrosion present difficulties, if they occur to an appreciable extent in the bowl, since this is the most expensive and carefully made part of a centrifuge. Such changes may not only alter the performance but change the balance or mechanically weaken the parts. Corrosion-resisting materials, such as monel, chrome-iron, chromium-nickel-iron, or protective coatings, such as tin, plated by the hot dipping process, and rubber coatings, are used to meet special conditions.

ENERGY AND POWER REQUIREMENTS

The stored energy in the moving parts may be expressed as

$$E = \tfrac{1}{2} I\omega^2 \qquad (22)$$

where E is the energy in kilogram meters; I is the polar moment of inertia in kilogram meters; ω is the angular velocity in radians per second.

This stored energy is usually quite considerable and may take the full input from the motor or line shaft for periods of 1 to 10 min.

Equation (22) may be restated in a form to give the approximate values for the energy stored in rings of 1-cm. square section and different radii for the mean section, at various speeds, assuming the wall section to have a specific gravity of 7.83, the same as steel. This form is

$$E = 2.75 \times 10^{-9} r^3 \text{ (r.p.m.)}^2 \qquad (23)$$

where E is the stored energy in kilogram-meters; r is the mean radius of the section in centimeters; r.p.m. is the speed in revolutions per minute.

The accompanying curves (Fig. 15) show the range of values for stored energy.

The power lost in friction of the bearings and of the driving mechanism of underdriven machines is mainly in the top bearing on the high-speed spindle, next to the bowl, which may represent as much as half of the total machine friction because of its high speed and the large diameter needed to get a sturdy construction for the support of the bowl. Next in order of magnitude is usually the lower bearing, which is generally of much smaller diameter than the top bearing. A worm and worm wheel have generally a relatively small friction loss. A belt drive may likewise have a high efficiency if the belt is not too thick and stiff, and if of adequate width for the load to be carried. The friction in other bearings, if any, is usually a small factor in the friction losses. All these considerations point to the need for attention and supervision on the part of the operator to see that the high-speed spindle bearings (particularly the top bearing) are kept well lubricated and are inspected regularly so that they can be repaired or replaced immediately, if required.

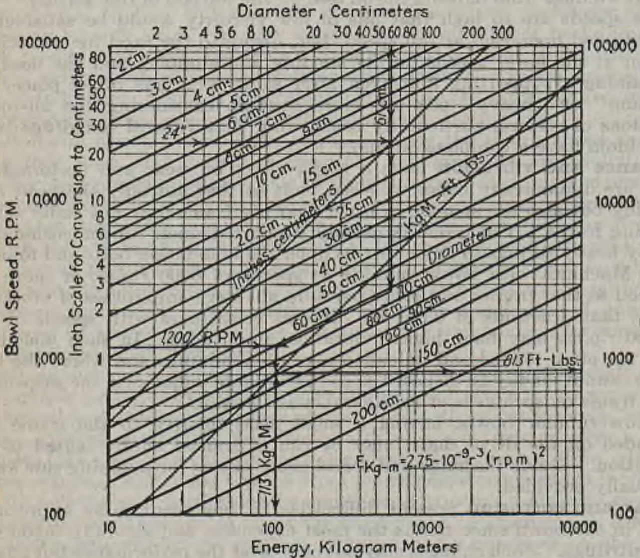


Fig. 15.—Energy in rotating steel rings per square centimeter cross section.

The power lost in accelerating the through-put and in the interior passages is one of the inherent factors incident to centrifuging. In a batch centrifuge of the basket or centrifugal-dryer type, where the bowl is loaded, brought up to speed for the centrifugal treatment, drying, etc., and then stopped (usually by brakes employed to save time), the energy stored is irretrievably lost. In such cases the acceleration of the load along with the bowl is calculated by the methods already described for that stored in the bowl. In continuous centrifuges the liquid fed into the bowl must be continuously accelerated up to the linear velocity of the largest diameter part of the bowl through which it passes (if the liquid is to be subjected to the full centrifugal effect). The power required for load acceleration may be expressed as

$$L = \frac{1}{2}\frac{G_2}{g}V^2 = \frac{1}{2}\frac{G_2}{g}r^2\omega^2 \qquad (24)$$

where L = power, kg. m. per sec.
G_2 = through-put kg. per sec.
r = radius (in meters) of the largest part of the bowl with which the liquid rotates at the full speed of this part.
ω = angular velocity in radians per second.
g = acceleration of gravity, m. per sec. per sec. = 9.81.

The power may also be expressed in horse power as

$$\text{H.p.} = 5.167 \times 10^{-9} G_3 R^2 \,(\text{r.p.m.})^2 \qquad (25)$$

where G_3 is through-put in pounds per minute, and R is the radius in feet.

If slippage does not occur, the energy imparted to the liquid at the maximum radius it reaches in the bowl is recoverable on its return toward the center line before its exit over the discharge lip. The energy in the "discharged" liquid or liquids at the point of discharge is that corresponding to the radii of the discharge "lips" (rather than the radii of the largest part of the bowl they may pass through), plus that small quantity of energy represented by the liquid velocity radially outward for emergence. This latter quantity is relatively insignificant and is in any case supplied by the pumping pressure due to the centrifugal pumping "head" between the check ring and discharge-lip radius, or radii, and so may best be included with the energy required to overcome liquid friction through the interior passages.

If the discharge lips are provided with shapes for discharge passages which direct the stream "backward," the reaction tends to drive the bowl forward, and so a considerable fraction (about half) of the energy in the liquid discharge may be recovered depending upon the tangential angle and the discharge-passage friction.

The liquid friction in the interior passages absorbs an amount of energy which may be as large as that in the liquid discharge.

The friction head is of course supplied by the centrifugal pumping head, produced by the difference in radii of the check ring near the center and the discharge lip. No part of this energy can be "recovered," although the value may be kept small by designing passages of ample size and with smooth curves, so that a small "head" will supply it and so allow the use of discharge lips of small radii.

If a discharge lip (or weir) leads to nozzles discharging in a radial or partially radial direction, a separate determination may be made for the energy required by calculating the centrifugal head between the outer and inner radial positions of entrance and exit of the nozzle.

RELATION OF SIZE, DRIVING POWER, AND COSTS

Cream Separators. The accompanying set of curves (Fig. 16) gives representative data for the through-put of liquid-liquid, continuous, centrifugal separators and customers' prices as cream separators, of the standard type using a simple belt drive and the corresponding data for the net horse power required for operation at full liquid load. (For motor costs see pp. 2713, 2718, 2719.)

Oil Purifiers. The set of curves (Fig. 17) gives values for the maximum through-put of continuous liquid-liquid centrifuges and customers' prices, as standard oil purifiers, and the corresponding data for the driving power of the standard types. Special features and accessories are not included in these price figures as the addition of motors, double pumps for inlet and return, heaters, and supplementary blotter presses, such as are used for insulating oil, all mounted upon a single movable platform or truck would add costs about equal to that of the centrifuge alone.

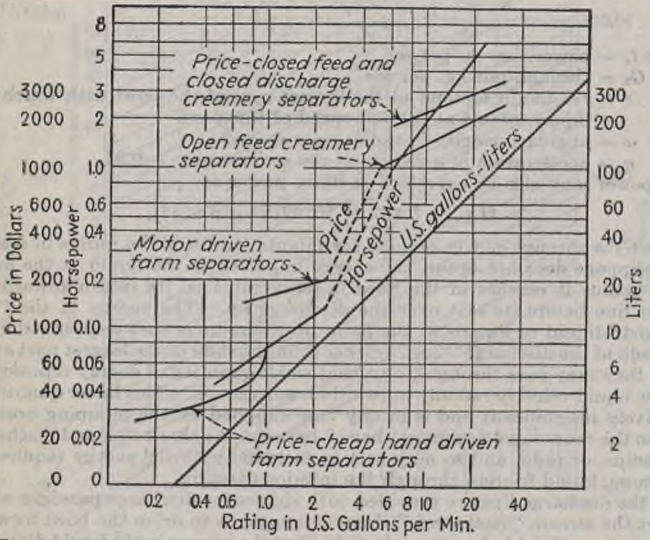


FIG. 16.—Milk separators—relation of rating, price, and power requirements.

Basket Centrifuges. Basket centrifuges cannot readily be rated in terms of through-put but are usually rated by basket diameters. The curves in Fig. 18 give representative figures, showing the relation of holding capacity, price, and the horse power required for such standard work, as laundry "drying," dry-cleaner's solvent drying, and the handling of simple chemicals. The through-puts must be estimated from the drying characteristics of the materials handled, the "dryness" required and the operating schedule that may be maintained under the labor conditions, and other work that the operator may need to attend to.

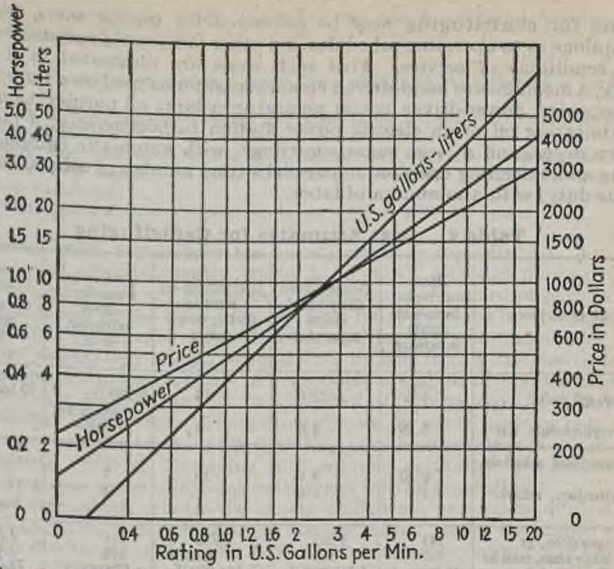


FIG. 17.—Oil purifiers—relation of rating, price, and power requirements.

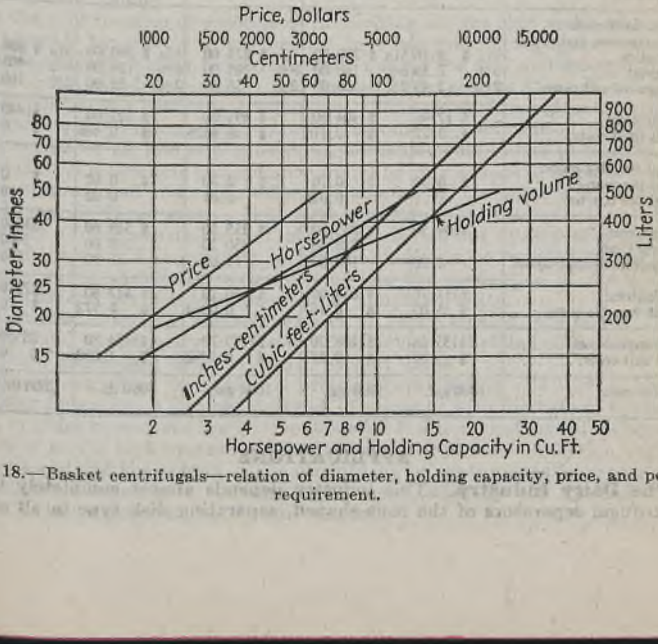


FIG. 18.—Basket centrifugals—relation of diameter, holding capacity, price, and power requirement.

Costs for centrifuging may be estimated for typical cases, based on assumptions as to operating schedules and other factors, representative of the usual conditions of service. Five such cases are illustrated in Table 2, namely, a medium-size hand-driven cream separator as used on a farm, a large creamery size, power-driven cream separator, a large oil purifier, as used for the lubricating oil of an electric power station turbogenerator, a large-size laundry dryer, and a large sugar centrifuge, with automatic equipment for loading and unloading operated in parallel with a number of others, on conditinuous duty, with a minimum of labor.

Table 2. Cost Estimates for Centrifuging

Type of service	Hand-driven farm-size cream separator	Power-driven dairy-plant cream separator	Lubricating-oil purifier electric power station	Centrifugal laundry dryer extractor, 48 in.	Sugar centrifugal sugar refinery, 48 in.
Capacity, gal./min.	1.6	22.0	15	3–200 lb. loads/hr.	10 tons per hr.
Power requirements, h.p.	0.09	4.0	4	7½	50
Daily operating schedule, hr.	0.67	9.0	24	7	24
Daily attention, man-hr.	1.0	1.0	1	2	8 (1 man, 3 machines)
Annual operation, hr.	243	3,285	8,780	1,941	7,200
Annual labor time, man-hr.	365	365	365	533	2,400
Annual through-put	23,350 gal.	4,350,000 lb.	7,884,000 lb.	1,074,000 lb.	72,000 tons
First cost	$122.50	$2800.00	$2750.00	$2900.00	$8,000.00
Annual fixed costs:					
Obsolescence and depreciation	1/15 $ 8.16	1/10 $ 280.00	1/10 $ 275.00	1/10 $ 290.00	1/10 $ 800.00
Interest	6% 7.35	6% 168.00	8% 165.00	6% 174.00	6% 480.00
Insurance and taxes	2% 2.45	2% 56.00	2% 55.00	2% 58.00	2% 160.00
Subtotal	$ 17.96	$ 504.00	$ 495.00	$ 522.00	$1440.00
Unit fixed costs	$ 0.77	$ 0.116	$ 0.0628	$ 0.486	$ 0.020
Annual variable costs:					
Cost per man-hr	$ 0.30	$ 0.70	$ 0.70	$ 0.60	$ 0.65
Cost per h.p. hr	....	0.025	0.01	0.02	0.005
Labor cost	$109.50	$ 255.50	$ 255.50	$ 319.80	$1560.00
Power Cost	....	328.50	350.20	218.00	5400.00
Repairs and supplies cost	8.20	100.00	75.00	75.00	150.00
Subtotal	$117.70	$ 684.00	$ 680.70	$ 612.80	$7110.00
Unit variable costs	$ 5.05	$ 0.157	$ 0.0864	$ 0.572	$ 0.0988
Total annual costs	$135.66	$1188.20	$1175.70	$1134.80	$8550.00
Total unit costs	$ 5.82	$ 0.273	$ 0.1492	$ 1.058	$ 0.119
Unit for costs	1000 gal.	1000 gal.	1000 gal.	1000 lb.	2000 lb. ton

APPLICATIONS

The Dairy Industry. This industry depends almost completely upon centrifugal separators of the cone-shaped, separating disk type in all of its operations for obtaining cream for table use and for making butter. The principal classes of uses are:

1. Farm milk separators to obtain cream from whole milk for shipment or for butter making.

2. Factory milk separators to obtain cream for table use, for butter making, and for ice-cream manufacture.

3. Factory whey separators to recover butterfat from whey after making cheese.

4. Factory milk clarifiers to remove sediment from milk before bottling for distribution to customers.

5. Factory, gathered-cream separators to skim old or gathered cream.

The **farm milk separators** are usually small in capacity, *i.e.*, less than 3 gal. per min., most frequently run by hand crank, although motor or gasoline-engine belt drives are available. A reasonable standard of performance is a butterfat loss in the skim milk not over 0.025 per cent by Babcock test, or about 0.085 per cent by Röse-Gottlieb or other extraction methods.

Factory separators are all power-driven by belt from a line shaft, electric motor, or steam-turbine wheel. A reasonable standard of performance is a butterfat loss of 0.01 per cent by Babcock test or 0.05 per cent Röse-Gottlieb.

Petroleum Refineries. The principal uses for centrifuges in petroleum refining have been for bright-stock dewaxing and the separation of sludge from acid treating. Solvent dewaxing and solvent extraction processes have also used various types of special centrifuges for the separation.

In the bright-stock process the slowly chilled oil, previously diluted with a solvent naphtha, is passed through the centrifuge, where the chilled amorphous wax discharges at the outer bowl lip and is swept continuously away by hot water. Removal of the naphtha by distillation yields an oil with a cold test only about 30° to 50°F. higher than the temperature to which the mixture had been chilled.

In the acid treating of gasoline or lubricating oil, the acid and acid sludge will separate by gravity in one or two days but may be separated in a few minutes after contacting by continuously mixing the acid and gasoline or oil (without too violent agitation) and immediately passing through a liquid-liquid, continuous centrifugal separator. This gives a complete control of the reaction, preventing overreaction and secondary deterioration, thus reducing gasoline or oil loss and conserving valuable constituents present in the crude oil. A reasonable performance would be only a trace of mineral acidity in the separated oil and a neutralization number for the organic acids of 0.05 to 0.10 mg. KOH per gram. It is quite important that the next stage, which is usually a fuller's-earth contact, should be carried out at once after centrifugal separation. The quantity of clay needed may be reduced 30 per cent or more, along with some saving of acid and the production of gasoline, or oil with several per cent less losses and of a higher quality.

In solvent dewaxing, the solvent and oil being treated, are mixed, refrigerated, and centrifugally separated. The solvent is distilled off, or otherwise recovered and reused with very small solvent losses.

Solvent extraction is employed principally to obtain oils with a higher viscosity index, *i.e.*, a lessened decrease of viscosity with increase of temperature, in order to combine low friction losses at low temperature with adequate viscosity at the high operating temperatures of internal-combustion engines at full speed and heavy load. The centrifuges, or other means for separation, are often arranged in a countercurrent series, whereby the finished oil is given its last extraction, by "fresh" solvent.

Vegetable Oil Refining. The process required (such as for cottonseed oil) is a treatment with caustic soda in a water solution to saponify organic acids, remove gums, etc., which may be accomplished by mixing in a moderate-size, continuous reaction vessel and passing the mixture at once through a liquid-liquid, continuous separating-type centrifugal separator. The soap stock is discharged at the outer lip and pure oil at the inner lip. The oil loss in refining depends on the free fatty-acid content which varies with age, and other conditions in the cottonseed-oil mills and upon crop conditions. The loss ranges between values of about 6 to 10 per cent of the raw oil with kettle refining and gravity settling, but with centrifugal separation the loss may be 2 per cent less, *i.e.*, 4 to 8 per cent of the raw oil. A better color is obtained in the refined oil and lower values for moisture and free fatty-acid content.

In **palm oil** and **olive oil,** the problem consists largely of getting a quick separation from water or saps to avoid fermentation and rancidity in the oil, resulting from standing while settling by gravity. The recovery of oil is increased 3 to 5 per cent and all may be equal to "first pressing" or "first grade," instead of about half "second grade" and up to 5 per cent of "third grade." Cast iron and tinned steel are suitable for bowls and covers, but all copper and brass must be avoided.

In **orange** and **lemon oil,** the problem consists of skimming the very small quantities of oil released from the rinds by crushing and expressing, from the water and juices. Monel or chromium-iron alloy bowls and covers are desirable to avoid corrosion.

Soya bean oil requires not only a refining process but also degumming, which may be accomplished either by solvent extraction or by a hydrolyzing process. In either case a centrifuge provides a quick and complete recovery of high-grade oil.

In both **fish oil** and **whale oil,** the problem consists of recovering the oil, free of water after cooking or "trying out" and free of shreds of flesh, bone, scales, etc. A nozzle type of liquid-liquid discharge bowl is used for the first stage separation, so that the solids are discharged through the nozzles along with enough oil-free water to sweep them along and out of the receiving cover. Oil-free water is discharged over the outer liquid lip, and oil with some water emulsion is discharged from the inner liquid lip. The oil and oil-water emulsion discharged at the inner liquid lip is given a second centrifugal purification through a liquid-liquid oil purifier, to recover water-free oil, and water that may be returned for reuse. Whale oil is now produced largely in apparatus installed for this purpose on specially built ships, which accompany fleets of whalers doing the chasing, killing, and capturing. The whale's body is hauled alongside or through a specially provided stern chute, cut up, and the blubber "tried out." The fresh oil is separated from water and is purified by passing through liquid-liquid, continuous-type centrifugal purifiers to remove water and solids. This ensures a clean oil which can be stored for months without acquiring rancidity during the whaling season, which lasts six to nine months, before return to port, thereby greatly enhancing the quality and consequent price for the oil. The gain may be as much as $15 per ton.

Wool-grease Recovery. This process consists in **skimming** the wool grease washed out of sheep's wool in the process of scouring it prior to carding. A great deal of dirt, sand, etc., is also washed out, requiring settling or decanting vats or the use of basket-type batch centrifuges, prior to skimming the wool grease out by means of continuous, liquid-liquid centrifugal separators.

The wool grease, particularly if quite clean and light in color, has some value, but it is even more important to remove the grease from the wash water before it is run off into sewers or water courses, to avoid stream pollution. For this reason, skimming types of bowls have been used in order to give the maximum purification to the water by removing the last traces of grease.

Valve-type bowls of the liquid-liquid solid type are now being used in wool-scouring processes, in which the solids are continuously discharged through the valves, the cleaned and degreased wash water is discharged at the outer discharge lip (to be returned for reuse in washing), and a wet grease is discharged at the inner lip for recovery of grease. This wet grease is usually given a second centrifugal separation, to remove the water and further to purify it preparatory to refining to pure lanolin or the production of fatty acids.

Lubricating-oil Purification. One characteristic requirement common to all cases of lubricating-oil purification is the need for a centrifugal purifier in constant operation upon a continuously recirculated portion of the lubricating oil, whereby an amount equal to the whole amount of oil in the system is separated from any water that may be present and purified of sediment and solids of all kinds every few hours. A liquid-liquid, continuous-separator type bowl is employed in all cases except for small Diesel engines where water is practically always absent, or in locomotive or marine Diesels where water is not available for water washing the oil. This requirement of constant centrifugal purification is not so much for the purpose of saving oil, important as that is for marine installations, as to reduce wear on the turbines and engines and to avoid stoppages, unexpected outages or interruptions of service, and repair costs.

It is important in all continuous, by-pass purification systems to keep in mind that the criterion of the purification should be the condition of the oil in the whole system and *not* the purification effect of a single passage through the purifier. In general, and with few exceptions, the centrifugal purifier should be operated at a high through-put rate, even though at a somewhat lessened contrast between the feed and the purified oil discharged from the purifier.

The larger number of passes more than makes up for the somewhat smaller purification coefficient (provided the sediment is not too small in size to be removable). It may be shown that the optimum through-put rate is

$$C = \sqrt{\frac{s}{u}} \tag{26}$$

where s is the rate of sediment production, and u is the ratio of sediment in the outgoing oil to that in the feed at unit through-put.

Steam turbine oil is subjected to heat in the bearings, air contact, water contact, and cooling in the water-cooled cooling coils. The heating and air contact promote oxidation, with the production of organic acidity; and in the later reaction stages, sludge. These reactions, in turn, decrease the ease of separation of oil from such water as may come from steam-gland leaks, cooling-water leaks from water-cooled bearings or in water-cooled oil coolers, or condensation on surfaces or by absorption, solution and reprecipitation from the atmosphere, or even, as a result of the chemical reactions in the oil itself, from dissolved oxygen.

The emulsion sludge as well as the soluble sludges (metallic soaps, etc.) tend to precipitate on cooling and thus to coat the surfaces of water-cooling coils, to interfere with heat transfer, and to involve difficult cleaning operations for their removal. On the other hand, the *freshly* produced organic acidity is many times as soluble in water as in oil and so may readily be washed out by mixing a small stream of boiling water with the oil and immediately separating this water in a continuous liquid-liquid centrifugal separator. The continuous centrifugal separator further protects the turbine from danger of water leaks into the oil and removes worn bearing particles and dirt from the oil. A reasonable maximum performance for a centrifugal purifier on wet turbine oil is that at which it leaves a moisture content of 0.05 per cent and correspondingly less at lower capacities. The neutralization number and steam emulsion number may become stabilized, but at values that vary markedly with the oil, the operating temperature, and other service conditions. Neutralization numbers of 0.5 to 1.5 mg. KOH per gram of oil and steam emulsion numbers of 300 to 600 sec. are frequently found in steam-turbine oil in service, when centrifugally purified by a continuous by-pass system. In such a system there should be present only traces of sediment or sludge.

Diesel-engine lubricating oil seems to be slightly subjected to oxidation, but is markedly affected by cracking or burning, owing to the high temperatures, and accumulates worn metallic particles owing to the high bearing pressures and the reciprocating motion, as well as dirt from the combustion-air and breather connections. Some residues come from the fuel oil and sometimes fuel dilution occurs through failure to fire, owing to lack of compression, bad mixture, moisture in the fuel oil, etc. The sediment in the oil consists largely of finely divided carbon, semicolloidal in size, along with small proportions of worn metal and metallic oxides, sand, dirt, etc. The latter types of sediment are readily removable by centrifugal clarifiers for liquid-solid clarification, but the carbonaceous material requires a high degree of clarification for satisfactory removal. By suitably large centrifugal clarifiers on a continuous by-pass purification, the total sediment in the oil in the system may become stabilized at values of 1.0 to 1.5 per cent. It appears undesirable to allow the sediment to reach values much over 2.0 per cent.

If hot water is available, washing the oil with water may be resorted to, using water proportions up to 40 per cent, and then immediately passing the well-mixed water and oil through a liquid-liquid, continuous centrifugal separator, thereby flushing most of the finely divided carbon out of the oil and the bowl along with the water and pocketing only the dense sediment in the bowl. This sediment may be cleaned out of the bowl by hand upon stoppage of the centrifuge. Hot-water washing markedly improves the centrifugal purification of the oil.

Rolling-mill oil usually has large amounts of water splashed in from the cooling water on the rolls and large amounts of mill scale and dirt. Purification requires a liquid-liquid, continuous-separating type of centrifugal bowl, with sludge pockets of considerable size and with gravity presettling tanks for catching the major part of the water and the coarser sediment. A reasonable performance for the maximum through-put appears to be not more than 0.050 to 0.200 per cent of moisture and 0.05 per cent sediment left in the purified oil, depending upon oil temperature and viscosity, and correspondingly lower values at less than maximum through-put.

Gasoline-engine crank-case oil purification is carried out by centrifugal purifiers mounted alongside the engine and operating continuously on a part or all of the lubricating oil being pumped to the engine bearings in only a few types of automobiles at the present time, because of the moderate size of such engines and the difficulty of getting adequate attention to cleaning or making repairs by drivers or garage repair men.

Batch reclamation on oil dumped from the crank cases of fleets of cars or trucks is quite common. If the quantities and conditions are such that the oil is to be used only as fuel, then a simple purification to remove free water and coarse sediment is adequate, which can be done by standing and gravity settling on a small scale where ample time is available, or instantly in liquid-liquid, continuous-separating types of centrifugals provided with sludge pockets of adequate size.

If the oil is to be reused in gasoline engines, water washing and treatment with neutralizing chemicals, such as sodium silicate and trisodium phosphate, may be carried out first, followed by settling or centrifugal separation, with subsequent removal of dilution in some cases and of fuller's-earth treatment for restoration of color in others. These last two steps are quite often considered unnecessary, since some dilution is sure to occur in service, and color is of itself no indication of service value. Furthermore it should be borne in mind that dilution, unless quite excessive, is not altogether a disadvantage since it performs an automatic self-regulating function. Since dilution is generally greatest in cold weather, it renders starting easier and delivery of oil through the lubricating tubes more certain, while in hot weather the lessened dilution causes higher actual viscosities and therefore better adaptation to high temperatures in the bearings and on the cylinder walls.

Block-test lubricating oil needs purification, primarily for the removal of worn metallic particles incident to the running in of newly made parts and of chips and filings and shop dirt left on the parts. Their removal avoids cutting and scoring of parts and hastens the running in operation. Gasoline engines are quite usually driven by gas and not by gasoline during the block-test run, so that dilution is not a factor. A continuous, by-pass purification, using a liquid-liquid, continuous-separating type of bowl, with sludge pockets of ample size to hold the solid sediment is usually employed. Water, spilled or leaking from the jacket water-cooling system, is often present and a stream of clean water may be passed through the centrifugal along with the oil to assist in its purification by water washing the oil.

Diesel fuel purification requires the removal of water and solid sediment. Even small proportions of water, particularly if unevenly distributed, will cause mis-firing and even stoppage of the engine. In marine service, where sea-water leakage to the oil storage tanks is almost sure to occur, since these tanks are usually placed next to or against the outer hull, or where sea water is pumped in to provide ballast in emptied oil tanks, water separation is of great importance. Solid-sediment removal is desirable in order to reduce piston and cylinder or cylinder-liner wear and clogging fuel-injection nozzles. Water may be separated partially by baffled, gravity settling tanks, but sediment is not removable to any appreciable extent in this way. The liquid-liquid, continuous-separating type of centrifugal, provided with ample sludge-holding pockets, will remove both water and sediment and may be built to operate even during fairly heavy seas.

A reasonable standard of performance for a centrifugal purifier is somewhat difficult to state, because of the wide range of viscosities and specific gravities of Diesel fuel.

For crudes and residues of 300 sec. Saybolt Furol at 50°C. (about 640 centistokes) the moisture residue may be kept to between 0.5 and 1.0 per cent, provided the oil is heated close to the boiling point of water. In any case, fairly effective centrifugal separation is obtainable only when the viscosity at the temperature of separation is not much greater than 300 sec. Saybolt Universal (*i.e.*, about 65 centistokes).

The sediment under similar conditions as to temperature and viscosity at the temperature of centrifuging may be kept within values of 0.05 per cent. It is well to bear in mind, in regard to ash content, that some ash in oils is due to mineral compounds held in solution and so is not separable by any means.

Transformer-oil purification requires principally the removal of the last, minute traces of moisture. Even new oil that has been thoroughly dried at the refinery picks up some moisture in shipment, either by solution from atmospheric humidity or condensation on the upper walls of the containers, or it is produced chemically by the reaction of the oxygen dissolved in the oil from the air and combining with constituents of the oil. Some small amounts of pipe scale, rust, etc., may be picked up also, but this is easily separated in the course of the removal of moisture in a liquid-liquid type of continuous centrifuge. The advantage of the centrifuge over the blotter press, which functions not as a filter but as an absorber of moisture, lies in the fact that the centrifuge continues to operate indefinitely at constant and full effectiveness, while the blotter press works with a decreasing effectiveness and at some unpredictable moment must have the blotter papers exchanged for new or dried ones. The blotter paper may break and allow oil to go through unpurified for some time since such a break is not visible to the operator. A reasonable standard of performance at maximum through-put of the centrifuge is the leaving of not more than 0.001 per cent of moisture in new oil at temperatures not much above that of the room. Owing to the solubility of water in insulating oils, which may amount to 10 to 30 parts per million by weight at room temperatures and ten or more times this in oil at 80°C., it is futile to centrifuge insulating oils at elevated temperatures, except as a preliminary step to break any emulsion that might be present in a used oil, to be followed, after cooling and standing long enough to precipitate the dissolved water, by a second centrifuging cold. An alternative is to centrifuge, once only, at the ambient temperature, but at such a reduced through-put as will give the desired purification. Even if centrifuged at elevated temperatures, resulting in a brilliantly clear oil with less than 0.001 per cent of *free* water and having a dielectric breakdown of 25,000 to 40,000 volts between 1-in. disks separated by a gap of 0.100 in., *just after being centrifuged*, such oils will, *upon cooling*, precipitate out some of the dissolved moisture, and the oil will become cloudy and may show only a few thousand volts breakdown value. After several years of service, transformer oils develop organic acidity and finally sludges from oxidation and other reactions. The sludge tends to deposit upon, and so to cause overheating of, the coils by interfering with their dissipation of heat. The organic acidity cannot be **removed** by centrifuging but may be **reduced** by water washing and then centrifuging, or by treating with mild alkalies, such as sodium silicate or trisodium phosphate solutions and then centrifuging. The sludge and water present in old used transformer oil is directly removable by centrifuging, and this simple mechanical separation is at present the general practice in electric power systems. The treatment with neutralizing chemicals, followed by centrifugal separation which restores the oil to its original neutralization number, may be followed by contacting with 1 to 2 per cent by weight of fuller's earth. The diffusion of a small amount of hot water into the oil facilitates the settling of the fuller's earth and then, after settling, the upper layers may be decanted off by means of a floating draw-off pipe to a centrifugal separator for final purification. This results in the production of a reclaimed oil that is as good as, or better than, new oil for reuse.

Electric switch oil is subjected to the heat of the electric arc between the opening contacts of the electric switch, producing gases (principally hydrogen) and finely divided carbon. Water and metallic particles may also be present which are readily removed by a liquid-liquid type of continuous centrifuge, leaving, however, some semicolloidal carbon in the oil which may then be readily removed by subsequent passage through a paper-blotter press dressed with two or more thicknesses of dry paper.

Cable oil requires purification for the removal of the last traces of moisture and sediment, in order to obtain a homogeneous insulating material combining high dielectric breakdown strength, high electric resistivity, low energy loss under alternating electric stresses, and freedom from tendency to change or to deteriorate in service.

A liquid-liquid type of continuous centrifugal separator may be used for the first-stage separation to remove moisture and sediments, waxes, etc., followed by passage through a blotter press dressed with two or more thicknesses of blotter filter paper. The centrifugal purification should remove all the free water, particularly if carried out at about room temperature so that change of the blotter papers which absorb moisture may be rendered very infrequent. However, blotter papers, particularly when newly installed, allow some fibers to be swept off the back side and into the oil, so that a considerable portion of the oil first passed through the press should be returned to the system and rerun.

Varnish and Lacquer Clarification. Clear varnish needs clarification for removal of dirt and fine sediment which otherwise spots or breaks the smooth, shiny surface of the dried varnish. A high clarification coefficient is necessary, resulting in a need for very effective types of liquid-solid, clarifying types of bowls and run at relatively low through-puts to obtain the maximum possible clarifying effect. It is not feasible to state in words a standard for a reasonable performance since this is judged by flowing a small quantity of the clarified varnish over a small piece of flawless plate glass and observing the surface at an angle which reflects the light obliquely. Under such a test the clarified varnish should show no specks or at most only two specks on a plate 10 by 6 in., while if more than four specks show, the varnish should be rejected.

Pigment lacquer clarification is desirable to remove not only the dirt and sediment that may have been picked up by the lacquer and, more particularly, the coarser ground pigment particles which may be reclaimed for regrinding but also partly dried surface films, etc. Only a moderate clarification coefficient is needed, because of the high specific gravity of the pigments and their relatively large size. A simple type of hollow bowl is generally used, though sometimes multiple concentric cylindrical shells are provided. The machine is frequently equipped with an adjustable drive for speeds between one-third and full speed. By adjusting the speed and through-put, any desired degree of clarification may be obtained. The glass-plate or metal-panel method is used for judging the setting and degree of clarification to be employed for each grade or kind of pigmented varnish.

Dry-cleaners' solvent clarifiers have been used for many years to remove continuously the dirt and sediment picked up from the clothes in the washer, by-passing the solvent continuously while the washing is in progress through a liquid-solid type of centrifugal clarifier. Simple forms of bowls have generally been used, of either the double-concentric, cylindrical-shell, medium-speed type or the single-shell, tubular, high-speed bowl The moderate-speed double-shell bowl has been most widely used. The practice abroad has been to use bowls with several concentric cylindrical shells, or bowls with conical disks much smaller in diameter than the shell, to give adequate dirt-holding space.

A typical plant would have one or two washers, 36 in. in diameter and 54 in. long, each of which would have with it a centrifugal clarifier with a bowl having a dirt-holding volume of 330 cu. in. (5400 cc.).

Dry cleaners use "dryers" of the centrifugal-basket type to remove the solvent held in the washed clothes when removed from the washer. The clothes retain a weight of solvent equal to 1¼ to 1¾ times the normal dry weight of the goods, which should be removed mechanically rather than by being evaporated off, not only to conserve solvent but also to avoid leaving solvent and other odors on the clothes. The basket centrifuge should leave not more than 15 to 30 per cent of solvent by weight for removal in the heated tumbling dryer or the cabinet air-blower type of steam-heated dryer to be evaporated off. (This "dry cleaning" does not remove the normal water content present in cloth, although it may be reduced somewhat, *i.e.*, be 8 per cent instead of 10 per cent.)

A typical basket dryer would be about 26 in. in diameter, with a basket depth about half the diameter and run at speeds of 700 to 900 r.p.m. Such a basket will hold up to 90 lb. of clothes (normal dry weight) and require about 5 to 6 min. for centrifuging and about 3 min. for loading and 2 min. for unloading, thus making a cycle in 10 to 11 min. and allowing about four to five loads per hour, when allowances are made for the operator to attend to other duties when not loading or unloading.

Laundry drying must be carried out in two stages, first the mechanical removal, second drying by heat. When removed from the washer, clothes hold two or three times their "dry" weight of water, which may be reduced to 20 to 30 per cent in a centrifugal extractor but is usually reduced only to 50 per cent since ironing or pressing requires some moisture. Dry clothes normally contain 6 to 14 per cent moisture and, if dried below this, become harsh and brittle. A piece of cloth does not seem wet to the hand until it carries about one-third of its dry weight of moisture. Wet-wash laundries operate on a larger scale than dry-cleaning plants and use the largest sizes of extractors, up to 72 in. diameter. Power is required mainly for acceleration, which may take four times that to maintain full speed.

Sugar refineries employ centrifugal dryers of the basket type for the removal of syrup from the crystallized sugar after concentration in evaporators.

The requirements of large-scale, low-cost, competitive production have led to the use of the largest possible sizes, up to 60 in. in diameter, with a holding capacity of about 15 cu. ft. In addition, the operation must be as nearly automatic as possible and involve a minimum of labor, attention, and cost. For these reasons, mechanical aids of all kinds find a great field of usefulness, such as time switches for starting and accelerating, running and stopping the driving motors, means for filling the basket, mechanical unloaders, dumping mechanisms, etc.

Breweries grow yeast cells in mash tubs or copper vats of large size, such as 12,000 gal. (50,000 l.) with a yeast-cell content which may, with certain grain yeasts, be 3 per cent by volume but with other mashes may reach 9 to 10 per cent. Whether grown for alcohol or beer, or for the yeast itself, it is desirable to terminate the growth definitely and quickly and to separate completely beer and yeast instead of using the relatively slow, gravity settling methods formerly employed. This is now accomplished by centrifugal concentrator bowls, employing nozzles connected to passages leading from the bowl-shell wall so as to pick up and sweep out along with a small proportion of the liquid, all the yeast cells, which, being heavier than the wort, are thrown to the outermost parts of the bowl, while the wort travels toward the center between cone-shaped separating disks and is discharged through a top connection near the center.

Such machines are built in sizes up to about 15-in. diameter (38 cm.), operated at speeds of 5000 r.p.m., with through-puts of 85 gal. (320 l.) per min. on 3 per cent yeast and 60 gal. (230 l.) per min. on 9 per cent yeast, and requiring about 5 h.p. for drive. The concentration possible depends inversely upon the yeast richness, because a concentrate of about 60 per cent by volume of yeast cells will not flow readily enough to be easily discharged through the nozzles. On this basis, a practical limit would be a concentration to 5 per cent of the original volume for a 3 per cent yeast and to about 15 per cent of a 9 per cent yeast.

After the wort in which the yeast grew has been separated, it is customary to dilute the yeast concentrate to the original volume with cold clean water and then to reconcentrate by running through a centrifugal separator a second time, before passing the concentrate to filters for final recovery.

Rubber latex as taken from the trees consists of about 30 to 40 per cent by weight of rubber particles dispersed in a milk-like sap. The rubber particles have a lower specific gravity than the serum, and although they will coagulate on standing and can be made to coagulate very readily by acidification, their concentration is far less readily effected than is the case with cream from cow's milk, in spite of the similarity of dimension of the dispersed particles (about 3μ), and their closely similar specific-gravity differentials.

The upper limit of concentration of about 73 per cent at which the concentrate of rubber latex practically ceases to flow is appreciably lower than that for cream, which is about 80 per cent.

Concentration to about 60 per cent is desirable for the two reasons of saving in shipping weight and the better quality of product when made from centrifugally concentrated latex.

In spite of the presumable ease of concentration, practical results from centrifugal concentration have only recently been obtained. This has been done by specially designing liquid-liquid concentrator bowls with quite high separating coefficients through the use of extra high speeds and with appropriate internal-passage constructions.

Latex concentrators are expected to produce a concentrate with at least 60 per cent dry rubber-content solids, equivalent to 61 to 62 per cent total solids, and not over 12 to 14 per cent solids in the serum.

It is worth noting that centrifugal concentration also removes considerable proportions of water-soluble proteins. This may be accentuated by remixing with water and recentrifuging, one or more times. Rubber prepared by centrifuging has improved mechanical and electrical properties; recentrifuging further improves the electrical properties markedly.

Chemical plants make use of basket centrifuges for the recovery of precipitates or crystals from mother or wash liquors in a great many processes such as caustic slimes after evaporation, ammonium sulfate, dinitrobenzol, anthraquinone, nitrotoluene, etc. Wherever a relatively liquid-free cake is desirable, or a spray wash is needed, the basket centrifuge offers advantages over the filter press or rotary filter, often resulting in half the liquid-residue percentage otherwise attainable. It is important to keep in mind that the smaller diameter baskets permit higher centrifugal separating effects, which may offset their smaller load-holding space and consequently greater labor and operating cost per unit of cake.

A typical operating schedule employing a 40-in. diameter perforated basket, holding about 7 cu. ft. or 500 to 600 lb. and taking 10 h.p., would be as follows:

Loading	2–3 min.
Accelerating to full speed, 900 r.p.m.	3–4 min.
Running at full speed	1 min.
Slowing down (with brakes)	2 min.
Unloading (bottom dumping)	2–4 min.

The number of loads run through must allow for some "lost time," division of attendance with other duties, etc., so that from three to five loads may be run per hour.

Machine-shop chip-handling and cutting-oil reclamation requires the use of both a basket centrifuge for the removal of the cutting oil from the chips and one or two liquid-liquid, continuous-separating bowls with ample sediment-holding capacity for purification of the oil.

The chips may need to be broken into small pieces first, to facilitate handling and to reduce their volume, for which one of the several types of swing-hammer or ring-roll crushers is suitable. In this way, wiry, spiral chips may be broken so as to get 50 to 60 lb. per cu. ft., instead of about 10.

When loaded into a typical 40-in. basket centrifuge taking 350 lb. per load, run by a 10-h.p. motor, the centrifuging schedule may be as follows:

Loading	5.5 min.
Accelerating	1.5
Running at full speed	3.5
Slowing down (with brake)	1.0
Unloading	5.0
	16.5 min.

With such a schedule from 2½ to 3½ loads may be run per hour, and with a spare basket this may be increased to 4 to 6 loads per hour.

Metal chips hold from 20 to 25 per cent of their weight of oil and after centrifuging only 1 to 2 per cent, thus effecting a 90 to 96 per cent recovery of oil (60 gal. oil per ton of chips).

This oil is dirty, septic, and full of fine metallic particles. In order to sterilize as well as to wash out dirt and chips, water may be added, and the oil and water heated to boiling and then allowed to settle overnight to separate the water and sediment, which may be drawn off from the bottom of a cone-bottomed settling tank. The oil may be drawn off from the top and after heating may be passed through a liquid-liquid, continuous-separating type of centrifugal bowl provided with a large sludge-holding space to retain the fine solids. A plant handling 5 tons of chips per day and recovering 300 gal. oil would have a storage and heating tank for the dirty oil with a volume of about 450 gal. for the oil and water and a small accessory tank with heating coils and thermostatic control to maintain a uniform temperature of 70°C. (158°F.) on the oil as it passes into a centrifugal separator, requiring 2 h.p. for drive, which will separate the water and sediment at a through-put rate of 3 to 4 gal. per min., leaving not more than 0.05 per cent water and 0.5 per cent sediment in the purified and sterilized oil.

Machine-shop grinding requires the use of a coolant and lubricant on the grinding wheel and the piece being ground. It is not possible to obtain the best finish without a very clean coolant. The coolants used may be special oils, or special oils or soap compounds dispersed in water. Centrifugal clarifiers have recently been applied to these coolants, resulting in much smoother surfaces, in absence of scratch lines that are difficult to remove in later polishing operations, and in material reductions in grinder-wheel wear.

Activated sewage sludge may be concentrated by valve-bowl clarifiers which discharge a concentrated sludge containing 6 to 10 per cent of solids by dry weight and a clarified water containing not more than 0.05 to 0.10 per cent, suitable for return to the secondary tanks. The concentrated sludge is passed into the digester tank. The increased concentration would allow the use of smaller and less expensive digesters and seems also to yield a greater amount of gas.

Miscellaneous applications include a large number of varied kinds of cases, requiring one or another of the types of separations already described, of which a few may be listed as examples, such as the recovery of seed oysters in oyster hatcheries, the recovery of blood serum free from red corpuscles in slaughterhouses, the clarification of viscose in the rayon industry, the clarification of medicinal preparations, the purification of essential oils, the recovery of caffeine from coffee extract, the recovery of oil and the purification of wash water in washing oily rags for reuse in machine shops, the recovery and separation of oil from steam-engine condensate, the clarification of fruit juices, etc. Wherever a liquid-liquid separation or a liquid-solid clarification, or some combination of these, needs to be carried out to a high degree of perfection, centrifuging should be given consideration.

SEPARATION OF DUSTS AND MISTS

BY EVALD ANDERSON

CLASSIFICATION OF SUSPENDED MATTER OR GAS DISPERSOIDS

Solid and liquid suspended matter in gases, herein designated **gas dispersoids,** or simply **dispersoids,** may be divided into two main classes according to their formation: **mechanical** gas dispersoids and **condensed** gas dispersoids. Mechanical dispersoids are formed by dispersing larger masses into finer particles, as by grinding solids and spraying liquids, and these will herein be called **dusts** and **sprays,** respectively. Condensed dispersoids are formed by condensation from the vapor phase of the substance and these will herein be called **fumes** when solid and **mists** when liquid.

One of the most important characteristics of such dispersoids affecting their separation from the gas is their size, and herein lies an important difference between the two classes. Although no rigid dimensions can be given, it can be assumed that much the larger portion, by weight, of mechanical dispersoids will consist of particles whose average diameters will be between 5 and 50 microns (1 micron = 1/1000 mm. or about 1/25,000 in.), while with condensed dispersoids, the corresponding diameters may be from 0.3 to 3 microns.

The corresponding ratio of the masses and volumes of the average particles in the two classes is, therefore 5000:1 which emphasizes the reason for the well-known differences in behavior of the two classes of dispersoids in separating them from the dispersion medium.

Of considerable practical interest are certain dispersoids, intermediate between the above two classes, which result from spray drying of liquids containing solids in solution, or from the combustion of dispersed organic material containing non-combustibles, such as powdered coal. Here the final dispersoid is the solid residue of the spray droplet or the ash residue of the coal particle, and these originally mechanical dispersoids therefore may approach in size those of the condensed class.

Condensed dispersoids although originally very minute nearly always agglomerate into larger particles and may often grow to equal, or to exceed, the size of the other type. Raindrops and snowflakes are familiar although perhaps extreme examples.

Table 1. Measurements of Diameters of Various Dispersoids

Mechanical dispersoids	Diameter, μ	Condensed dispersoids	Diameter, μ
Cement dust	40μ	Pigments	1–4μ
Sprayed zinc dust	15μ	Alkali fume	1–5μ
Flour-mill dust	15μ	Tobacco smoke (tar mist)	0.25μ
Talc dust	10μ	Condensed zinc dust	2μ
Coal dust	10μ	NH_4Cl fume	0.1–1μ
Silica dust	5μ	ZnO fume	0.05μ
Powdered coal ash			5–10μ
Spray-dried milk			1–3μ

Mechanical dispersoids carried by gases containing condensable constituents also act as centers of condensation, so that appreciable portions of the condensed constituent may be attached to the mechanically dispersed particles. An example of this is cement-kiln dust where the alkali salt condenses in minute particles on the surface of the irregular dust fragments.

Separation Methods. The common separation methods in commercial use may be conveniently divided into the following classes:

1. **Gravitational**—as ordinary settling chambers.
2. **Inertial**—as baffle chambers or centrifugal apparatus.
3. **Filtration**—as bag filters.
4. **Sprays**—as scrubbing towers.
5. **Electrical**—as Cottrell electrical precipitators.

In all of the methods listed, collection by contact and adhesion on surfaces is also involved and in certain cases this action may predominate, as in wire-hung settling chambers or diffusion chambers for certain fumes.

PRINCIPLES AND APPLICATIONS

Laws of Particle Motion. For the motion of particles immersed in fluids, whether gases or liquids, it is found that the ratio of the inertia forces to the viscous forces gives a criterion for dynamically similar fluid motions about the particles:

$$\frac{\text{Inertia force}}{\text{Viscous force}} = \frac{Du\rho}{\eta} = Re = \text{Reynolds number} \tag{1}$$

where, in any consistent units,

D = diameter of sphere (corresponding dimensions of geometrically similar particles).

u = relative velocity between particle and fluid.

ρ = fluid density.

η = fluid viscosity.

When the resistance to the motion of a particle in a fluid, as determined experimentally, is expressed as a dimensionless drag coefficient C and is plotted against the Reynolds number Re, a plot consisting of a single curve is obtained covering an exceedingly wide range of conditions. In Fig. 1 are given such a plot and tabulation of data for spherical particles. Similar curves for other shaped particles are given by Rouse ("Fluid Mechanics for Hydraulic Engineers," Chap. XI, McGraw-Hill, New York, 1938) and by Prandtl and Tietjens ("Applied Hydro- and Aeromechanics," Chap. V, trans. by Den Hartog, McGraw-Hill, New York, 1934).

The drag coefficient C is defined by

$$C = \frac{2F}{\rho u^2 A} \tag{2}$$

where, in consistent units,

F = frictional force on particle.

A = projected area of particle normal to direction of motion.

Over the range of Reynolds numbers generally encountered in unit operations, the flow around suspended particles may be divided into three regions: streamline (when viscous forces predominate), intermediate, and turbulent (when inertia forces predominate). The region above a Reynolds number of 200,000 is important only in ballistics and aircraft design. For spherical particles, the curve of Fig. 1 may be approximated by the three straight lines shown. (Shepherd, private communication, 1933.) For a Reynolds number less than 0.0001, the magnitude of the particle diameter approaches that of the mean free path of the fluid medium, and a correction factor must be applied.

For gravity settling, a particle reaches a constant settling speed when the net gravitational force F_g on the particle equals the frictional resistance F. The net gravitational force is obtained by correcting the weight of the particle to account for the buoyancy of the displaced fluid. If ρ_s is the actual particle density, then $(\rho_s - \rho)$ includes the buoyancy correction. For gravity settling of particles in gases, the buoyancy correction is small compared to the experimental error and can ordinarily be ignored, so that ρ_s can be substituted for $(\rho_s - \rho)$ in the following equations:

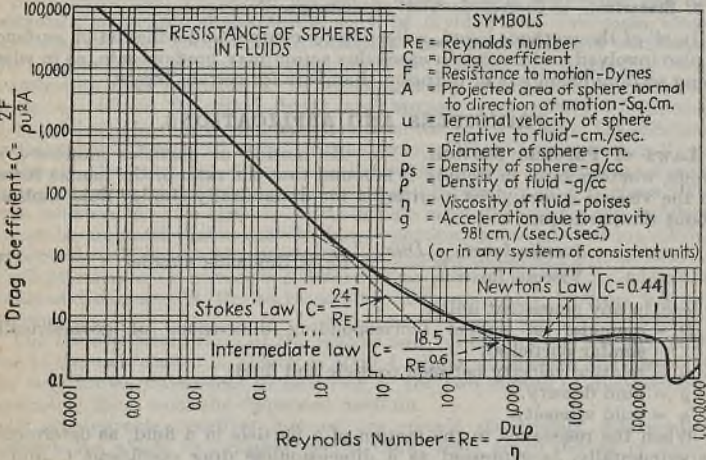


FIG. 1.—Resistance of spheres in fluids. See tabulated values on p. 1853.

For spherical particles

$$F_g = g\pi D^3(\rho_s - \rho)/6 \tag{3}$$

where, in consistent units,

g = acceleration due to gravity.

ρ_s = particle density (actual, *not* bulk density).

Hence, at constant settling velocity, $F = F_g$, and

$$C = \frac{2F}{\rho u^2 A} = \frac{g\pi D^3(\rho_s - \rho)}{3\rho u^2 A} \tag{4}$$

or, substituting $\pi D^2/4$ for A and rearranging,

$$u^2 = \frac{4gD(\rho_s - \rho)}{3\rho C} \tag{5}$$

From Eq. (5) the various laws of settling corresponding to the regions of Fig. 1 can be derived by substituting the respective function of *Re* for *C*. In Fig. 2 these laws, together with their limitations, are presented in tabular form to indicate their relationship to particle size, free-settling velocities, methods of measurement, and methods of collection.

Values of Drag Coefficient C for Spherical Particles*

Re	C	$C\,Re$	$C\,Re^2$
0.1	240	24.0	2.4
0.2	120	24.0	4.8
0.3	80	24.0	7.2
0.5	49.5	24.8	12.4
0.7	36.5	25.6	17.9
1.0	26.5	26.5	26.5
2	14.4	28.8	57.6
3	10.4	31.2	93.7
5	6.9	34.5	173
7	5.4	37.8	265
10	4.1	41.0	410
20	2.55	51.0	1.02×10^3
30	2.00	60.0	1.80
50	1.50	75.0	3.75
70	1.27	89.0	6.23
100	1.07	107	10.7
200	0.77	154	30.8
300	0.65	195	58.5
500	0.55	275	138
700	0.50	350	245
1,000	0.46	460	460
2,000	0.42	840	1.68×10^6
3,000	0.40	1,200	3.60
5,000	0.385	1,920	9.60
7,000	0.390	2,730	19.1
10,000	0.405	4,050	40.5
20,000	0.45	9,000	180
30,000	0.47	14,200	426
50,000	0.49	24,500	1.23×10^9
70,000	0.50	35,000	2.45
100,000	0.48	48,000	4.8
200,000	0.42	84,000	16.8
300,000	0.20	60,000	18.0
400,000	0.084	33,600	13.4
600,000	0.10	60,000	36.0
1,000,000	0.13	130,000	130
3,000,000	0.20	600,000	1.8×10^{12}

* These are average values based on the combined data of Allen, Arnold, Bacon and Reid, Dorr and Roberts, Flachsbart, Jacobs, Knodel, Krey, Liebster, Lunnon, McKeehan, Millikan, Schmidt, Schmiedel, Shakespear, Silvey, Wieselsberger. For values of Re less than 0.1: $C = 24/Re$; $C\,Re = 24$; $C\,Re^2 = 24\,Re$.

In connection with the separation of suspended particles from gases, by far the most important laws of settling correspond to the region of Fig. 1 where viscous forces predominate. Here, for values of Re up to about 2,

$$C = \frac{24}{Re} = \frac{24\eta}{Du\rho} \tag{6}$$

Substituting this experimental value of C in Eq. (5) gives

$$u = \frac{gD^2(\rho_s - \rho)}{18\eta} \tag{7}$$

Equation (7), theoretically derived, is known as Stokes' law.

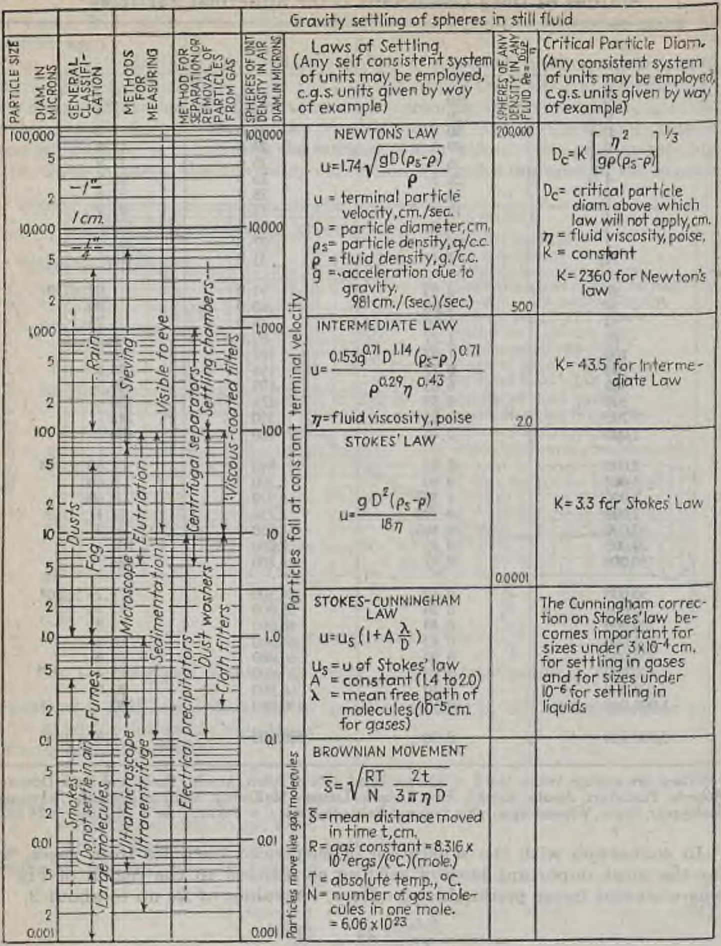


FIG. 2.—Characteristics of gas dispersoids.

For values of Re between 2 and 500 approximate values of u can be calculated by using the straight-line approximation of Fig. 1, which has been called the intermediate law.

Here

$$C = \frac{18.5}{Re^{0.6}} \tag{8}$$

Substituting in Eq. (5) gives

$$u = \frac{0.153g^{0.71}D^{1.14}(\rho_s - \rho)^{0.71}}{\rho^{0.29}\eta^{0.43}} \tag{9}$$

When it is desired to calculate the free-falling velocity of a given sized particle directly from the experimental curve of Fig. 1, trial-and-error methods must be used. To obtain a direct solution, a separate plot of CRe^2 versus Re can be drawn by selecting points from Fig. 1 and computing CRe^2. For spherical particles

$$CRe^2 = \frac{4gD^3\rho(\rho_s - \rho)}{3\eta^2} \tag{10}$$

and can be computed directly, as it is independent of u. The corresponding value of Re can then be determined from this separate plot, and the velocity u determined from the relation $Re = Du\rho/\eta$.

Further references and a more detailed discussion of particle motion are given in a paper by Lapple and Shepherd [*Ind. Eng. Chem.*, **32,** 605–617 (1940)].

The Gravitational Method. This method, as exemplified by the ordinary settling chamber, is applicable only to mechanical dispersoids because its effectiveness depends on the rate of fall of the particles, and this rate, being dependent on the size and weight of the particles, becomes so slow for the condensed dispersoid that the method here becomes impracticable. In Table 2 are listed the rates of fall of various sized spherical particles of density 2 g. per cc. in still air at room temperature. They were computed by means of the curve of Fig. 1 and Eq. (5).

Data on actual dust-fall rates in industrial applications with moving gases and irregularly sized particles are extremely meager. The indications are, however, that the rates are generally close to the theoretical but may, at times, be as much as 50 to 60 per cent lower, owing to the presence of eddy currents and the irregularity of particle shape. From Table 2 it is evident that particles of less than 200-mesh size will fall too slowly to settle in the ordinary dust chamber. It will also be observed from the corresponding Reynolds numbers that, contrary to prevalent usage, Stokes' law does not apply for the particle size range generally encountered in dust-settling chambers, except for the smallest sizes. The intermediate law (Fig. 2), however, will usually be applicable for the larger sizes.

Settling chambers should be designed to give sufficient time of settling for the smallest size of particles it is desired to remove and should obviously be made wide and low rather than high and narrow. Chains or wire curtains to minimize eddies are desirable and may tend to collect the finer dust through contact.

The actual settling rate of the smallest particle to be collected should preferably be determined experimentally. When this is impractical, but where this particle size is known, the theoretical settling rate can be calculated from the following approximate numerical expression for Stokes' law (for particle sizes under 200 mesh):

$$u = \frac{D^2\rho_s}{10^4} \text{ ft. per sec.} \tag{11}$$

where D is the average diameter of the particle in microns and ρ_s is its specific gravity.

Table 2. Theoretical Rate of Fall of Spherical Particles in Still Air

$\rho_s = 2.0$ g. per cc. $\rho = 1.2 \times 10^{-3}$ g. per cc. $\eta = 1.8 \times 10^{-4}$ poises

Diameter of particle		Rate of fall, ft. per sec.	Reynolds number, *Re*
Microns	Equivalent Tyler mesh		
0.1	...	0.000004*	8.1×10^{-9}
1.0	...	0.00021*	4.3×10^{-6}
10	...	0.020	4.0×10^{-3}
44	325	0.38	3.4×10^{-1}
74	200	0.93	1.4
104	150	1.6	3.5
147	100	2.7	8.0
208	65	4.1	17.1
295	48	6.0	36.0
417	35	8.7	74.0
833	20	17.1	290.0

* Stokes-Cunningham correction factor included.

Thus, if the smallest particle to be settled out is 50μ and its specific gravity is 4, the theoretical rate is

$$u = \frac{(50)^2 4}{10^4} = 1 \text{ ft. per sec.}$$

Assuming 50 per cent of this figure as the actual rate, the settling rate for such a particle is, therefore, about 0.5 ft. per sec.

For ordinary industrial mill or kiln dust carried by gases of average velocity, where the particle size is unknown and where it is impractical to determine the settling rate by experiment, the order of magnitude of the settling rate of the smallest particles to be collected can be estimated from the following empirical formula:

$$u \text{ in ft. per sec.} = \frac{\rho_s(1.0 - 1.2X)}{3} \tag{12}$$

where ρ_s is the specific gravity of the particle and X is the fraction of the total dust to be settled out, provided X is less than 0.8. This equation is based on two series of tests [*Trans. Am. Inst. Chem. Engrs.*, **16**, 77 (1924)] with metallurgical and cement dust of size largely under 200 mesh. Comparison of actual settling rates of the slowest particles settled out and such rates calculated from the above expressions are shown below:

Table 3. Comparison of Actual and Calculated Settling Rates

Dimensions of dust chamber, ft.		Gas velocity, ft. per sec.	Collection, per cent	Specific gravity of particles	Settling rate, ft. per sec.	
Height	Length				Actual	Calculated
29	220	7.0	57	8	0.95	0.84
21	140	3.7	40	3	0.55	0.52

If u is the actual settling rate in feet per second of the smallest particle to be collected, Q the gas flow in cubic feet per second, A_c the cross-sectional area in square feet, and L and H the length and height in feet, respectively, of a proposed settling chamber, then obviously the dimensions of the chamber must be made such that

$$LA_c u = HQ \tag{13}$$

Apparatus. Except perhaps for the so-called Howard dust chamber (Fig. 3), which depends for its effectiveness on making H in Eq. (13) very small, by the use of a multiplicity of superimposed shelves, and on the large surface for collection by contact, there is at the present time practically no apparatus on the market for dust collection by the gravitational method. It, therefore, ordinarily devolves on the engineer in charge of the plant to design and construct the necessary equipment.

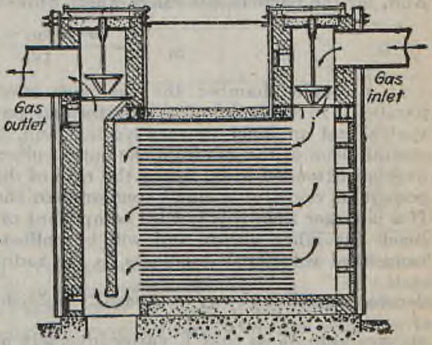


Fig. 3.—Howard dust chamber.

The Centrifugal or Inertial Method. Although with certain specific apparatus some condensed dispersoids may be collected by means of the inertial method it, like the gravitational method, is chiefly applicable to mechanical dispersoids like dusts and sprays; but due to the much larger separation forces available, smaller particles may be collected. As gravity is the separating force in the settling chamber, so the centrifugal force causes the separation of the dispersoids from the gas in centrifugal or inertial apparatus. This force, being directly proportional to the square of the gas velocity and inversely proportional to the radius of curvature of the gas stream, can therefore be made very large by making the gas velocity large and the radius of curvature of the gas stream small. Table 4 shows the relation of this separating force and the radius of curvature for a constant gas velocity of 60 ft. per sec., in terms of gravity (*i.e.*, the force of gravity taken as 1).

Table 4. The Centrifugal Separating Force in a Circular Gas Stream with a 60-ft. per Sec. Tangential Velocity for Different Radii

(Expressed in terms of gravity, *i.e.*, gravity = 1)

Radius of Curvature of Gas Path	Separating Force
Gravity	1
10 ft.	11 times gravity
5 ft.	22 times gravity
2.5 ft.	45 times gravity
1.0 ft.	112 times gravity
6 in.	224 times gravity
4 in.	336 times gravity
2.5 in.	538 times gravity
1.5 in.	896 times gravity

Although the laws of particle motion discussed herein are not strictly applicable to moving gases, particularly where the motion is highly turbulent, it may, nevertheless, be assumed that here also the resistance to the motion of the particles is such that the rate of motion is proportional to the net separating force and that the resisting or frictional force depends on the same factors, as in still gases. Bearing in mind the uncertainty of these assumptions, one may then calculate the separating velocity u in inertial or cyclonic apparatus in the same manner as the settling velocity in gravitational apparatus.

To calculate the separating velocity of a spherical particle in a centrifugal field, it is only necessary to replace the term g, the acceleration of gravity, by the radial acceleration of the particle $\omega^2 r$ or V^2/r, where ω is the angular velocity and V the tangential velocity of the particle and r is the radius of curvature of the path. Thus, the general Eq. (5) becomes, for a consistent system of units,

$$u^2 = \frac{4\omega^2 r D(\rho_s - \rho)}{3\rho C} \tag{14}$$

And, for the particle size range where Stokes' law applies, Eq. (7) becomes

$$u = \frac{\omega^2 r D^2(\rho_s - \rho)}{18\eta} \tag{15}$$

In a baffle chamber the gas path may be simple and approximately parallel. In cyclonic apparatus the gas path involves an outer downward vortex and an inner upward vortex, and, depending somewhat on design, the diffusion of the gas from the outer vortex to the inner vortex takes place over an extended area; hence the rate of diffusion, or radially inward component of velocity, is small compared to the tangential velocity of the gas. If u is larger than this inward component of velocity, the particle will never reach the inner vortex and will be collected. In cyclonic apparatus the tangential velocity V increases as the radius of curvature of the gas path decreases so that $V \sim \frac{1}{r^n}$. Recently published test data indicate that n may approximate $\frac{1}{2}$ so that V varies inversely as the square root of r [Shepherd and Lapple, *Ind. Eng. Chem.*, **31**, 972–984 (1939)]. Other experimental evidence and theoretical considerations suggest the value of n may approach 1 [see Lissman, *Chem. & Met. Eng.*, **37**, 630–634 (1930)]. For purposes of illustration assume that the separating action takes place in the outer vortex, and consider a tangential velocity V such that V^2/r (where r is the radius of the cyclonic apparatus) corresponds to the average acceleration present in the outer vortex. At the same time in the apparatus the average gas path will cover an angular distance θ, depending upon the particular design. The time available for separation will then be $r\theta/V$, giving a separation distance s,

$$s = \frac{r\theta u}{V} \tag{16}$$

By substituting for u from Eq. (14), and noting that $V = \omega r$, Eq. (16) becomes

$$s = \theta\sqrt{\frac{4rD(\rho_s - \rho)}{3\rho C}} \tag{17}$$

For the minimum particle sizes of interest in cyclonic collection where

Stokes' law applies, Eq. (15) is used, giving

$$s = \frac{\theta V D^2(\rho_s - \rho)}{18\eta} \tag{18}$$

Equation (18) may be used to calculate the tangential gas velocities necessary to move particles across the gas stream in apparatus with varying widths s, of the gas path, where s may correspond to the radial extent of the outer vortex in cyclonic apparatus. Table 5 shows such values computed for spherical particles of specific gravity 2.0 separated from air at atmospheric temperature and pressure. Values of V are given for θ equal to 4π radians, corresponding to the approximate conditions in certain cyclones.

Table 5. Relation of Tangential Gas Velocities, Particle Size, and Separating Distance in Cyclones

$\theta = 4\pi$ radians

Separating distance s, ft.	Particle diameter D, microns				
	1	5	10	20	50
	Tangential gas velocity V, ft. per sec.				
0.5	6,400	256	64	16	2.6
1.0	12,800	512	128	32	5.1
2.0	25,600	1024	256	64	10.2
4.0	51,200	2048	512	128	20.5

Table 5, being based on Eq. (18), assumes the applicability of Stokes' law which does not usually apply to particles much over 20 μ. The values listed are primarily intended to show order of magnitude. For conditons where Stokes' law does not apply, the required tangential gas velocities to separate a given sized particle will be somewhat higher than listed and will depend to a small extent on the radius of curvature.

Other investigators have concluded that the applicability of Stokes' law in a centrifugal field depends, not only on the particle diameter, but also on the tangential velocity and that Stokes' law in this case is valid only for values of VD smaller than about 1000 to 2500.

Minimum Size Particles Collectable. Cyclones approximate classifiers rather than collectors, and a significant factor in cyclone operation is the minimum size particle collectable.

Rosin, Rammler, and Intelmann [*Z. V. deut. Ing.*, **76**, 443–447 (1932)] have derived the following equation in a consistent system of units to determine the diameter, $D_{\text{min.}}$, of the minimum size particle to be practically completely separated from the gas stream in a given apparatus:

$$D_{\text{min.}} = \sqrt{\frac{9\eta b}{\pi N V(\rho_s - \rho)}} \tag{19}$$

where N = number of revolutions made by the gas stream in the apparatus.
b = width of gas stream.

Equation (19) is derived from Eq. (18) by letting $\theta = 2\pi N$ and $s = b$ and assumes a constant spiral or tangential velocity V across the width b.

Other investigators have derived similar expressions for $D_{\text{min.}}$. Thus, Lapple and Shepherd replace b with $e/2$ where e is diameter of exit duct [*Ind. Eng. Chem.*, **32**, 605–617 (1940)], and Lissman replaces b with $r/2$ where r is radius of cyclone (private communication).

Experiments by Shepherd and Lapple indicate that N may range from 0.5 to 3 and may average 1.5 for cyclones of normal proportions. It is also possible to determine N by experiment.

In Table 6 results from recent tests with three sizes of cyclones—6 in., 9 in., and 24 in.—with similar proportions are given. Here the minimum size particle, $D_{\text{min.}}$ nearly completely collected (98 to 99 per cent), was 5μ, 10μ, and 20μ for the 6-in., 9-in., and 24-in. cyclones, respectively.

Substituting the numerical values involved in Eq. 19 (N assumed to be 1.5, V taken as ratio of gas-volume rate to inlet area was 1340 cm. per sec., b was 2.5 cm, and η 1.8×10^{-4} poises) gave $D_{\text{min.}}$ 4.6μ, 6μ, and 10μ for the 6-in., 9-in., and 24-in. cyclones, respectively.

More comprehensive data are, however, needed before a general equation applicable to all conditions can be developed.

A certain proportion of particles smaller than $D_{\text{min.}}$ are always separated, partly because their initial positions are closer to the periphery and partly owing to agglomeration, but, until the effect of dust concentration and agglomeration is better understood, no satisfactory theory can be developed. In the absence of agglomeration Lapple and Shepherd [*Ind. Eng. Chem.*, **32**, 605–617 (1940)] have shown from theoretical consideration that a fractional separation X should be obtained on particles of size D_X smaller than $D_{\text{min.}}$ when

$$D_X = D_{\text{min.}}\sqrt{X(2-X)^{0.5}} \tag{20}$$

The following tabulation of data from tests with small-diameter cyclones, of the type shown in Fig. 8 indicate the degree of collection of fine particles.

Table 6. Collection Efficiency of Cyclones

Approximate particle-size distribution of suspended dust (density = 3 g. per cc.)

0–5μ	5–10μ	10–20μ	$+20\mu$
26 %	10 %	21 %	43 %

Pressure drop—4 in. water Inlet dust concentration—2 to 5 gr. per cu. ft.

Diameter of cyclone, in.	Per cent collected		
6	Total 90	-5μ 66	$+5\mu$ 98
9	Total 83	-10μ 60	$+10\mu$ 99
24	Total 70	-20μ 47	$+20\mu$ 98

It should be pointed out that all quantitative and theoretical considerations given above assume that the dust involved is completely dispersed and present in small concentration (less than 5 gr. per cu. ft.) so that particles will have no effect on one another. In practice, dusts are apt to show agglomerating tendencies which result in a larger apparent particle size than actual particle size. High dust concentrations will result in large particles carrying out small particles by mechanical means. Both these factors will result in higher efficiencies than predicted from an actual particle-size analysis. While Table 5 indicates that 1μ particles cannot be separated by a cyclone under conditions ordinarily employed, in certain cases efficiencies as high as 98 per cent have been realized on dusts having an actual particle size of from 0.1 to 2.0μ owing to the predominant effect of agglomeration.

Pressure Drop and Gas-volume Capacity of Cyclones. Lissman and Miller (Paper read at 1940 annual meeting, A.S.M.E., New York, Dec. 3, 1940), investigating cyclones of varying diameters and proportions with an involute entrance, arrived at the following empirical expression for the specific gas flow rate Q/a^2, for air at atmospheric pressure,

$$\frac{Q}{a^2} = K\left(\frac{A}{a^2}\right)\left(\frac{e}{a}\right)\sqrt{\Delta HT} \qquad (21)$$

where Q = air flow rate, cu. ft. per min.
a = diameter of cyclone, in.
A = area of cyclone inlet, sq. in.
e = diameter of cyclone outlet, in.
ΔH = pressure drop, in. water.
T = temperature of air, °F. abs.

When the ratio A/a^2 ranges from ⅛ to ½, and the ratio of e/a ranges from ¼ to ¾, then K is nearly constant and has an average value of 0.67. The range of cyclone diameter, a, over which this formula has been found accurate, is from 4 to 24 in. The length of these cyclones approximated 2 to 4 times a.

The given ranges of A/a^2 and e/a cover the normal proportions used in practice. For values of A/a^2 below ⅛ and normal values of e/a the value of K increases. For values of e/a below ¼ and normal values of A/a^2 the value of K decreases.

For gases at pressures other than atmospheric and densities different from air the following correction factor must be applied to formula (21)

$$\frac{Q'}{a^2} = \frac{Q}{a^2}\sqrt{\frac{1}{P\rho_g}} \qquad (21a)$$

where Q' = gas flow rate, cu. ft. per min.
P = pressure of gas, atm.
ρ_g = specific gravity of gas with respect to air.

Shepherd and Lapple [*Ind. Eng. Chem.*, **32**, 1246–1248 (1940)], investigating cyclones of the type shown in Fig. 6 (tangential rectangular inlet), arrived at the following empirical expression for cyclone friction loss:

$$F_{cv} = 16.0\frac{bh}{e^2} \qquad (22)$$

where F_{cv} is expressed as number of inlet velocity heads, b is the inlet width, h is the inlet height, and e is the exit duct diameter in consistent units. The tests covered the following range in proportions: (b/a) from 1/12 to ¼; (h/a) from ¼ to ½; (e/a) from ¼ to ½ (a is the cyclone diameter).

They hold that the cyclone friction loss and not the pressure drop is the measure of the static pressure and power which a fan must develop, and relate the friction loss to the pressure drop by the following expression:

$$\Delta H_v = F_{cv} - 1 + \left(\frac{4bh}{\pi e^2}\right)^2 \qquad (22a)$$

where ΔH_v is also expressed as number of inlet velocity heads.

For converting from inlet velocity heads to inches of water, the following formula may be used:

$$\text{One inlet velocity head, in. of } H_2O = 0.003\rho V^2$$

where ρ = density of gas, lb. per cu. ft.

V = cyclone inlet velocity based on the area bh, ft. per sec.

If the inner side of the inlet duct of a cyclone with tangential entrance is extended past the cyclone cylinder wall and into the annular space halfway to the opposite wall, the friction loss is reduced by over 50 per cent, and the constant 16.0 in Eq. (22) becomes 7.5 [Shepherd and Lapple, *Ind. Eng. Chem.*, **31**, 972–984 (1939)]. Such modifications to reduce the friction loss ordinarily also reduce the collection efficiency.

Further experimental study is necessary, particularly on pressure and velocity distribution in the cyclone and on relation between average entrance velocity and actual velocity near cyclone periphery, to provide satisfactory basis for any pressure-drop–gas-volume capacity theory.

Pressure Drop and Collection Efficiency of Cyclones. For dust with a reasonable uniform particle-size distribution, the collection efficiency of a well-designed cyclone increases with the pressure drop. Recent experiments (F. Miller, private communication), involving a pressure-drop range of from 1 to 8 in. of water, indicate that, at least for the smaller size cyclones and properly dispersed dust, the relation between the collection efficiency E (in per cent) and the pressure drop ΔH (in inches of water) can be expressed by the equation:

$$E = C_1 + C_2 \log \Delta H \tag{23}$$

where C_1 and C_2 are numerical constants. With 9-in. diameter cyclones and for the dust specified in Table 6, C_1 was 77 and C_2 was 8. The constants would be expected to depend to a large extent on the particle-size distribution of the dust.

Equipment

Baffle-chamber Type. In the ordinary baffle chamber, the distance s which the dispersoid particles must move to get out of the gas stream is seldom less than 3 ft., while the angular distance θ is ordinarily π radians or 180 deg. per pass. Since the dispersoids not separated ordinarily tend to redisperse between the different passes, it is doubtful if the successive separating actions are very effective. Under these conditions, a velocity of over 60 ft. per sec. would be required to separate 50μ particles. The actual gas velocities in such baffle chambers are ordinarily made less than this, and it might therefore be concluded that, except for their surface-adhesion action, the baffles in such chambers are of little use.

The obvious prerequisite of any dust-collection apparatus—that the separated dust be not again dispersed in the gas—is particularly important in the baffle type of apparatus for the collection of solid gas dispersoids, and with the increased gas velocities necessary to ensure the needed separating force, special precautions must be taken to ensure retention of the separated material.

In baffle chambers for the collection of liquid dispersoids, redispersion is naturally absent, except at very high gas velocities, and this also holds where solid dispersoids are collected in baffle apparatus with the baffles coated with a dust-retentive liquid like oil.

The ordinary large baffle chamber, like the settling chamber, is usually designed and built by the engineer in charge at the works and is therefore not available in the market. Better results, however, undoubtedly could often be obtained by a more careful analysis of the problem.

Dry Collectors. Special baffle chambers for the collection of dry dust are manufactured and sold, as for example the so-called **reverse-nozzle** type and **inverse-vortex** type; but no exact data as to costs and performances have been made available. Private tests on the reverse-nozzle type have shown satisfactory collection (over 90 per cent) of 100μ particles at draft losses of from 1 to 2 in. water. The collection efficiency rapidly decreases with the particle size and except for collection by surfaces is practically nil for dispersoids of 20μ or less. The first cost has in some instances been about 25 cts. per cu. ft. per min. of gas treated.

So-called "wet baffle chambers" are also used for ordinary dust, but their action more closely resembles that of spray towers and will be discussed under that heading.

Baffle Chambers for Wet Collection. Well-known examples of this type of collectors are perhaps the old so-called P-and-A tar extractor and the Calder-Fox acid scrubber. In both of these, high velocities and small radii of curvature are employed to the end that very fine spray particles and even some types of mists are collectable. Thus, in the Calder-Fox apparatus (Fig. 4), s is usually on the order of $\frac{1}{10}$ in., θ about $\frac{1}{2}\pi$, and V 100 ft. per sec. or more, with a pressure drop of from 3 to 10 in. so that according to Eq. (18) 2 to 4μ particles should be collectable. The practical results are even better, since certain types of sulfuric acid mists are easily separated from the gas. So long as the collected material flows freely, no difficulties of operation are usually encountered.

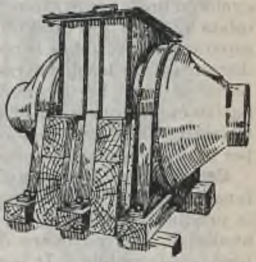


FIG. 4.—Calder-Fox scrubber. (*Chance & Hunt, Ltd.*)

Perhaps the most common apparatus of this type is the zigzag baffle chamber used for the separation of spray particles in the ordinary air-conditioning apparatus as shown in Fig. 5. This, however, is not suitable for condensed dispersoids.

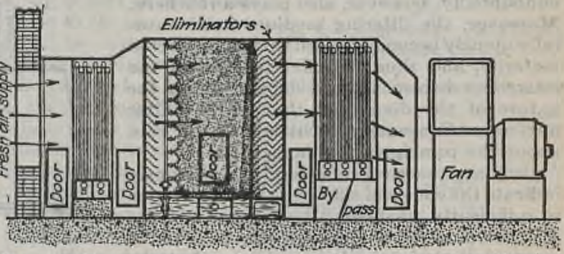


FIG. 5.—Zigzag spray eliminator in Sturtevant air-conditioning system. (*B. F. Sturtevant Co.*)

The ordinary spray chamber with baffles really is a wet baffle chamber, rather than a spray chamber. It will, however, be discussed under the latter.

Centrifugal or Cyclonic Apparatus—Cyclones. The most common example of this type is the ordinary cyclone (Fig. 6). This is very satisfactory for coarse dust though it is evident from Table 5 that with the ordinary size having a separating distance s of about 3 ft., even if θ, the length of travel of the gas in the apparatus, is 4π radians or two complete revolutions, a 20μ particle would require a minimum velocity of 96 ft. per sec.

Tests have shown that the pressure drop in cyclones ranges from two to five times the entrance-velocity head, so that with high velocities the power requirements often become prohibitive (see p. 1861).

The first cost will depend on local conditions but should not exceed 30 cts. per cu. ft. per min. and may be as low as 5 cts. per cu. ft. per min.

Contrary to prevalent opinion, the efficiency of a cyclone is the same whether under pressure or suction provided in-leaks of air or gas from the outside be prevented. The **induction dust collector** type of cyclone is shown in Fig. 7.

Multiple Cyclones. Another type of centrifugal collector is the **Multiclone** (Fig. 8) where small cyclonic units are arranged in parallel with common gas inlets and gas and dust outlets. In this apparatus the separating distance s is only 1 or 2 in. and the angular distance θ at least 4π radians, so that very fine dispersoids can be removed at reasonable velocities. Satisfactory collection of dispersoids as small as 5μ is claimed. The first cost is from 5 to 25 cts. per cu. ft. per min. of gas at pressure drops of from 2 to 5 in. water.

Centrifugal Dry-gas Cleaners. So-called "cinder fans" and the like, where the circular motion of the gas is produced by impellers in the apparatus itself, are available, but no exact data as to performance have been made public. It is unlikely, however, that their efficiency exceeds that of the standard large cyclone.

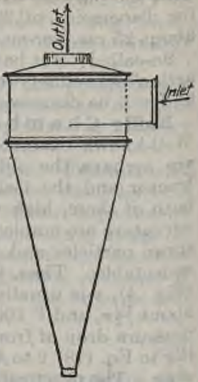


FIG. 6.—Standard long-cone cyclone.

The Filtration Method. It is ordinarily considered that the filtration method depends only on what might be called the sieving action, the filter being a sieve of a certain size, through which the particles like gas molecules and certain dispersoids smaller than the meshes of the filter pass and upon which the larger particles are retained. Collection by contact and adhesion undoubtedly, however, also plays a role here. Moreover, the filtering medium itself naturally quickly becomes coated with the collected material, and thus the effective size of the interstices becomes more dependent on the nature of the dispersoid than on the filter medium. Consequently little can be said about the principles of this method.

Ordinary practical considerations obviously indicate the choice of a filtering medium which is sufficiently unaffected by the gas and the dispersoid as to have a reasonable life. It must also obviously be so chosen that the dispersoid is retained and at the same time does not produce too great a draft loss.

The collected material must ordinarily be dislodged from the medium by mechanical means, and the frequency of such mechanical cleaning of the medium also determines the nature of the medium itself, since the draft loss increases with the amount of material collected on the medium so that the greater this cleaning frequency, the "tighter" can the medium itself be.

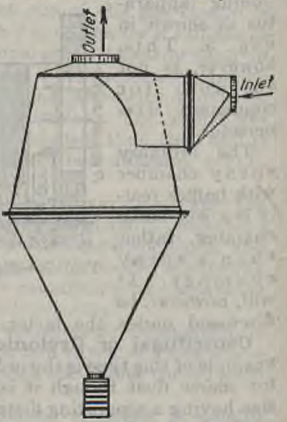


FIG. 7.—Induction-type cyclone. (*Federal Pneumatic Systems, Inc.*)

By far the most common filtering medium is cloth. This can, of course, be used only for relatively cold, non-corrosive gases, although a good quality wool cloth may stand as much as 150°C. and small amounts of acid.

For other gases, asbestos and glass-cloth, steel wool, etc., and for analytical purposes so-called fritted glass, etc., are also used, although these materials are of course more expensive than ordinary cloth.

The **bag house**, shown in Fig. 9, is the best known of this type of apparatus and usually consists of bags or tubes from 6 to 30 in. in diameter and from 6 to 30 ft. long. These are usually suspended from their closed, upper ends and have their lower open ends fastened and connected to a gas inlet header but may also be connected to headers at both ends. In either case, the lower header also serves as the dust hopper. The collected material is dislodged from the bags periodically, usually not oftener than once an hour, by means of hand- or mechanically operated shaking devices. Owing to the nature of the cloth, the draft resistance must usually not be permitted to much exceed 1 in. water, and, under these conditions, the gas-volume capacity is ordinarily from ½ to 1 cu. ft. per min. per sq. ft. of bag area, depending on the dust concentration.

With proper filtering mediums, the efficiency of this method is very high, except sometimes immediately after the filtering medium has been shaken and cleaned.

The cost of such bag houses on large scale may be from 50 cts. to $1.00 per cu. ft. per min. of gas treated. The cost of the bags themselves may run from 2 to 3 cts. per sq. ft., and these must be replaced at intervals varying from 30 days to several years, depending on the nature of the service.

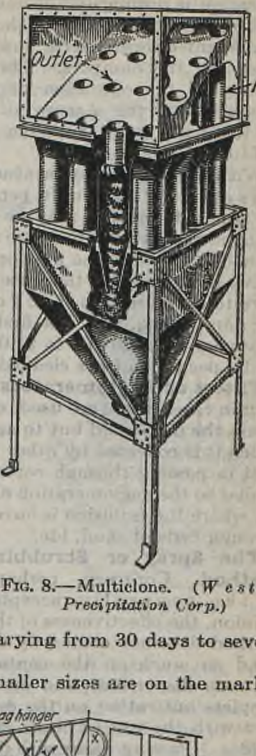


FIG. 8.—Multiclone. (*Western Precipitation Corp.*)

Although such ordinary bag filters in the smaller sizes are on the market, this type of apparatus, like settling chambers, is perhaps most often constructed by the works engineer.

Mechanical Filters. Since, as above stated, the rate of gas flow, or the draft loss, of a filter depends largely on the amount of collected material which is permitted to accumulate on the filter medium and therefore depends on the frequency of cleaning, many types of what may be called mechanical "filters" are available. In these, the cleaning or shaking operation proceeds at regular, rather short intervals, by means of automatic mechanisms. The filter units of this type may be either tubular or flat screen type (see Fig. 10, the Sly dust filter). They are ordinarily built in reasonably small units, to handle for example 1000 cu. ft. per min. and may be so arranged in parallel that one unit after another

FIG. 9.—Typical bag-house filtering system.

can be taken out of service through closing off the gas stream and the filters shaken (see Fig. 11, the Dracco Perfecto Filter). This cleaning or shaking operation is ordinarily automatic and usually comprises closing the dampers, shaking the filter units, either pneumatically or mechanically, sometimes accompanied by a reverse flow of cleaned gas through the filters, and, lastly, reopening the dampers. The frequency of the operation depends naturally on the nature of the service but may be from four to fifteen times an hour.

With this type of apparatus, gas flows of from 2 to 6 cu. ft. per min. per sq. ft. filter area may be used without unduly high pressure drop. Although for very fine dust or very heavy dust loadings this rate may have to be reduced to ½ to 1 cu. ft. per min. per sq. ft. The first cost may vary from 20 cts. to $1.00 per cu. ft. per min. of gas cleaned.

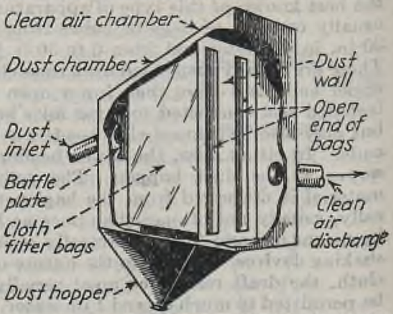


Fig. 10.—Sly dust filter. (*W. W. Sly Manufacturing Co.*)

Filters as Agglomerators. In certain cases filters are used, not to retain the dispersoid but to agglomerate it in passing through the filter, after which it is collected by other means. An example of this is the collection of mist in passing through cotton, glass wool, or fritted glass. This action is similar to the agglomeration of emulsoid particles, such as water dispersed in oil, where the emulsion is forced through beds of sand, etc.

The Spray or Scrubbing Method. Contrary, perhaps, to the generally accepted opinion, the effectiveness of this method does not ordinarily depend so much on the contact between dispersoids and spray droplets but rather on the contact with the wetted walls and baffles. Passing gases with dispersoids through straight, empty flues or towers fitted with ordinary sprays has, therefore, little effect, partly because the number of spray droplets is wholly inadequate, considering the number of dispersoids, and partly because the latter nearly always are surrounded by a film of gas which prevents contact with the spray droplets. Nor is the mere passage or bubbling of the gas through a column of liquid very effective, even when comparatively high columns are used, unless the gas itself be dispersed very highly throughout the liquid.

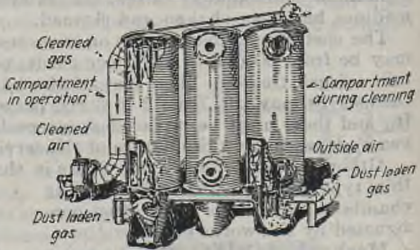


Fig. 11.—Dracco "Perfecto" filter. (*Dracco Corporation.*)

Spray Towers. As has been stated above, the open tower or flue, fitted with sprays, producing a finely divided "rain" or else several solid "curtains" of spray liquid, and employing fairly low gas velocities, is very ineffective. A proper spray tower should provide baffles which are continually wetted by the spray and so arranged that the tendency is to force the dispersoids against these by centrifugal action. Hence the gas velocity must be fairly high. These towers are therefore really centrifugal apparatus.

Some spray towers are merely cylindrical or rectangular flues with built-in partitions to make the gas take a zigzag course (Fig. 12) and usually with sprays located between these partitions. Others, which are built vertically, are filled with gas-distributing material (Fig. 13), such as board grids, terra cotta shapes, and coke, through which the gas passes, usually upward and countercurrent to the spray solution which is supplied from the top. Such towers usually operate with relatively high pressure drop, and they are very efficient with solid mechanical dispersoids, where choke-ups can be prevented, and with both liquid mechanical dispersoids and some liquid condensed dispersoids. Solid fumes are, however, largely unaffected. These towers can ordinarily easily be constructed by the works engineer, and several types of gas distributing or "filling" material are available (see pp. 1198 to 1206). Such towers are also sold by manufacturers, but data on costs and performances are not available.

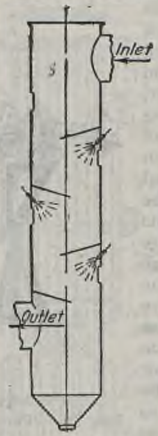


FIG. 12.—Baffle spray tower.

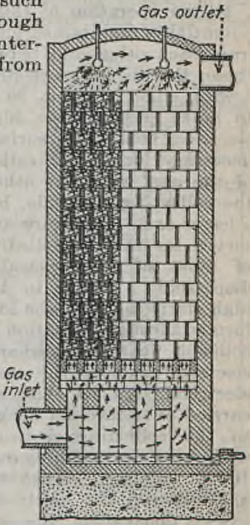


FIG. 13.—Packed spray tower.

"Wet" Baffle Chambers. The ordinary centrifugal baffle apparatus, even when so constructed as to have the needed centrifugal action, often fails with solid mechanical dispersoids because the separated dust is again redispersed. This is largely prevented by permitting a liquid to flow over the baffles, which thus serves the double purpose of retaining the separated dust and removing the same from the apparatus. Both in principle and in effect, such chambers practically are identical with the spray towers just described, though the gas path is usually simpler.

Similar in action is the so-called "filter" wherein the filter medium is merely a screen, or similar coarse material, which is coated with a dust-retentive oil.

Mechanical Washers. Mechanical washers are of two main types, one with relatively coarse spray where the mechanical feature consists in the mechanical dispersion of the spray in the path of the gas, and one where both gas and spray liquid are intimately dispersed together. The former, while effective on dust, is probably no more so than a properly designed spray chamber and is more expensive to operate. The latter is effective even on fine fume, but its power requirement is so high that it is usually not practical for ordinary mill or furnace gases. Thus for "fine" cleaning of blast-furnace combustible gas, the Theisen washer, (Fig. 14) is widely used.

The Electrical Precipitation Method. Although several inventors have been instrumental in bringing the art of electrical precipitation to its present stage, the three men who were chiefly responsible for the early pioneering work were Cottrell in America, Lodge in England, and Möller in Germany. On this account the method is known today as the Cottrell electrical precipitation method in America, Asia, and Australia; as the Lodge-Cottrell method in England; and as the Cottrell-Möller method on the continent (in Germany most often as E.G.R.—Elektrische Gas Reinigung).

In the operation of this method, a unidirectional electrical difference of potential is maintained and current set up between two electrodes insulated from each other and between which the gas is passed. Ordinarily one of these, the discharge electrode, is of small cross section and curvature, like a wire, edge, or point, so as to make possible the high electrical field at its surface necessary for the ionization of the gas, while the other, the collecting electrode, has a lesser or no curvature and serves for the precipitation of most of the separated dispersoids. Owing to the high electrical field, the ions formed through ionization by collision near the discharge electrode and of the same electrical sign as this, are carried over through the gas to the collecting electrode with velocities probably over 100 ft. per sec. When the gas between the electrodes carries dispersoids, the ions attach themselves to the particles which thus themselves become ions and are therefore also forced toward the opposing or collecting electrode. The velocity of these "particle ions" is, however, much less than that of the simple gas ions although undoubtedly considerably higher than ordinarily surmised.

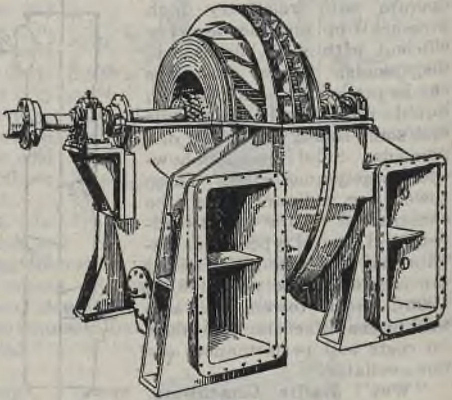


Fig. 14.—Theisen disintegrator gas washer. (*Freyn Engineering Co.*)

In the so-called "two-stage" or precharging method the first stage only is equipped with corona-forming discharge electrodes to produce the gas ions necessary for charging the dispersoids, while the second stage has non-discharging precipitating electrodes opposing the collecting electrodes, between which is maintained a corona-free electric field for precipitating the particle ions from the first stage.

In the ordinary so-called "single-stage" method, corona discharge is maintained throughout, which, because of the space charge of the gas ions, makes for high-intensity precipitating fields. This corona field also recharges neutralized or discharged particle ions and moreover exerts a pressure on the precipitated material so as to hold it on the collecting electrodes and prevent redispersion.

The corona discharge besides serving to ionize or charge the dispersoids and maintain the space charge may also produce a certain amount of chemical activity, such as ozone and nitric acid formation in air. The two-stage method with its minimum corona is therefore particularly useful in air cleaning where the dust concentration is low and where minimum chemical activity is desired.

The single-stage method is, however, far more generally applicable since for the ordinary concentration of dispersoids in industrial gases its corona precipitation field is necessary to prevent redispersion of the precipitated material at reasonable gas velocities.

Since the gas ions are molecular, they are smaller than the smallest fume or mist particles. The number of these gas ions ordinarily present in the gas in a precipitator is also probably a thousand-fold larger than the greatest possible number of dispersoid particles in that gas. Consequently the effectiveness of this process is not limited by the size, weight, or number of the dispersoid particles. Moreover, since the electrodes can be built from material to withstand both high temperatures and chemically active gases, the process is practically universally applicable to the separation of gas dispersoids.

The precipitation or collection efficiency of an electrical precipitator is a function of the time that the gas remains in the active field and can be made to approach 100 per cent as closely as desired. Since, however, the size and cost of the apparatus increase with the efficiency, an economic limit is necessarily set, which usually lies between 90 and 99.9 per cent.

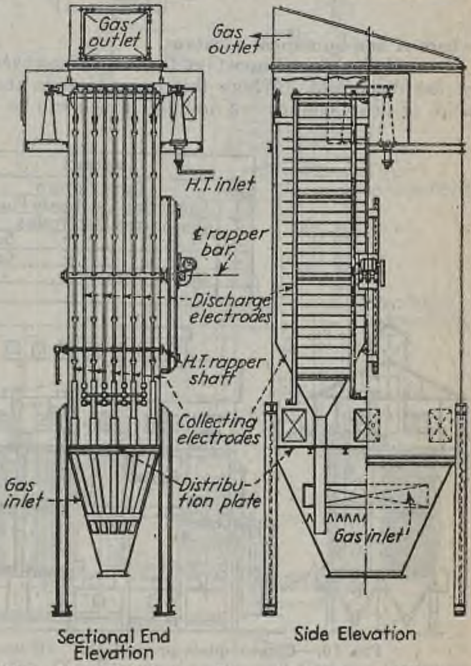


FIG. 15.—High-duty plate precipitator. (*Western Precipitation Corp.*)

The size of the precipitator for a given gas volume depends on the precipitation rate of the dispersoid in question in the type of precipitator used. This precipitation rate depends to a certain extent on the size of the particles and their characteristics, as, for example, their surface conductivity, since sufficient surface conductivity is necessary to permit the discharge of the dispersoid ions when reaching the collecting electrode.

For any given dispersoid of uniform size and character in a given precipitator, the precipitation efficiency E is related to the time t that the gas remains in the active field of the precipitator by the equation

$$\log (1 - E) = t \log K \tag{24}$$

where K is the so-called precipitation constant for that type of precipitator and dispersoid and is always less than 1, and may vary from 0.05 to 0.50 for conducting dispersoids and normal electrode spacing, the time t being measured in seconds.

The precipitation rate increases very rapidly with the voltage, between the electrodes, and this is therefore always maintained at a maximum just below the break-down point for the gas. For a given precipitator, dispersoid, and efficiency, the precipitator voltage V bears the following approximate relation to the precipitation constant

$$\log K = AV \tag{25}$$

where A is a numerical constant.

One of the most important factors is, however, the surface conductivity of the dispersoid. Where this is inadequate the addition of water vapor, acid, or other conducting material which can be absorbed on the dispersoid surfaces increases the surface conductivity and often increases the precipitation rate many fold [see Schmidt and Anderson, *Electrical Engineering*, **57**, 332 (1938)].

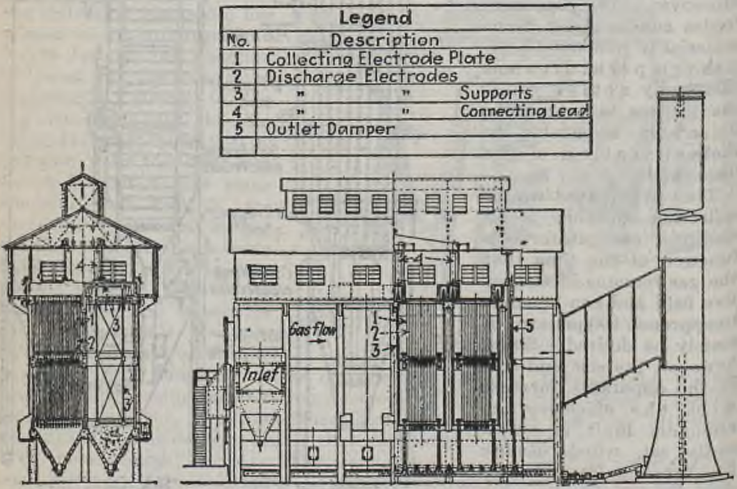


Fig. 16.—Cement-plate precipitator. (*Western Precipitation Corp.*)

Apparatus. Cottrell electrical precipitators may be divided into two main classes, the so-called **plate type** (Figs. 15 and 16), where the fundamental unit is the duct between two parallel plates, screens, or rows of rods, chains, or wires; and the so-called **pipe type** (Figs. 17, 18, and 20), where the unit is a pipe, which may be square, round, or any other shape. The discharge or precipitating electrodes are in each case wires or edges. These are ordinarily placed midway between the collecting electrodes or in the center of the pipe, and in the duct may be either parallel or perpendicular to the gas flow. Where the collecting electrodes are screens, or rows or planes of rods or wires, the gases are usually passed parallel to the plane but may also be passed through it.

Sectional or composite plate collecting electrodes permit continuous removal of the precipitated material and thus prevent redispersion. The high-duty plate precipitator in Fig. 15 is of this type.

Except where liquid dispersoids are being collected, or in case of film precipitators where liquid such as water is circulated over the collecting electrode surface, thus continuously removing the precipitated material, this material is dislodged from the electrodes either periodically or continuously by mechanical rapping or scraping.

Figures 19 and 16 show interior and elevation view of **rod-curtain** and **plate precipitators.** Figure 20 shows a two-stage, water-film, pipe precipitator for cleaning ventilating air at an ore-crushing plant, and Fig. 21 shows a two-stage plate precipitator used chiefly in connection with air conditioning.

The best spacing between the opposing electrodes is somewhat dependent on the actual density and character of the gas and the concentration and character of the dispersoid but is ordinarily from 3 to 6 in., although with two-stage precipitators, particularly where the dispersoid concentration is low, electrode spacings approximating ½ in. may be used. The smaller the spacing, the lower the necessary voltage, but mechanical difficulties in maintaining proper alignment, as well as disturbances from the precipitated material, are increased.

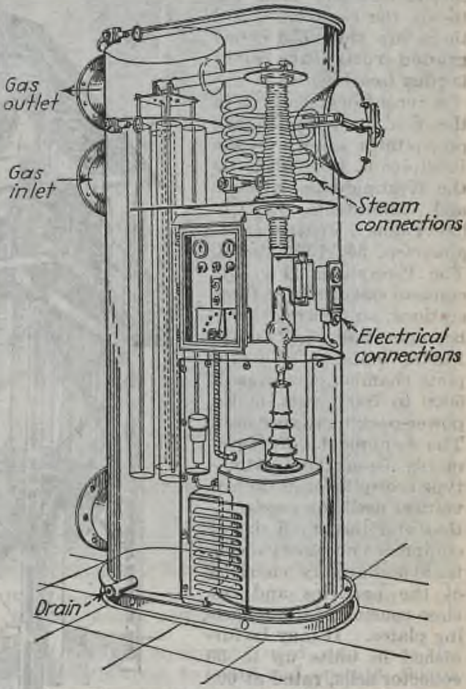


Fig. 17.—Pipe precipitator with electron-tube power unit. (*Research Corporation.*)

The choice of the electrode type depends on the character of the gas and dispersoids and to a certain extent on local conditions. It is almost entirely a matter of cost since equal efficiencies can be obtained with every type when properly dimensioned.

As already stated, the size of the precipitator for a given efficiency is proportional to the gas volume and depends on the type used and on the precipitation rate of the dispersoid. The proportions of the unit must, however, be so chosen that the gas velocities do not exceed the critical limit beyond which the precipitated material is again redispersed in the gas.

The electrodes are usually made from metal and most often steel. Other conducting material like carbon is also used where the gas conditions require it. Where the precipitated material is liquid so as to form its own conducting film, insulating material, such as glass, terra cotta, or wood, is also used.

Besides the ordinary highly conducting electrodes, special types constructed of material with fairly low conductivity, like concrete, are also used. In the latter, conducting rods are so placed as to tend to equalize the resistance throughout the field between the electrodes, and these are therefore called **graded resistance electrodes** (see Fig. 16).

A recent development in the field is a low-voltage precipitator known as the Precipitron, developed by the Westinghouse Electric and Manufacturing Co. [see Penney, *Electrical Engineering*, **56**, 159 (1937)]. The Precipitron (Fig. 21) consists essentially of three sections: an ionizing chamber operating at 12,000 to 14,000 volts, a collector plate chamber operating at 5000 to 7000 volts, and a power-pack rectifying unit. The equipment differs primarily from the Cottrell-type precipitator in the low voltage used for precipitation and the small size of equipment necessary owing to a preliminary charging of the particles and the close spacing of the collecting plates. It may be furnished in units up to 50 collector cells, rated at 600 cu. ft. per min. per cell. The over-all dimensions of a collector cell are approximately 8¼ in. wide, 19 in. high, and 36 in. long. The plate spacing is usually ⁵⁄₁₆ in., but for the higher dust concentrations a spacing of ½ in. is employed.

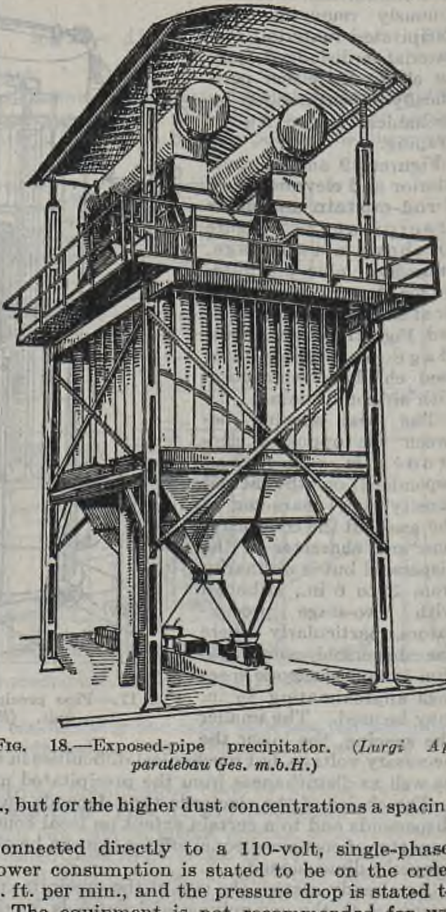


FIG. 18.—Exposed-pipe precipitator. (*Lurgi Apparatebau Ges. m.b.H.*)

The Precipitron can be connected directly to a 110-volt, single-phase, 25- or 60-cycle line. The power consumption is stated to be on the order of 30 to 80 watts per 1000 cu. ft. per min., and the pressure drop is stated to be less than 0.1 in. water. The equipment is not recommended for use at temperatures above 350°F. due to structural limitations. When it serves as a dust collector, dust loadings should not exceed about ½ grain per cu. ft. since at higher loadings the frequency of cleaning becomes excessive. Its chief field of application is in cleaning atmospheric air where a high degree of cleanliness is required, but it has been applied in collecting a number of industrial dusts of low electrical conductivity. The chief difficulties encountered in removing mists or dusts of high conductivity, such as sulphuric acid, are found in the possible short circuiting of the insulation. The Precipitron costs range from about \$0.25 to \$0.35 per cu. ft. per min. of gas handled for Cor-Ten steel construction.

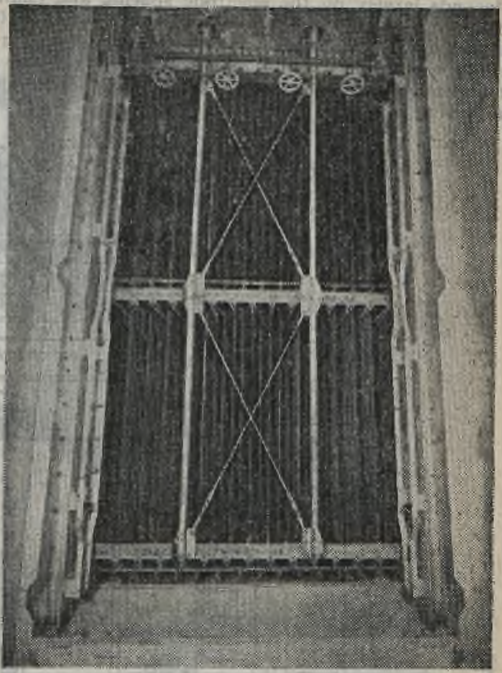


FIG. 19.—Interior view of rod-curtain type Cottrell electrical precipitator. (*Western Precipitation Corp.*)

Electrical Equipment. The equipment used to produce the strong electrical fields necessary consists ordinarily of a transformer (Fig. 22) which steps up the ordinary line voltage of 110 or 550 volts to from 50,000 to 75,000 volts, and a synchronous motor-driven mechanical rectifier with the necessary switchboard and regulating instruments. Electron, mercury-arc, and contact rectifiers are also used, but since the precipitation rate is but little affected by the type of rectifier equipment used, and since the mechanical rectifier is ordinarily the cheapest to build and operate, this is almost universally the standard equipment, except for the smaller size and for two-stage precipitators (see Figs. 17 and 21).

Table 7. The Relation between Electrode Spacing and Arcing Voltage

Diameter of pipe, in.	Spacing, in.	Arcing voltage, kv.	
		Peak	Root mean square
4	2	59	45
6	3	76	58
9	4.5	90	69
12	6	100	77

Voltage and Power Requirement. The voltage required in a precipitator depends largely on the electrode spacing and character of the gas and is directly related to the breakdown or arcing voltage for the gas and electrode arrangement. Table 7 shows the maximum arcing voltage from a central small wire to pipes of different diameters with gases at atmospheric pressure and temperature of about 100°F. and containing water vapor, air, and CO_2 as well as mist.

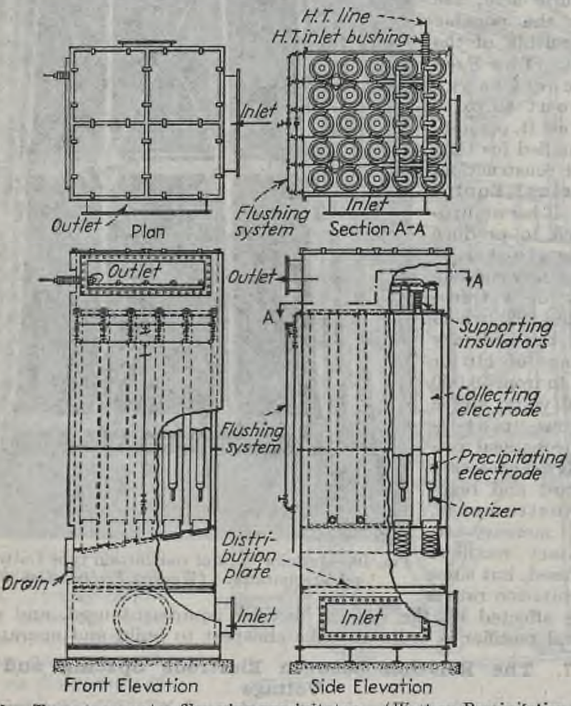


FIG. 20.—Two-stage water-film pipe precipitator. (*Western Precipitation Corp.*)

The maximum precipitator voltage varies approximately directly as the density of the gas (hence nearly inversely as the temperature) but is even more affected by the characteristics of the deposit, since even small amounts of poorly conducting material on the electrodes may markedly lower the arcing voltage. The data given in Table 7 are for the negative polarity of the discharge electrode. For positive polarity the arcing voltages are very much lower.

The power required in electrical precipitation is also very largely dependent on gas and dispersoid conditions but is in any event nearly proportional to the size of the precipitator. It is lower, however, for two-stage than for single-stage operation. Hence for dispersoids with low precipitating rates,

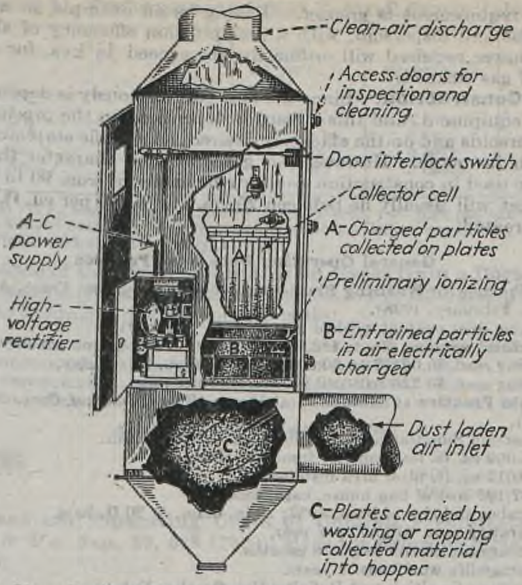


FIG. 21a.—Sectional view of Precipitron. (*Pangborn Corporation.*)

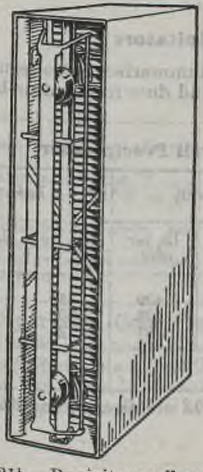


FIG. 21b.—Precipitron collector cell. (*Westinghouse Electric & Manufacturing Company.*)

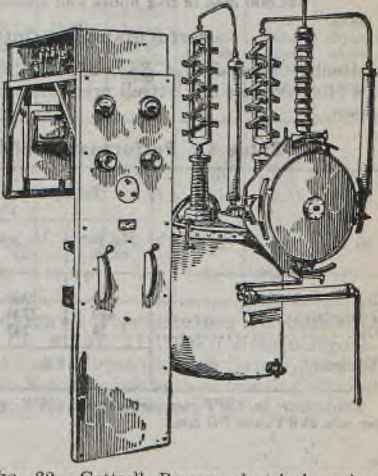


FIG. 22.—Cottrell Process electrical equipment. (*Western Precipitation Corp.*)

the power requirement is greater. Taking as an example an average condition of gas and dispersoid, with a precipitation efficiency of about 90 per cent, the power required will ordinarily not exceed ⅓ kva. for 1000 cu. ft. per min. of gas cleaned.

Cost of Construction. Since the first cost obviously is dependent on the size of the equipment, and this in turn is dependent on the precipitation rate of the dispersoïds and on the efficiency desired, no specific statement in regard thereto can be made. However, for gases of such character that ordinary steel can be used in construction and for efficiencies of from 90 to 98 per cent, the first cost will usually lie between 25 cts. and $1.00 per cu. ft. per min. of gas to be treated.

General Operating Data from Practice

1. German Practice for **Washing Blast-furnace Gas** (Boynton, *Trans. Am. Inst. Min. Eng.*, February, 1928).
    - Maximum dust content washed gas, 0.04 gr. per cu. ft.
    - Cost central washing plant, $12,500 per 100 tons of pig iron per day.
    - Operating cost, $0.00177 to $0.00283 per 1000 cu. ft. per day.
    - Operating cost, $0.225 to $0.40 per ton iron per day.
2. **Bag-house Practice** at Metallurgical Plants (Eilers, *Internat. Cong. Applied Chem.*, 1912).
    - *A.* American Smelting and Refining Co., Murray, Utah:
        - 165,000 cu. ft. lead smelter fume per minute.
        - 570,012 sq. ft. filter area installed.
        - $127,195 cost of bag house, bags, flues, etc.
        - Number and size of bags, 4032; 18 in. diam. × 30 ft. long.
        - Operating cost, $16,500 per year.
        - Average-life cotton bags, 18 months.
        - Average-life wool bags, 4 years.
    - *B.* American Smelting and Refining Co., Omaha, Neb.:
        - 124,000 sq. ft. filter area installed.
        - $42,000 cost of bag house and accessories.

Performance of Cottrell Precipitators

Healy [*Chem. & Met. Eng.*, **37**, 366–368 (1930)] summarizes some results on the effectiveness of Cottrell precipitators of mist and dust from sulfur-burner gases.

Table 8. Performance of Hot Cottrell Precipitator

	Total dust		$PbSO_4$		Fe_2O_3	
	Mg. per cu. ft.	Lb. per day	Mg. per cu. ft.	Lb. per day	Mg. per cu. ft.	Lb. per day
In	453	2960	96	620	357	2340
Precipitated	419	2720	82	529	337	2191
Out	34	240	14	91	20	149
Efficiency	91.8%		85.4%		93.7%	

Temperature in, 820°F.; temperature out, 655°F.; pressure drop, 0.2 in. H_2O; gas volume, 2060 cu. ft. per min. at 0°C. and 760 mm.

Table 9. Performance of Cold Cottrell Precipitator

	H_2SO_4, mg. per cu. ft.	As_2O_3, mg. per cu. ft.	$PbSO_4$, mg. per cu. ft.
In	168.0	5.2	7.75
Out, primary	4.2	0.13	0.019
Out, secondary, No. 1	0.23	0.0023	0.0031
Out, secondary, No. 2	0.015	0.00013	0.0007

Temperature in, 85°F.; temperature out, 70–95°F.; pressure drop, 1 in. H_2O; gas volume, 2800 cu. ft. per min. at 0°C. and 760 mm. The power consumption of the hot Cottrell was 3.5 kw.; that of the cold Cottrell was: primary, 0.8 kw.; and secondary, 3.2 kw.

Performance of Cottrell Precipitator on Gases from a Doré Furnace. Clark and Heimrod [*Trans. Electrochem. Soc.*, **61**, 77 (1932)] give the following results of the treatment of Doré furnace gases:

Gas volume	3200 cu. ft. per min. at 125°F.	
Inlet concentration mg. per cu. ft.	Gold, 0.013	Silver, 0.913
Outlet concentration, mg. per cu. ft.	Gold, 0.0002	Silver, 0.0061
Collection efficiency, per cent	Gold, 98.3	Silver, 99.3

Analysis of Collected Material

Cu	15.51%
Ag	2100.8 oz. per ton
Au	483.9 oz. per ton

Investment and Operating Costs of Cottrell Precipitators. Landolt [*Chem. & Met. Eng.*, **29**, 588 (1923)] gives the following table for typical installations:

Table 10

	Metallurgical fume	H_2SO_4 mist concentration	H_2SO_4 mist, SO_2 gases	Boiler-house gases (powdered coal)	Zinc roaster gases	Producer gases (detarring)	Spray-drying solids recovery
Investment per rated cu. ft. per min.	\$0.56	\$1.80	\$3.75	\$0.11	\$3.00	\$1.28	\$0.50
Temperature gases treated, °F.	500	250	70	300	1200	100	150
Cleaning cost per 100,000 cu. ft.	\$0.05	\$0.09–0.10	\$0.18	\$0.01	\$0.16	\$0.11	\$0.05
Power consumption, kw-hr. per 100,000 cu. ft. per hr.	0.75	1.00	0.67	0.25	0.55	0.88	0.56
Efficiency of removal of suspended matter, per cent	90–95	98	99.9+	90	95	95–98	95

Comparative Cost Figures for Cottrell Precipitators on Cement-kiln Gases. Schmidt [*Trans. A. I. Ch. E.*, **21**, 11 (1928)] gives comparative approximate average figures for different types of precipitators, treating cement-kiln gases, in the following table, made on the basis of treating 100,000 cu. ft. of gas per minute at 90 per cent collection efficiency:

Table 11

Type of precipitator	Space required, cu. ft.	Approx. cost dollars	Operating power, kw.	Labor, man-hr. per day
Iron pipe electrodes	57,000	$100,000	30	48
Iron plate electrodes	50,000	80,000	20	24
Graded resistance electrodes	35,000	55,000	10	12

AIR FILTERS

By C. E. Lapple

The equipment previously described is intended primarily for the treatment of process dusts. Air filters are employed in the elimination of atmospheric dust. The difference in application is not so much one of quality of dust as it is of quantity. Process dust concentrations may run as high as several hundred grains per cubic foot, although usually not exceeding 20 grains. Atmospheric dust concentrations are generally below 5 grains per 1000 cu. ft. The following table gives average atmospheric dust concentrations that may be expected in various districts:

District	Dust Concentration, Grain per 1000 Cu. Ft.
Rural or suburban	0.2–0.4
Metropolitan	0.4–0.8
Industrial	0.8–1.5
Dusty factories or mines	4.0–80.0

(1 grain per 1000 cu. ft. = 2.3 mg. per cu. m. = 0.065 mg. per cu. ft.)

In the elimination of atmospheric dust, no attempt to recover the dust is usually made. Air washers may also be employed for cleaning air, but these are installed primarily for humidifying or cooling the air and dust removal is only of secondary importance.

Air filters may be classified into three groups on the basis of the type of filter medium employed: viscous, dry, or automatic.

Viscous filters are so called because the filter medium is coated with a viscous material to catch and to retain the dust. The filters are supplied in units of convenient size (generally of the order of 20 by 20 in. face area) to facilitate installation, maintenance, and cleaning. Each unit consists of an interchangeable cell or replaceable filter pad and a substantial frame which may be bolted to the frames of other similar units to form an air-tight partition between the source of dusty air and its destination (Fig. 23). Felt liners are sometimes used to make the assembly of individual cells air-tight. Some types of cells may not have a separate cell frame but are clamped directly to the superstructure.

Typical commercial viscous filter units are shown in Fig. 24. The filter pad may consist of one of a wide variety of materials, including glass fibers, animal hairs, wood shavings, corrugated fiberboard, split wire, or metal screening. These are coated with a dust-collecting liquid, such as mineral oil, and chemicals of high viscosity and flash point to act as a dust holder.

In the matter of servicing or reconditioning, the viscous-type filters fall into two classes. With most units employing a metallic medium, the unit cells are taken out and washed or steamed, reoiled, and then placed back into service. With the other type the cell or cell pad is discarded, once the maximum allowable dust load has accumulated, and is replaced by a new one.

Dry filters are supplied in units similar in size to the viscous filters, except that the depth is usually greater. The various filter media used have, as a rule, smaller passages for air flow than the viscous media, and hence lower

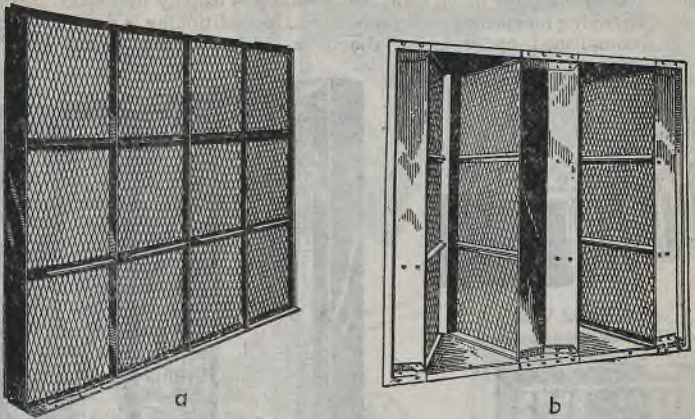


FIG. 23.—Typical types of filter-bank installation. *a*, flat or L-type installation. *b*, V-type installation.

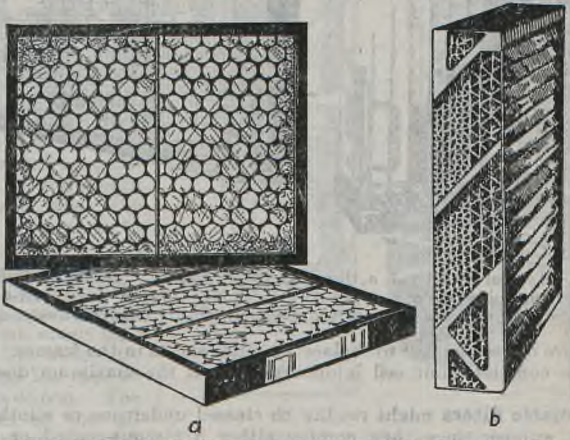


FIG. 24.—Typical viscous unit filters. *a*, throwaway type, Dustop. (*Owens Corning Fiberglas Corporation.*) *b*, cleanable type, Air-Maze Type B, cutaway open end view. (*Air-Maze Corporation.*)

air velocities must be used in order that the pressure drop will not be excessive. This low velocity necessitates a relatively large filter surface to handle a given gas volume and the filter media are usually arranged in the form of pockets to bring the frontal area within customary space requirements.

Typical dry filters are shown in Fig. 25. The filter media are generally sheets of cellulose pulp, cotton, felt, or spun glass. Filters using felt or similar materials are generally reconditioned by vacuum or dry cleaning. Where the air contains much soot, dry cleaning is usually necessary. With filters employing inexpensive cellulose media, reconditioning is most economically accomplished by replacing the filter medium. Mechanical loading devices are often supplied to replace the filter sheets in the frames. In some cases the complete unit cell is discarded when the maximum dust load is reached.

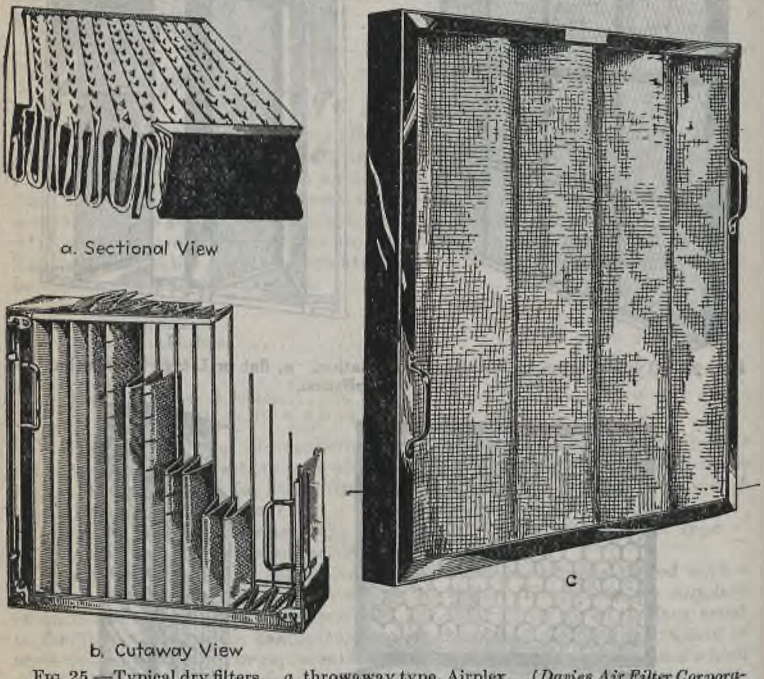


Fig. 25.—Typical dry filters. *a*, throwaway type, Airplex. (*Davies Air Filter Corporation.*) *b*, replaceable medium type, Airmat PL-24, cutaway view. (*American Air Filter Company.*) *c*, cleanable type, Amirglass Sawtooth. (*Amirton Company.*)

Automatic filters might readily be classed under one or another of the previous groups, since they employ either a viscous-coated or dry-filter medium. They form a distinctive group in the air-filter field, however, in that the cleaning operation is essentially continuous and automatic. Most commercial automatic filters are of the viscous type and consist of perforated, crimped, or woven metallic screens in series (see Fig. 26). The apertures are graded so that the air first meets the larger openings and is subjected to the finest filtering action just before it leaves. The screen curtains are drawn around in a vertical direction, either continuously or intermittently. The oil bath serves to rinse out the dust and coat the screen with a fresh film of oil. The dust is then allowed to settle out as a sludge in the bottom of the hopper. Such filters may be furnished with a hand crank or motor drive as desired.

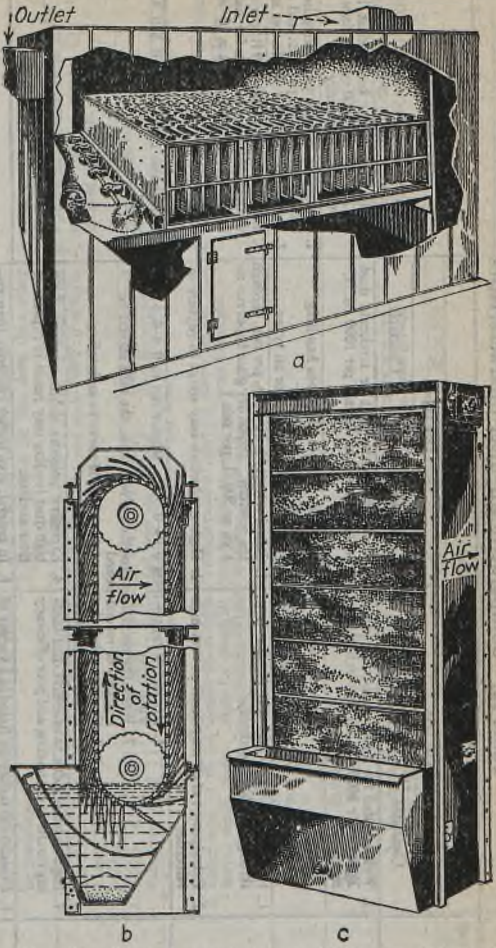


FIG. 26.—Typical automatic air filters. *a*, dry type, Airmat Dust Arrestor. (*American Air Filter Company.*) *b*, Multi-Panel. (*American Air Filter Company.*) *c*, Staynew Model A. (*Staynew Filter Corporation.*)

The Airmat Dust Arrestor (Fig. 26*a*) is a dry automatic filter. It can, however, only be considered automatic when applied to dusts that are relatively nonsticky and easily shaken off. It is also used as a dust collector rather than an air filter since it can handle relatively high dust loads. The air flow must be stopped or diverted, however, when the filter is vibrated.

General Design and Performance of Air Filters. The characteristics of the various types of air filters are compared in Table 12. The pressure drop through a filter increases as dust accumulates. The filter should be replaced when the pressure drop starts to increase rapidly or else the air capacity will decrease. The maximum allowable pressure drop ranges from about 0.20 to 0.50 in. water depending on the type of filter medium. The cleanliness of the filtered air may also suffer if cleaning or replacement of dirty air filters is neglected. The dust loading of the air handled generally determines the life of unit filters. For this reason automatic filters become increasingly attractive as the dust concentration of the air to be cleaned increases, since dust capacity is not usually an

Table 12. Comparative Air-filter Characteristics

	Unit filters				Automatic filters
	Viscous type		Dry type		
	Cleanable	Throwaway	Throwaway	Cleanable	
Dust capacity	1. Well adapted for heavy dust loads (up to 2 grains per 1000 cu. ft.) due to high dust capacity		1. Well adapted to light or moderate dust loads of less than 1 grain per 1000 cu. ft.		1. Well adapted for heavy dust loads (> 2 grains per 1000 cu. ft.) since it is serviced automatically
Filter size	1. Common size of unit filter is 20- × 20-in. face area handling 800 cu. ft. per min. at rated capacity 2. Face velocity is generally 300–400 ft. per min. for all types				1. Automatic viscous units supplied to handle 1000 cu. ft. per min. and over 2. Face velocity is 350–750 ft. per min.
Air velocity	1. Rated velocity is 300–400 ft. per min. through the filter medium 2. Entrainment of oil may occur at very high velocities		1. Rated velocity is 10–50 ft. per min. through the medium. (Some dry glass types run as h gh as 300 ft. per min.) 2. Higher velocities may result in rupture of filter medium		1. Rated velocity is 350–750 ft. per min. through the filter medium for viscous types. For dry types, it is 10–50 ft. per min.
Resistance	1. Resistance ranges from 0.05–0.30 in. when c'ean to 0.4–0.5 in. when dirty 2. When the resistance exceeds a given value, the cells should be replaced or reconditioned 3. Cycling cells 'n large installations will serve to maintain a nearly constant resistance				1. Resistance runs about 0.3–0.4 in. water
	4. High resistance due to excessive dust loading results in channeling and poor efficiency		4. Excessive pressure drops resulting from high dust loading may result in rupture of filter medium		
Efficiency	1. Commercial makes are found in a variety of efficiencies, these depending roughly on filter resistance for similar types of media 2. Efficiency decreases with increased dust load and increases with increased velocity up to certain limits		1. In general, give higher efficiency than viscous type, particular y on fine particles 2. Efficiency increases with increased dust load and decreases with increased velocity		
Operating cycle		1. Well adapted for short-period operations (less than 10 hr. per day) due to relatively low investment cost			1. Well adapted for continuous operation
	2. Operating cycle is 1–2 months for general "average" industrial air conditioning		2. Operating cycle is 2–4 weeks for general "average" industrial air conditioning		
Method of cleaning	1. Washed with steam, hot water, or solvents and given fresh oil coating	1. Filter cell replaced. Life may in some cases be lengthened by shaking or vacuum cleaning but this is not often successful		1. Vacuum cleaned, blown with compressed air, or dry cleaned	1. Automatic. Filter may clog in time and cleaning by blowing with compressed air may be necessary

Space requirement	*1. Well adapted for low head-room requirements* *2. Form of banks can be chosen to fit any shaped space* *3. Space should be allowed for a man to remove filter cells for cleaning or replacement*				*1. Have a high head-room requirement* *2. Take up less floor space than other types*
	4. Requires space for washing, reoiling, and draining tanks			4. Requires space for mechanical loader in some cases	
Type of filter medium	1. Crimped, split, or woven metal, glass fibers, wood shavings, hair—all oil coated		1. Cellulose pulp, felt, cotton gauze, spun glass 2. Dry medium cannot stand direct wetting. Oil impregnated media are available to resist humidity and prevent fluff entrainment		1. Metal screens, packing, or baffling
Character of dust	1. Not well suited for linty materials			1. Not well suited for handling oily dusts	1. Not suited for linty material if of viscous types
	2. Well adapted for make-up air and granular materials		2. Well adapted for linty material 3. Better adapted for fine dust than other types		
Temperature limitations	1. All metal types may be used up as high as 250°F. if suitable oil or grease is used. Those utilizing cellulosic materials are limited to 180°F.		1. Limited to 180°F. except for glass types which may be used up to 700°F. if suitable frames and gaskets are used		1. Viscous may be used up to 250°F. If suitable oil is used. Dry type limited to 180°F.
Initial cost	1. Higher first cost than throwaway	1. First cost is relatively low. Frames are generally permanent but cells are replaced		1. Higher first cost than throwaway	1. Highest first cost of all
Operating cost	1. Power costs are comparable for all unit-type filters				1. Power costs are somewhat higher for automatic types
	2. Labor cost to remove, clean, and replace the cells is comparable with the cost of replacement of throwaway-type cells for medium-sized installations of 10,000–50,000 cu. ft. per min.	2. Replacement costs are comparable to labor costs for cleanable types for medium-sized installations (10,000–50,000 cu. ft. per min.)		2. Same as for Viscous Cleanable	2. Little labor required to inspect, replace oil, or hand crank filter
	3. Maintenance exclusive of filter cells and depreciation are roughly comparable and vary more between different commercial makes in these groups than between different types				3. Depreciation and maintenance costs are the highest for automatic filters
Purchase cost (including frames), cost per 1000 cu. ft. per min.	$15 to $30	$6 to $10	$5 to $10 for complete throwaway unit $8 to $30 for units with replacable medium	$5 to $50	$50 to $100

important item with such filters. The operating cycle or life of a unit filter may be estimated from the following formula:

$$t = 500{,}000\left(\frac{L}{EQC}\right) \tag{26}$$

where t = length of operating cycle, days.
L = dust capacity per cell, lb.
Q = air volume handled per cell, cu. ft. per min.
C = dust concentration in air to be filtered, grain per 1000 cu. ft.
E = filter efficiency, per cent.

This equation is based on a material balance of dust entering and leaving the filter cell. Average dust concentrations encountered in various types of localities were given previously. Dust capacities for unit filters generally range from 0.5 to 4.0 lb. per standard 20- by 20-in. unit. In general, the life of viscous-type filters handling average city air may range from 2 to 5 months, while for dry-type filters it will range from 1 to 3 months. For the average chemical plant the life will be from one-half to one-third that of the same filter handling "average" city air and may at times be considerably less due to the higher dust loadings involved.

The matter of a proper schedule for servicing filters cannot be too strongly stressed if satisfactory operation is to be obtained. The over-all time cycle for reconditioning washable viscous filters is generally about 24 hr. Unless they are allowed to drain sufficiently, an entrainment of oil by the filtered air may result.

Ordinary air filters will have a negligibly low efficiency in separating particles below 0.5μ in diameter. For particle sizes over 10μ the filter efficiency for most makes is generally over 85 per cent. An oil impregnation of dry-filter medium has also proved useful in eliminating any possibility of lint carry-over from the medium itself. Many of the dry media have also been fireproofed by suitable treatment.

Power costs may be computed by the following approximate formula with reasonable accuracy (based on an over-all fan-motor efficiency of 63 per cent):

$$\text{H.p.} = \frac{\text{cu. ft. per min.} \times \text{resistance (in. water)}}{4000} \tag{27}$$

The additional power required for the operation of automatic filters is proportionately very small and may be neglected.

The approximate purchase cost ranges for commercial steel filters are given in Table 12. For special service, filters may be obtained of stainless steel, galvanized iron, cadmium-plated steel, brass, or aluminum.

Miscellaneous Air-filter Equipment. A special type of equipment applicable to air filtering is one employing a water spray over glass fibers. An example of this is the Fleischer capillary conditioner. A recent development in the field is the utilization of an electrostatic field in addition to the usual filtering medium in order to secure a greater degree of air cleanliness. The Electromatic air filter (American Air Filter Co.) is an automatic viscous-type filter employing an electrostatic field, and Annis (Glendale, Calif.) has developed a felt filter employing such a field. See also Precipitron, p. 1872.

SECTION 16

CRUSHING, GRINDING, PULVERIZING, AND OTHER METHODS OF COMMINUTION

BY

S. B. Kanowitz, B.S., Chemical Engineer, Raymond Pulverizer Division, Combustion Engineering Company, Inc., New York; Member, American Association for the Advancement of Science, American Chemical Society, American Institute of Mining and Metallurgical Engineers

Allan P. Colburn, M.S., Ph.D., Acting Head, Division of Chemical Engineering, University of Delaware, Newark, Del.; Member, American Chemical Society, American Institute of Chemical Engineers

R. V. Kleinschmidt, A.B., A.M., S.B., S.D., Physicist and Research Engineer, Arthur D. Little, Inc., Cambridge, Mass.; Member, American Society of Mechanical Engineers, American Physical Society; Fellow, American Association for the Advancement of Science

H. G. Houghton, B.S., S.M., Assistant Professor of Meteorology, Massachusetts Institute of Technology, Cambridge, Mass.; Member, Physical Society, American Meteorological Society; Fellow American Association for the Advancement of Science

S. L. Godshalk, B.S., Ch.E., Chemist, E. I. du Pont de Nemours & Co., Inc. (Finishes Division), Philadelphia

Lawrence H. Bailey, B.S., M.S., Chief Engineer, F. J. Stokes Machine Co., Philadelphia

D. J. Van Marle, Ch.E., Chemical Engineer, Buffalo Foundry and Machine Co., Buffalo, N. Y.; Member, American Chemical Society

CONTENTS

PAGE

CRUSHING, GRINDING, AND PULVERIZING

OTHER METHODS OF COMMINUTION

CRUSHING, GRINDING, AND PULVERIZING*

BY S. B. KANOWITZ

THEORY OF CRUSHING AND PULVERIZING

Size reduction of materials of various kinds is an important operation in many industries. Machinery used for this purpose may be divided into three classes, depending upon the nature of the force applied. In the first class, breaking is accomplished by continuous pressure; this class includes equipment used in coarse crushing. In the second class, reduction is effected by blow or impact, and in the third by grinding or abrasion; the latter classes include machinery used in fine crushing and grinding.

Reduction of a material may be accomplished by crushing and splitting, or by cutting and tearing action. The ability of a material to withstand reduction generally depends upon its hardness, but the manner in which it is fractured is frequently an important factor. A foliated talc, or a flake graphite, is easily reduced to a certain size beyond which further reduction is accomplished only with great difficulty. Other factors affecting the grinding characteristics of a material are water of combination, hygroscopicity, tendency to agglomerate, combustibility, and sensitiveness to changes in temperature. Glauber's salt, for instance, gives off water of crystallization at a comparatively low temperature, causing clogging of equipment if the temperature rises above a certain point; calcium chloride is so hygroscopic that it may actually dissolve in the moisture absorbed; most barium compounds have a tendency to agglomerate, even when absolutely dry; synthetic resins and gums become soft and plastic beyond a certain temperature; other materials may burn or char; certain chemicals and dyestuffs are unstable and may ignite or explode if the temperature is excessive; many mineral pigments, such as ochres and siennas, tend to change in color at high temperature.

The main object in most pulverizing operations is to obtain a product smaller than a certain specified size; the higher the percentage of impalpable powder contained, the more desirable the finished product. This is particularly true for a product finer than 100 mesh or 200 mesh. Where a coarser product is required, a minimum percentage of fines is usually desired. The type of pulverizer selected plays an important part in this respect. A set of rolls will produce a material with less fines than a hammer mill grinding to the same maximum particle size. Another factor of importance, which is affected by the type of pulverizer used, is the particle shape. It will be found that in the case of certain mineral pigments, such as the oxides of iron, a product of a richer color will be obtained when the pigment is ground in one type of mill than in another.

As crushing and pulverizing are primarily operations of size reduction, determination of size and particle shape of feed as well as of the finished product is of primary importance in the efficiency of the operation. Three methods of particle-size determination are in use to-day, sieving, elutriation, and microscopic examination. The special ranges of size to which each of these methods is applicable are the following: (*a*) Sieving is rapid and accurate to as low as 74μ. Separation is determined by the minimum cross section of the particles which pass through all screens having apertures greater than

---

* This section is a condensation of material now in preparation for a detailed treatise on Crushing, Grinding, and Separating.—S. B. Kanowitz.

the minimum cross sections. (*b*) Elutriation has its greatest usefulness from 5 to 100μ. Within that interval separation is supposed to follow Stokes's law. The diameter may be calculated as that of a spherical particle which has the same settling velocity as the actual particle, or it may be measured microscopically. (*c*) Microscopic sizing has been developed for the range of a few tenths of 1μ to about 10 or 20μ; but, with care, it may be used up to 100μ or more. The particle diameter may be taken in several ways, but the least diameter is usually missed because flat particles generally lie with their minimum thickness in a vertical direction. (For more complete data on sieving and elutriation, see Sec. 15, page 1728.)

A great deal of confusion exists in comparing silk bolting cloth with testing sieves. The size of opening in silk bolting cloth of a certain mesh is not the same as in a sieve of the same mesh. It will often be found that there is no standard testing sieve corresponding to a particular screen opening as it falls between two standard screens.

Work expended in overcoming friction and inertia of machine parts, in rubbing between particles and between particles and crushing surfaces, in producing deformation without breaking, and in generating electricity in the crusher is not "useful" as the word is employed in the discussion of efficiency. An indication of useful work can, however, be reached by measuring the reduction in size of particles in the crushing operation and attributing thereto certain energy values in arbitrary units. When this is done, a statement of relative efficiency can be made as follows:

$$E = \frac{WT}{P}$$

where E = relative mechanical efficiency; W = number of units of size reduction per unit weight of material crushed; T = number of units of weight of material crushed per unit of time; and P = number of units of power used.

Opinion is divided as to the method to be employed in assessing the equivalence of useful work and size reduction. Two methods have been proposed. These are named respectively Kick's method and Rittinger's method from their proponents. In the former the energy required is proportional to the volume reduction of the particles. To each size of material is assigned an ordinal number, representing the work expended in arriving at that size. The energy spent on a given sample is taken as proportional to the sum of the products of the weight percentages retained on the various screens and the ordinal numbers corresponding to these screen sizes. The useful work is taken equal to the total energy expended upon the final product minus the energy expended on the feed. The basis for the method is given by Stadler in *Trans. Inst. Min. Met.*, **19**, 478 (1909–1910).

Rittinger's law states that the work required is proportional to the new surface formed. Much controversy has taken place regarding the correctness of the two methods. From a theoretical point of view both leave much to be desired. For crushing large particles Kick's law appears to give the better results, whereas finer materials conform more nearly to Rittinger's law.

The hardness of a material as measured by the Moh scale is not always a criterion of its resistance to crushing or even pulverization, unless the material is to be pulverized very fine. It is, however, a fairly good indication of the abrasive character of the mineral, a factor which determines the wear and tear on the grinding mediums. Arranged in increasing order of hardness the scale is as follows: 1, talc; 2, gypsum; 3, calcite; 4, fluorite; 5, apatite; 6, feldspar; 7, quartz; 8, topaz; 9, corundum; 10, diamond.

TYPES OF CRUSHERS AND PULVERIZERS

Gyratory Crushers (Fig. 1*a*). The housing of a gyratory crusher has the shape of the frustums of two cones placed together with the narrow sections in the center. An eccentric at the lower end gives the shaft a gyratory motion; this causes the head to approach or recede from the concave surfaces, breaking the feed on its downward path. The three general types are the suspended-spindle type, the most widely used; the supported-spindle type, an uncommon design; and the fixed-spindle type, rapidly growing in favor. The first two are usually designated by numbers, about one-half the feed opening (in inches); in the third type a number equal to the gape is used. In addition, the first type is designated by the letter K, the second by the letter D; the length of the receiving opening is usually designated by the circumferential distance (along the outer edge of the feed opening) between adjacent faces of the spider arms multiplied by the number of spider arms; this length is about eight times the gape in lever-type crushers below size 9, seven times in larger sizes. A No. 7-D crusher is of the supported-spindle type with 14- by 112-in. feed opening; a No. 20-K crusher is of the suspended-spindle type with 40- by 280-in. feed opening.

Among the common types is the *Austin* gyratory, which is made in six standard sizes, with hourly capacities from 10 to 150 tons; feed opening from 8 by 24 to 19 by 57 in.; normal ring size 2¼ to 4 in., and smaller ring size 1 to 3 in. The *Superior-McCully* is made in nine standard sizes, 15 to 25 to 300 to 500 h.p., all equipped with two feed openings from 8 by 34 to 60 by 182 in. and it has a range of output from 25 to 1420 tons per hr. for finest setting, 1½ to 6½ discharge opening, and 47 to 1900 tons for coarsest setting, 2½ to 10½ in. Secondary reduction is made on a fine-reduction McCully in three standard sizes, with feed openings 6 by 40, 10 by 52, and 18 by 68 in., respectively, and with the following hourly capacities: 24 to 69 tons, 50 to 75 h.p.; 100 to 214 tons, 100 to 150 h.p.; and 310 to 735 tons, 150 to 200 h.p. Discharge openings range from ½ to 4 in.

The *Traylor Bulldog* gyratory is used for primary and secondary crushing. Fourteen standard sizes are made with receiving openings varying from 2¼ by 14 in. to 72 by 242 in.; primary capacity, for ½- to 12-in. ring sizes from ¾ to 3400 tons per hr.; for ⅜- to 9-in. ring sizes from ½ to 2500 tons per hr.; secondary capacity from ¾ to 700 tons for ½- to 7-in. ring sizes, and ½ to 370 tons for ⅜- to 4¼-in. ring sizes. The Traylor finishing gyratory type T comes in five sizes, 25 to 200 h.p., with 4½- by 27½-in. to 20- by 80-in. receiving opening. Hourly capacities for ring sizes 1 to 2¾ in. range from 23 to 310 tons; for ring sizes ½ to 2 in., 6 to 225 tons.

The *Allis-Chalmers Newhouse*, used mainly for secondary crushing, has no foundation but is suspended by three cables which absorb much of the vibration. If necessary, it may be mounted on a foundation with spring cushions under the base. Operating characteristics for four sizes of this type are: ring size, 5, 7, 10, and 14 in. respectively; discharge openings, 3/16 to 1½, ¼ to 2, ⅜ to 2½, and ½ to 3 in.; hourly output, 10 to 44, 35 to 134, 60 to 200, 130 to 440 tons; motors 30, 60, 100, and 200 h.p.

The *Allis-Chalmers type R crusher* is a new development for fine crushing. It employs a built-in hydraulic jack to control the vertical position of the mantle relative to the concave ring, and a relief valve permits tramp iron to pass through without harm to the crusher. A feed up to 2½ in. in size can be reduced to a size from less than ¼ in. to less than ¾ in. at rates of 20 to 33 tons per hr., while with coarser products the rate may run up to 46 tons per hr. A power input of 25 to 30 h.p. is required for the only size available.

The *Kennedy-Van Saun gearless gyratory* is driven by a built-in synchronous motor or by belt to a pulley on the vertical shaft. The standard crushers of this type are available with feed-opening sizes from 3 by 8 in. to 66 by 235 in., crush to from ½ to 10 in. at ½ to 3600 tons per hr., and require from 1 to 250 h.p. The fine crushers are built with feed openings from 1¾ to 14 in. wide.

The fixed-spindle or pillar-shaft gyratory, known as the *Telsmith Breaker*, has a rigid shaft which does not rotate or gyrate; the full stroke is exerted on the largest particles as they enter the bowl, thus speeding up the crushing. The main advantages are rigid shaft eliminating breakage, compact and rugged construction, decreased head room, and effective increased receiving opening. Seven standard sizes are made, from 15 to 20 h.p. to 100 to 125 h.p., and receiving opening from 6¾ by 35 in. to 25 by 106 in. Hourly capacities range from 17 to 18 tons for the smallest size to 300 to 350 tons for the largest, discharge opening from 1 to 4 in. The Telsmith reduction or secondary crusher has a large bowl, much wider at the bottom than at the top, allowing free escape of the material. Four standard sizes are made, 25 to 65 h.p., 5- to 8-in. feed opening, ⅞- to 1⅝-in. discharge opening, with hourly capacities ranging from 18 to 21 to 85 to 100 tons.

Jaw Crushers. These may be divided into three groups: the *Blake*, with movable jaw pivoted at the top, giving greatest movement to the smallest lumps; the *Dodge* (Fig. 1*e*), with the movable jaw pivoted at the bottom, giving greatest movement to the largest lumps; and modifications of the two, giving nearly equal movement to all sizes. The Blake has a removable crushing plate, usually corrugated, fixed in a vertical position at the front end of a hollow rectangular frame. A similar plate, at a suitable angle, is attached to a swinging lever (movable jaw) suspended from a shaft resting in the sides of the frame. Movement is accomplished through a knuckle action by the rising and falling of a second lever (pitman) carried by an eccentric shaft. The vertical movement is communicated horizontally to the jaw by two plates (toggles).

The *Traylor type A Blake*, in 11 standard sizes, from 7 to 75 h.p., with feed opening from 7 by 10 to 24 by 36 in., has an hourly output ranging from 6 to 53 tons with 2- to 5-in. ring size, and 2.5 to 41 tons for 1- to 3-in. ring size. Type B, in four sizes, 75 to 130 h.p., with feed openings from 15 by 60 to 30 by 60 in., ranges from 94 to 180 tons per hr. for 4- to 6-in. ring size, and 54 to 125 tons for 2- to 4-in. ring size. The *Traylor type H crusher* is distinguished by a welded-plate construction, making it lighter in weight, and curved jaw plates, believed to be more efficient for secondary and fine crushing. It is available in sizes from 10- by 16-in. to 24- by 36-in. openings with capacities varying from 10 to 170 tons per hr. when producing minus ⅞-in. to minus 4½-in. stone. The maximum power requirements of the sizes are from 15 to 75 h.p. The more rugged *Bulldog* type, in seven sizes, from 80 to 250 h.p., with feed opening from 28 by 36 in. to 56 by 72 in., crushes from 105 to 535 tons per hr. for 5- to 10-in. ring size, and 44 to 310 tons for 3- to 7-in. ring size.

The Dodge has been found impractical in sizes above 11 by 15 in. It will crush larger pieces than any breaker of similar size and gives a more uniform product than a Blake or a gyratory. Typical performance data for 4-, 7-, and 9-h.p. *Traylor Dodge* crushers with 6 by 9-in., 7 by 11-in., and 11 by 15-in. openings are 3 to 5, 6 to 8, and 10 to 20 tons per hr., respectively, crushing to a 1½-in. product.

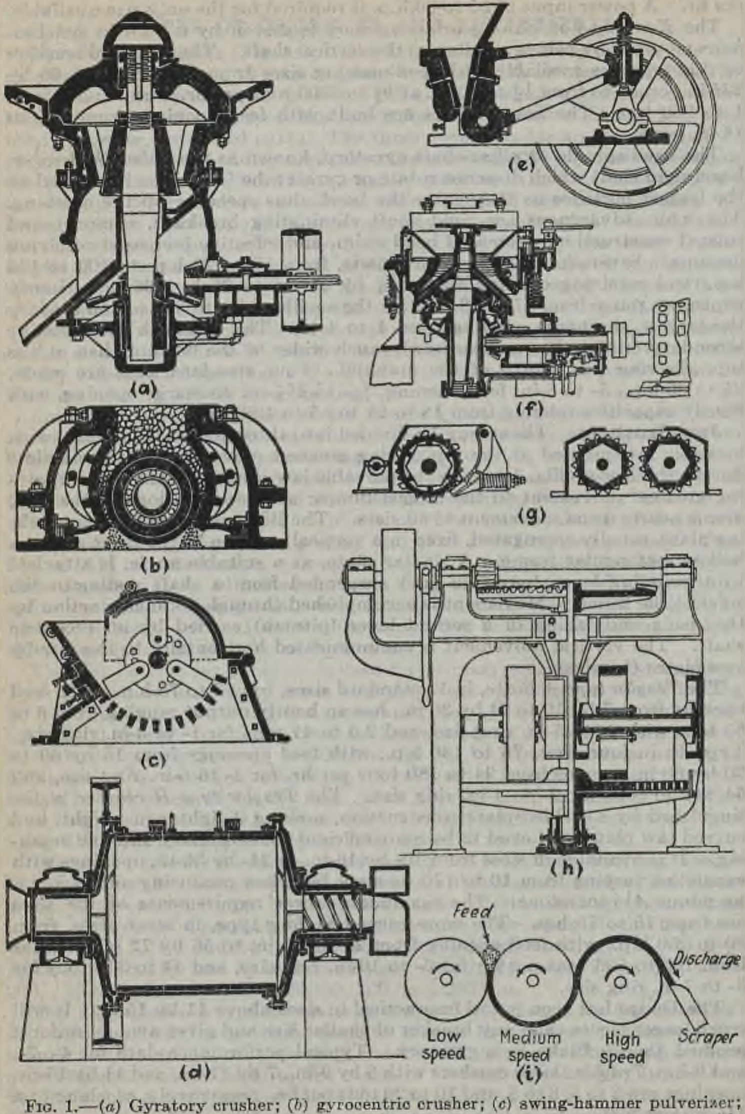


FIG. 1.—(*a*) Gyratory crusher; (*b*) gyrocentric crusher; (*c*) swing-hammer pulverizer; (*d*) ball mill; (*e*) jaw crusher; (*f*) cone crusher; (*g*) roll crusher; (*h*) pan or chaser mill; (*i*) roller mill. (*Badger and McCabe, "Elements of Chemical Engineering."*)

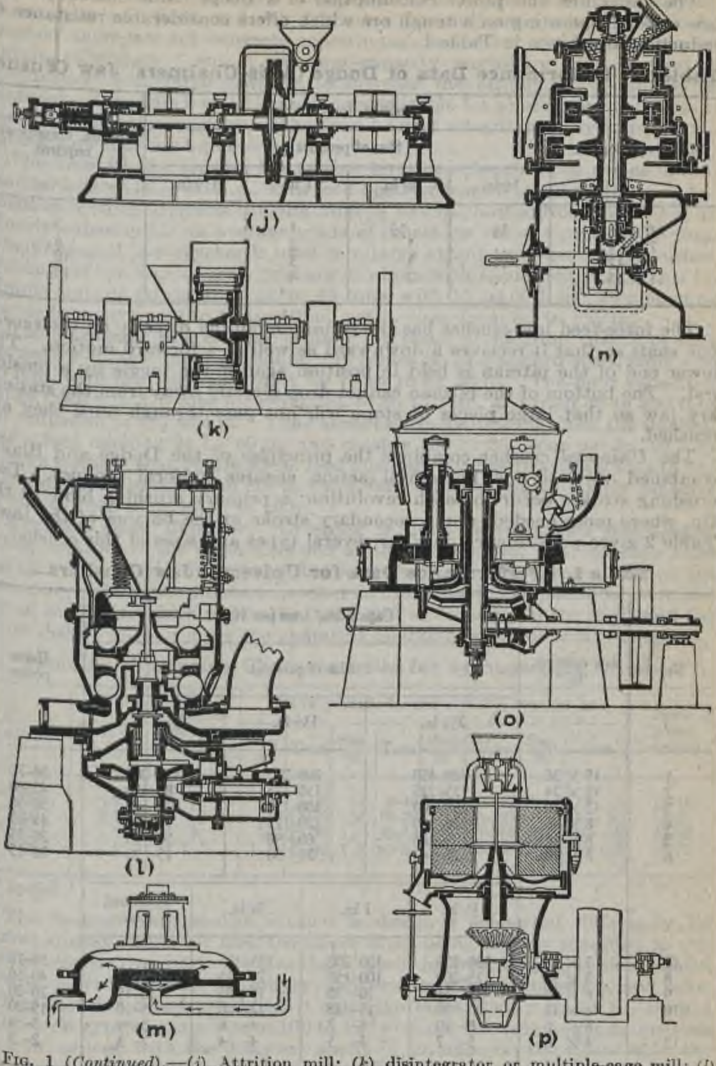


FIG. 1 (*Continued*).—(*j*) Attrition mill; (*k*) disintegrator or multiple-cage mill; (*l*) Fuller air-separation mill; (*m*) colloid mill; (*n*) vertical-beater mill; (*o*) Raymond low-side air-separation mill; (*p*) buhrstone mill. (*Badger and McCabe, "Elements of Chemical Engineering."*)

The capacities and power consumption of a *Dodge* (*Allis-Chalmers*) type jaw crusher operating on a tough ore which offers considerable resistance to reduction are shown in Table 1.

Table 1. Performance Data of Dodge (Allis-Chalmers) Jaw Crusher

Size of opening, in.	Capacity, tons per hr.				Approx. horse power required
	Size of product				
	½ in.	¾ in.	1 in.	1½ in.	
4 × 6	¾	½	1	..	3
7 × 9	..	1	2	3	6
8 × 12	..	1	3	4	10
11 × 15	..	2	4	6	15

The force-feed jaw crusher has the pitman mounted directly on the eccentric shaft so that it receives a downward as well as a forward motion. The lower end of the pitman is held in position against the toggle by a tension rod. The bottom of the pitman cannot drop directly away from the stationary jaw so that large pieces of stone will not pass through until they are crushed.

The *Universal* crusher combines the principles of the Dodge and Blake; combined vertical and horizontal action ensures uniform product. Two crushing strokes occur for each revolution: a primary crushing blow at the tip, where most needed; and a secondary stroke at the bottom of the jaws. Table 2 gives performance data for several types and sizes of this crusher.

Table 2. Performance Data for Universal Jaw Crushers

No.	Jaw opening, in.	Capacities, tons per 10 hr.				Horse power
		Size of product				
		2½ in.	1½ in.	¾–1 in.		
1	15 × 36	300–450	200–275	150–200		50–70
2	12 × 24	175–225	150–175	80–120		40–60
3	12 × 20	150–200	100–150	75–100		30–50
4	8 × 36	200–300	120–175	95–150		45–65
5	8 × 18	125–175	60– 90	50– 75		20–35
6	8 × 10	25– 40	20– 30	15– 20		10–15
		1½ in.	1 in.	¾ in.	¼ in. and finer	
7	5 × 48	180–275	150–200	125–175	.....	50–70
8	5 × 36	125–200	100–150	75–125	.....	40–50
9	6 × 16	25– 35	20– 30	15– 25	10–15	20–30
10	6 × 12	20– 30	15– 25	12– 20	8–10	15–20
11	5 × 6	8– 10	5– 7	4– 6	3– 4	3– 4
12	4 × 6	5– 7	4– 5	4– 4	2– 3	2– 3

In Table 2, crushers 1 to 3 inclusive are used largely in the rock-quarry and mining industries, where a large feed opening and a high ratio of reduction are required. Crushers 4 to 6 inclusive are well adapted for crushing rejections, ¾ in. and finer, and will produce a fine uniform product suitable for crushed aggregate for concrete construction and other materials of similar nature. Crushers 7 and 8 are large-capacity recrushers and are used in conjunction with large crushers to increase the output of ¾- to 1½-in. material. Crushers 9 to 12 inclusive are suitable for producing small sizes of rock in moderate capacities. They can be used advantageously to produce a satisfactory feed for a pulverizer.

The *Telsmith* jaw crusher has neither levers nor toggles. It comes in four standard sizes, 12 to 30 h.p., with feed opening from 7 by 12 to 15 by 24 in., smallest bottom advisable ranging from ¾ to 2 in., largest from 1¼ to 3½ in. Hourly capacity for an average grade of limestone varies from 3 to 38 tons. The *Champion* jaw crusher is used to a large extent as a secondary crusher although it can do moderate primary crushing with reductions from 8 to 1 in. Hourly output ranges from 10 to 45 tons, with 15 to 50 h.p. Jaw opening varies from 10 by 20 to 10 by 40 in., set to close from ¾ to 1½ in.

The *Buchanan* jaw crusher is used for small or medium plants or as a secondary crusher in large plants. It comes in 10 standard sizes, from 4 to 60 h.p., with jaw opening from 4 by 12 to 18 by 36 in. Ten-hour capacities range from 8 to 375 tons for reduction to ½ to 2 in., and from 30 to 650 tons for reduction to 2½ to 4 in. The *Fairmount* comes in three sizes: the smallest has a feed opening 24 by 60 in. and crushes 100 to 250 tons per hr. to 5 to 6 in.; a medium size, 36- by 60-in. feed opening, crushes 150 to 500 tons to 6 to 8 in.; and a large size with 60- to 48-in. feed opening and an hourly capacity of 500 to 1500 tons, to 10 to 15 in.

Symons Disk Crusher. Crushing is done between two dish-shaped disks, rotating in the same direction at the same speed, their concave sides facing. The feed is introduced between the disks and thrown toward the periphery, where the finished product is discharged. Crushing is increased by an eccentric, which places the inner disk at an angle to the shaft and the outer disk. Table 3 gives the operating characteristics for this crusher.

Table 3. Operating Characteristics for Symons Disk Crusher

Size of mill, in.	Feed, in.	Minimum exit opening, in.	Size of product, in. and capacity, tons per hr.								Horse power required
			Size, in.	Tons	Size, in.	Tons	Size, in.	Tons	Size, in.	Tons	
18	1.5	⅜	⅜	5– 8	½	8–10	¾	10– 12	1	12– 15	12–18
24	2.5	½	½	12–15	¾	18–20	1	20– 25	1½	25– 30	18–25
36	3.5	¾	¾	25–30	1	30–45	1½	45– 60	2*	50– 65	40–50
48	6.5	1	1	45–60	1.5	60–80	2*	80–100	2½*	100–120	60–75

* Special.

The Symons vertical-disk crusher is designed somewhat differently for coarse crushing than for fine, the upper disks being slightly modified to suit the particular product. It has been used for cracking graphite, schist, pyrite, quartz, monazite, porphyry, copper ores, gravel, limestone, and coke. When used for coarse crushing, it can take materials up to 4.5 in. from a Blake or a gyratory at a rate of 100 to 120 tons per hr., reducing it in one pass to 1 in. or less, with the disks set about ⅞ in. open on the closed side, the power requirement being about 65 h.p. Under different conditions it reduces 3- to 4-in. material to ½ in. or finer in one pass, at a rate of 90 to 100 tons per hr., with about 55 h.p. Taking a feed of 1½ to 2½ in., it handles 20 to 35 tons per hr., crushing to ¼ in. and less with about 60 h.p. As a rule the disks should not be set closer than $\frac{3}{16}$ in.

Cone Crushers. In recent years the cone crusher has tended to replace the disk crusher as well as many other types of secondary crushers, as it is better suited for finer sizing. The conical head rotates so rapidly that every piece of rock is pinched several times as it progresses through the crusher, and very little oversize reaches the discharge opening. An umbrella-shaped head is fixed on a gyratory shaft without upper support or suspension bearing. Heavy springs hold the upper frame fixed; when choked by overfeeding or tramp iron, the springs allow the upper frame to rise at the point of stress. The two common types are the *Telsmith* and the *Symons* (Fig. 1*f*). Operating characteristics for the two types are given in Table 4.

Table 4. Operating Characteristics for Cone Crushers

Type of crusher	Width of feed opening, in.	Capacities, tons per hr.							Horse power
		Discharge							
		⅜ in.	½ in.	⅝ in.	¾ in.	1 in.	1¼ in.	1½ in.	
Telsmith No. 24......	3	15–20	20–25	......	25– 30	30– 35	....	....	25– 30
Telsmith No. 36......	4½	.....	35–45	45–55	55– 70	70– 90	90–110	110–130	50– 60
Telsmith No. 48......	7	.....	50–65	65–85	85–105	105–135	135–170	170–210	75–100
Symons 2 ft..........	3	.....	25	30	35	45	50	60	25– 30
Symons 2 ft..........	1⅝	20	25						
Symons 3 ft..........	4½	.....	40	55	70	80	85	90	50– 60
Symons 4 ft..........	6¾	.....	8	100	120	150	170	....	75–100
Symons 5½ ft..........	9¾	.....	130	160	300	275	300	375	150–200
Symons 7 ft..........	14	.....	225	......	330	450	560	600	250–300

The *Seymour impact crusher* is a secondary crusher. There is a tendency with this crusher to produce a more cubical product and to decrease the wear and maintenance associated with the average impact crusher. The velocity of the incoming feed is so timed in relation to the speed of the rotating impeller that the entire area of an impact plate strikes the material as it enters the crusher. The object is to utilize the full surface for crushing—the material, being struck at right angles with no slip, decreases wear on the crushing surfaces. After receiving its initial crushing by impact at the propeller, the material is projected horizontally forward and strikes a heavy top plate where it is further reduced in size. Owing to the design, a great deal of the crushing is accomplished by material impinging on material, which tends to decrease wear and to give a more rounded product.

This crusher has its greatest application in making chips, granules, and other finely crushed materials such as stone, sand, agstone, etc. It is particularly adapted for handling wet and dirty stone as there are no grids or grates to clog up. These units are manufactured in capacities from 25 to 225 tons per hr. using from 25 to 150 h.p. The exact capacity and power depend on the nature of the rock and the fineness of reduction.

Rotary Crushers. These crushers operate on the same principle as an ordinary household coffee grinder. A rotating vertical shaft carries a cone with large teeth at the top, for initial crushing, and small teeth or furrows at the bottom for finer crushing. The shell—the upper part of the machine—is equipped with corresponding teeth. A No. 4 *Bartlett and Snow* crusher, operating on burned lime or gypsum, has an hourly capacity of 3 to 5 tons under ½-in. size and requires 8 to 10 h.p.; a No. 1 small rotary grinds about ½ ton per hr. with 3 to 4 h.p.; the No. 2 unit produces about twice as much with 4 to 5 h.p. In all cases the discharge is between ¼ and ½ in.

The *Sturtevant* is well adapted for fine crushing of burned lime, gypsum, hard clays, and phosphate rocks. It comes in five standard sizes, from 1 to 25 h.p., with speeds ranging from 200 to 1150 r.p.m. Hourly capacities, with reductions to ¼ in. and below, vary from 1 to 10 tons.

Crushing Rolls. *Theory.* This term includes a variety of machinery consisting of one or more cylinders rotating in a horizontal plane; the rolls may be smooth or corrugated, the same size or different sizes, and may run at the same or at different speeds. The simplest type consists of a single roll crushing against a breaker; it is generally used for fairly coarse crushing. For finer products pairs of rolls are used, revolving toward each other. Argall (Richards and Locke, "Textbook of Ore Dressing") gives the following formula for the diameter of rolls: $0.0476\,D = Sn$, where D is the roll diameter in inches, and Sn the diameter in inches of the maximum particle in the feed. The angle of nip, the angle formed by the tangents to the roll faces at the point of contact with a particle to be crushed, is determined by the following formula:

$$\cos\frac{N}{2} = \frac{r + a}{r + b}$$

where r = radius of rolls, a = one-half space between rolls, b = radius of particle, and N = angle of nip. The angle of nip varies from about 0 to 50 deg.

The capacity increases with the length and the diameter of the roll. When the rolls are kept full, the crushing is done not only by the action of the rolls but by the attrition between the particles themselves. This is called "choke" crushing. In free crushing the rolls are fed at such a rate that each particle is crushed and ejected before the next is nipped. Free crushing produces a larger proportion of coarser sizes and is generally more advantageous, whereas choke crushing is resorted to for the production of a fine product if other types of crushers are not found more suitable.

One of the shafts moves in fixed bearings, the other in movable bearings. The distance between rolls is adjusted by shims or, for medium or smaller rolls, by a set of gears. A nest of powerful springs holds the movable roll against the shims.

The following procedure may be used to determine operating characteristics of rolls, bearing in mind that capacity is influenced by the character of the feed, fineness of reduction, and the required manner of operation. Capacity is in direct ratio to width and peripheral speed and may be calculated by the following formula:

$$C = \frac{T \times W \times S}{1728}$$

where C = cubic feet per minute, T = distance between rolls in inches, W = width of rolls in inches, and S = peripheral speed in inches per minute. This gives the theoretical capacity and is based on the rolls discharging a continuous, solid, uniform ribbon of material. On account of irregularity in the feed, the value obtained for C should be divided by 4 to obtain actual capacity.

The peripheral speed at which rolls usually operate is from 500 to 1200 ft. per min. occasionally as high as 1500 ft. Economical range of reduction is usually limited to a 12- to 16-mesh product. Roll diameter required for a

Table 5. Operating Characteristics for Traylor Roll Crushers

Size of roll, in.	Size of product, in.	Max. size of feed, in.	Capacity, tons per hr.	Speed, r.p.m.	Horse power
Type A:					
9 × 10	⅛	½	2	300–350	6
18 × 10	⅛	¾	5	250–300	8
24 × 12	⅛	¾	10	180–230	10
30 × 14	⅛	1	15	130–180	15
30 × 16	⅛	1	20	130–180	15
36 × 16	¼	1½	30	100–150	25
42 × 16	¼	1¾	35	85–100	35
Type AA:					
48 × 18	⅜	2	50	75–105	60
54 × 20	⅜	2¼	65	70– 95	70
60 × 24	½	2⅝	90	65– 85	90
72 × 24	½	3	100	50– 75	100
72 × 30	½	3	130	50– 75	125

Table 6. Operating Characteristics for Bartlett and Snow Roll Crushers

Roll size		Capacity, tons per hr. for various-size products					Horse power at given capacities for various sizes					Max. size of feed, in.
Diam., in.	Width, in.	1¼ in	1½ in.	2 in.	3 in.	4 in.	1¼ in.	1½ in.	2 in.	3 in.	4 in.	
Single-roll Crushers												
18	18	35	45	55	65	80	15	16	18	18	17	8
24	24	75	90	110	130	140	27	30	32	30	25	14
30	30	120	175	225	270	300	40	55	65	60	50	20
36	48	375	450	550	650	750	120	130	140	110	110	24

		Capacity, tons per hr., for various-size products						Horse power at given capacities for various sizes					
		½ in.	¾ in.	1 in.	1¼ in.	1½ in.	2 in.	½ in.	¾ in.	1 in.	1¼ in	1½ in.	2 in.
Two-roll Crushers													
26	24	...	...	50	75	100	150	..	..	39	43	46	50
26	36	...	...	100	135	175	250	..	..	66	69	72	77
30	36	...	...	110	150	190	265	..	..	71	75	78	80
37	36	...	...	115	155	200	280	..	..	74	78	81	85
37	48	...	...	165	220	280	390	..	..	100	105	110	113
Four-roll Crushers													
26	24	50	75	100	125	150	200	40	45	48	48	47	48
26	36	75	110	150	185	225	300	53	59	65	64	61	64
30	36	80	125	160	200	240	315	55	63	69	68	67	67
37	36	85	125	165	205	250	330	58	65	70	69	69	69
37	48	105	160	215	270	330	440	68	79	87	86	86	87

given reduction is determined by the maximum size of feed which can be nipped without slippage. This size is calculated from the formula

$$2X = 0.08r + 2a$$

where $2X$ = the maximum size of feed, r = radius of roll in inches, and $2a$ = distance between roll faces (also see Tables 7*a* and 7*b* for determining roll characteristics).

The *Traylor* rolls are made in two types, the four-tension rod type, and the heavy-duty type, both of which may be fitted with automatic lateral adjustments. Operating characteristics are given in Table 5.

Bartlett and Snow Roll Crushers (Fig. 1*g*). For many purposes the single-roll crusher is as satisfactory as the larger multiple-roll units. When crushing coal it should be confined to conditions where a product 1¼ in. or larger is desired, and where an excess of fines is not objectionable. This machine will turn out a product which will be satisfactory for stokers of the plunger type, but for chain-grade stokers requiring a more uniform size of coal the two-roll or the four-roll crushers give better results. The single-roll crusher works by abrasion rather than by cracking, which accounts for the high percentage of fines, whereas the two-roll crusher cracks the coal. When the capacity is large, or where the size of the product must be small, the four-roll crusher is the most economical in use. Operating characteristics of the three types are given in Table 6. The data are based on crushing bituminous coal.

Table 7a. Specifications of Buchanan Rolls

Size of roll, in.	Average r.p.m.	Horse power	Maximum 4 :1 reduction			
			Smooth		Corrugated	
			Size of cube, in.	Distance apart, in.	Size of cube, in.	Distance apart, in.
Type C and type CS:						
24 × 14	125–175	12– 30	1	¼	2	½
30 × 14	90–140	15– 25	1¼	5/16	2½	⅝
36 × 16	80–125	20– 35	1½	⅜	3	¾
42 × 16	70–100	30– 45	1¾	7/16	3½	⅞
CS 42 × 24	70–100	45– 75	1¾	7/16	3½	⅞
CS 48 × 24	65– 85	50– 80	2	½	4	1
Type E:						
60 × 24	60–100	70–125	2½	⅝	5	1¼
64 × 24	60–100	75–150	2¾	11/16	5⅜	1 5/16
64 × 30	60–100	100–150	2¾	1⅜	5⅜	1 5/16
72 × 24	50– 85	125–172	3	¾	6	1½
72 × 30	50– 85	150–200	3	¾	6	1½
78 × 24	50– 80	150–200	3¼	13/16	6½	1⅝
78 × 30	50– 80	170–225	3¼	13/16	6½	1⅝
78 × 36	50– 60	170–250	3¼	13/16	6½	1⅝

The *Jeffrey single-roll crusher* is available in sizes having openings from 18 by 18 in. to 36 by 54 in. with capacities of from 20 to 700 tons per hr. on hard bituminous coal. The power requirements vary from 7.5 h.p. for the smallest size to 25 h.p. for the largest.

The *Buchanan type E* roll crusher is also equipped with four steel tension rods, two above and two below the bearings. Each linear inch of roll surface develops a pressure of 12,000 to 36,000 lb. per sq. in., equivalent to a crushing strength of 35,000 to 100,000 lb. per sq. in. of effective crushing surface (about 0.3 of the width of the face). The rolls may be fitted with any type of shell, corrugated, grooved, or plain.

For crushing coarse material the roll speed should be less than for fine material, especially when corrugated shells are used. For soft and brittle materials a much higher speed may be used than given in the table, although a higher linear speed than 1500 ft. per min. is not advisable. The reduction ratio of 4:1 should not be exceeded; in fact, when feeding large pieces, especially if the rock is hard, it is advisable not to attempt a 4:1 reduction; 3:1 or even 2.5:1 gives better results. For small material, about one-third or one-

Table 7b. Capacities of Buchanan Rolls

Cubic feet per hour

Opening between faces, in.	Speed of roll face, ft. per min.											
	500	600	700	800	900	1,000	500	600	700	800	900	1,000
	Face of Roll, 14 In.						Face of Roll, 16 In.					
1/16	60	72	84	96	108	120	69	83	97	111	125	138
⅛	120	144	168	192	216	240	138	166	194	222	250	276
3/16	182	216	252	288	324	364	208	249	291	333	375	414
¼	240	288	336	384	432	480	277	332	388	444	500	552
5/16	305	360	420	480	540	610						
⅜	364	432	504	576	648	728	416	498	582	666	750	828
½	486	576	672	768	864	972	555	664	776	888	1,000	1,104
⅝	607	720	840	960	1080	1,214	694	830	970	1,111	1,250	1,380
¾	729	864	1008	1152	1296	1,458	833	996	1,164	1,332	1,500	1,656
⅞	850	1020	1176	1344	1512	1,700	971	1162	1,358	1,554	1,750	1,932
1	960	1152	1344	1536	1728	1,944	1111	1328	1,552	1,776	2,000	2,222
1⅛	....	....	....	....	....	......	1250	1494	1,746	1,998	2,250	2,484
	Face of Roll, 20 In.						Face of Roll, 24 In.					
¼	347	416	486	555	624	694	417	500	583	666	750	833
⅜	520	624	729	833	936	1,040	625	750	875	999	1,125	1,250
½	694	832	972	1110	1248	1,389	833	1000	1,166	1,332	1,500	1,666
⅝	867	1040	1215	1388	1560	1,734						
¾	1041	1248	1458	1665	1872	2,082	1250	1500	1,749	1,998	2,250	2,500
1	1389	1664	1944	2220	2496	2,778	1666	2000	2,332	2,664	3,000	3,333
1¼	1736	2080	2430	2775	3120	3,472	2080	2500	2,915	3,330	3,750	4,165
1½	2079	2496	2916	3330	3744	4,158	2500	3000	3,498	3,996	4,500	5,000
1¾	2430	2912	3402	3885	4368	4,858	2917	3500	4,081	4,662	5,250	5,831
2	2778	3328	3908	4440	4992	5,556	3333	4000	4,664	5,328	6,000	6,666
2¼	3125	3744	4374	4995	5625	6,250	3750	4500	5,247	5,994	6,750	7,497
2½	....	....	....	....	....	......	4168	5000	5,830	6,666	7,500	8,334
	Face of Roll, 30 In.						Face of Roll, 36 In.					
⅜	780	937	1094	1250	1406	1,560	936	1122	1,308	1,494	1,680	1,972
½	1041	1248	1460	1660	1870	2,082	1250	1496	1,744	1,992	2,240	2,500
⅝	1302	1560	1825	2080	2350	2,600						
¾	1562	1874	2188	2500	2812	3,120	1872	2244	2,616	2,988	3,360	3,744
1	2083	2500	2920	3330	3745	4,167	2500	2992	3,488	3,984	4,480	5,000
1¼	2605	3120	3650	4160	4680	5,200	3124	3740	4,360	4,980	5,600	6,248
1½	3124	3744	4380	5000	5616	6,250	3750	4488	5,232	5,976	6,720	7,500
1¾	3645	4370	5100	5824	6550	7,290						
2	4167	5000	5840	6660	7500	8,334	5000	5984	6,976	7,968	8,960	10,000
2¼	4687	5616	6570	7490	8425	9,374	5624	6732	7,848	8,964	10,080	11,248
2½	5207	6240	7300	8320	9370	10,400	6250	7480	8,720	9,960	11,200	12,500
3	....	....	....	....	....	......	7500	8976	10,464	11,952	13,440	15,000

quarter of the actual thickness that the rolls will nip, a reduction of 8:1 or even higher may be made with satisfactory results, if a preponderance of fines is not objectionable. Operating characteristics of the rolls are given in Tables 7*a* and 7*b*.

Example. Find the size and speed of rolls for crushing rock from 5¼ in. down to 1¾ in. at the rate of 230 tons per hr., assuming the rock to weigh 100 lb. per cu. ft. This makes 4600 cu. ft. per hr. The nearest proper roll diameter is 64 in. and the face of the roll is 24 in. (see Table 7*a*). Referring to Table 7*b*, we find the nearest larger capacity to 4600 to be 4662. Hourly tonnage capacity is therefore 4662 × 100 ÷ 2000 = 233 tons. The circumference of a 64-in. roll is 16.66 ft. We note from the table that the peripheral velocity for this condition is 800 ft. per min.; 800 ÷ 16.66 = 48 r.p.m., or about 50 r.p.m. Thus the 64-in. diameter by 24-in. face corrugated rolls at 50 r.p.m. will crush 230 tons of rock per hr. from 5.25 in. down to 1.75 in.

The Patterson gyrocentric crusher illustrated in Fig. 1*b* has a single moving part—the manganese roll which by rapid gyro-eccentric motion operates against two concave manganese crushing plates. The roll fitted with a shell does not revolve with the shaft but creeps slowly, thus distributing the wear over the entire surface. The revolving eccentric shaft imparts the gyro-eccentric motion to the roll, there being two crushing strokes to each revolution. The crushing action is compressive, which tends to produce a low percentage of fines. The standard size machines have a production range from ½ to 1½ in. in size with corresponding capacities from 15 to 85 tons per hour with a power consumption of from ½ to 1 h.p. per ton of material crushed.

Table 8. Screen Test of Product from Steam Stamp Mill

On 5 mesh through ⅝ in.	On 10 mesh through 5	On 20 mesh through 10	On 40 mesh through 20	On 60 mesh through 40	On 100 mesh through 60	Through 100 mesh	Loss
44.5%	11.1%	7.0%	6.2%	4.8%	3.0%	22.2%	1.2%

Table 9. Operating Characteristics of Gravity Stamp Mill

Weight of stamp, lb.	Drops per min.	Height of drop, in.	Screen, in.	Tons per stamp per 24 hr.	Water per stamp, gal. per min.
750	100	7	0.0148	2.5	3.5
900	88	10.5	0.0220	4	7–8
1050	95–98	5.5–8.0	0.0235	3.875	5
1500*	.....	....	0.0197	8.0–9.5	10–11

* *Nissen* stamp. All others regular five-stamp mortars.

Stamp Mills. Crushing is effected by the blow of a pestle or stamp upon material resting in a mortar. The end of the stamp, the shoe, and the bottom of the mortar, the die, are replaceable. Two main types are used, the *steam stamp* and the *gravity stamp*. Grinding in water is the usual practice, the crushed product being then discharged through a screen. The use of five stamps in one mortar is almost invariable in this country. Table 8 gives the sizing test of a pulp from a steam stamp crushing through a ⅝-in. round hole. The capacity of the stamp in this case was 653 tons per 24 hr.

The *Ross pounder* is in reality a stamp. It is used for crushing materials such as cochineal, olibanum, mustard, and other oily seeds. This unit is made singly or in multiples of any number. Operating results obtained in modern gravity stamp mills is given in Table 9.

Ball Mills, Tube Mills, and Pebble Mills (Fig. 1*d*). These mills have a cylindrical, conical, or spheroidal shell, rotating on a horizontal axis and charged with a grinding medium such as balls of steel, flint, porcelain, wood, or hard rubber, or with steel rods. The ball mill differs from the tube mill by being short in length; the length, as a rule, does not exceed the diameter. Standard ball mills use large balls on a coarse feed to produce a comparatively coarse product; the tube mill is usually long in comparison with the diameter, uses smaller balls, and produces a finer product. The compartment mill, a combination of the above types, consists of a cylinder divided into two or more sections by perforated partitions; preliminary crushing takes place at one end and finishing crushing at the discharge end. Rod mills are similar to tube mills but use rods as the grinding medium, depending on line contact instead of point contact for grinding. Rod mills deliver a more uniform and more granular product than other revolving mills, thus minimizing the percentage of fines which are detrimental in some industries. The pebble mill is a tube mill with flint pebbles as the grinding medium and is lined with silex or other non-metallic liners. The pot mill consists of a spheroidal shell with an inner bowl made of one solid piece of vitreous material.

Theory of the Ball Mill. It is difficult to formulate a general theory of the operation of a ball mill. Most of the research work on the subject has been done with various types of ores and, to a lesser extent, with cement clinker. When it is realized that the ball mill is used for wet and dry grinding, for batch, continuous, and closed-circuit grinding of materials with such diverse characteristics as hard silica, ores ground dry, and a mixture of oil and soft pigments, it will be appreciated that it is impossible to give general formulas valid for all operating conditions.

Experimental evidence presented in a paper by Coghill and DeVaney (Ball-mill Grinding, *U.S. Bur. Mines, Tech. Paper* 581, 1937) causes the authors to draw the following conclusions:

1. In wet-batch ball milling with ore charges from 200 to 350 pounds (about 75 pounds of ore was required to fill the interstices of the balls at rest) and speeds from 30 to 80 per cent critical, the slow speed gave the same type of grinding as high speed. Heavy ore charges yielded a little more selective grinding of the coarse particles than light charges. Best capacities were obtained with light charges and slightly better efficiencies were obtained with heavy ore charges. To split hairs about efficiencies at various speeds the reader will have to study the table and be his own judge.
2. Some of the characteristics of dry-batch ball milling were unlike those of wet grinding. In the dry work, efficiency as well as capacity was best with the light ore charge. Power decreased with decrease in the amount of ore in the mill; in wet grinding it increased with a decrease in the amount of ore in the mill. In dry grinding high speed was more efficient than low speed.
3. In comparing wet and dry grinding the tests were paired so that all the set variables were the same, except pulp consistency (wet or dry). With an intermediate weight of ore charge, selective grinding was of the same degree; with a heavy ore charge, wet grinding was more selective, and with the light ore charge, dry grinding was more selective.
4. In comparing wet and dry open-circuit ball milling, wet grinding gave 39 per cent more capacity and 26 per cent more efficiency.
5. A small ball volume was not satisfactory in the overflow type of dry mill because too much ore built up in the mill. When building up of the ore was prevented by simulating the low pulp-level mill, the small ball volume did good work.

6. With 60 per cent solids, pebbles the same size as balls did about the same type of work as balls when dolomite was ground, but they failed in selective grinding of chert. Pebbles gave about 35 per cent of the capacity and 81 per cent of the efficiency shown by the balls.

7. For hard and medium-hard ores, tetrahedrons were unsatisfactory for coarse grinding.

8. Very hard balls (Ni-hard) were better than ordinary balls; this was particularly so when the ore was very hard.

9. The efficiency of battered reject balls was about 11 per cent less than that of new spherical balls.

10. A ball mill as small as 19 by 36 inches duplicated the work of a plant-size mill. The tests led to the belief that if each of a variety of mills, large or small, is run under the same conditions, and if each applies a unit of work to a unit of ore, the effect (comminution), as indicated by the products, will be the same—that is, the same relation between cause and effect will maintain.

The F. L. Smidth and Company's mills find their principal use in cement plants; the best known types are the Kominutor, straight tube, Uni-Kom, Unidan, and the Pyrator.

The dry-tube mill is usually fed by a table feeder or a cradle feeder. It has silex or dragpeb lining. Flint pebbles are used with silex lining, and cylpebs in mills with dragpeb lining. The wet-tube mill is usually fed by gravity from a slurry trough in the form of an orifice or scoop feeder. It is furnished with peripheral or central discharge.

The *Kominutor* is a screen-type mill operated in closed circuit, usually the first in a two-stage unit where reduction is carried only to the size of a rather coarse mesh, which is fed to tube mills for final grinding. For dry grinding, the table feeder or cradle feeder is used. In wet grinding, water is added at the feed hopper, passing the slurry produced through the mill as in dry grinding.

The *Uni-Kom* is a four-compartment mill, a section with large diameter forming the first compartment, with a section of small diameter divided into three compartments. The enlarged section is fitted with liner plates and a special screening arrangement which by-passes fines to the first compartment in the second section and returns oversize to the granulating compartment for further reduction. The first chamber of the second section is equipped with grinding balls, the following compartments with cylpebs, graded downward in size.

The *Unidan* mill is a compartment mill, with three or more compartments, equally well suited for wet and dry grinding. Balls are used in the first compartment, which is also equipped with liners. The compartments for fine grinding are equipped with special rings and steel-alloy lining and with cylpebs graded downward in size toward the discharge end. An added feature of the mill is a special screen arrangement mounted within the mill body; the material does not leave the mill body until finally discharged at the outlet end.

The *Pyrator* mill is used for granulating, pulverizing, and drying damp material in a single unit; it consists of a two-compartment tube mill comprising a ball chamber with liners; a combined screening and ball-separating partition, and a fine-grinding chamber with special ring and alloy lining and cylpebs as grinding medium. The steel balls are heated and have the double capacity of acting as grinding medium and supplying the heat required for drying. Hot air circulates around the mill body which is provided with a jacket to retain the hot air in closed circuit, and the mill is provided with means for removing the balls, which are elevated, heated in a hot-air furnace, and returned to the feed end. When the material does not contain an excessive amount of moisture, this unit is very efficient, compact, and economical.

The *Allis-Chalmers ball granulator* is used for crushing 2- to 3-in. ore to a fineness of 10 to 80 mesh and also as a fine grinder for reducing 8- to 20-mesh material to −100 to −200 mesh. It has a variable pulp level which may be controlled to ensure maximum efficiency and is made in 25 standard sizes, from 3 by 5 ft. to 10½ by 12 ft. and from 25 to 750 h.p. Capacities range from 20 to 385 tons per 24 hr., reducing medium-hard quartz from 2 in. to 48 mesh, and from 30 to 600 tons with reduction from 2 in. to 14 mesh. The speed for the largest types is 17 r.p.m., and for the smallest 44 r.p.m.

The *Marcy ball mill*, used extensively for wet and dry grinding of ores, grinds a 2½-in. feed to 8 mesh without use of screens or classifiers. Operating characteristics, on 2½-in. feed, medium-hard rock, are given in Table 10.

Table 10. Operating Characteristics of Marcy Ball Mills

Size, ft.	Usual charge	Horse power required	Speed, r.p.m.	Capacity, tons per hr.					
				−8 mesh	−20 mesh	−35 mesh	−48 mesh	−65 mesh	−100 mesh
3 × 2	1,000	6– 8	35	20	15	20	10	7	4
4 × 3	2,500	15– 20	31	50	38	30	25	18	11
5 × 4	5,000	30– 40	29	125	94	80	63	44	25
6 × 4.5	9,000	65– 90	25	360	225	180	150	105	60
7 × 5	18,000	100–150	23	655	490	390	325	230	130
8 × 6	28,000	150–225	22	1000	750	600	500	360	200

The *Solo tube mill* is widely used in the cement industry; it is a compartment mill, generally with three compartments. The first compartment is loaded with balls ranging from 3 to 5 in.; the intermediate compartment has 1- to 2-in. balls, and the finishing compartment ⅞-in. balls. Operating characteristics of a mill grinding cement are given in Table 11.

Table 11. Operating Data of Solo Tube Mill, Grinding Cement

Size, diam. and length, ft.	6 × 36	6.5 × 39.5	7.25 × 43	8 × 49.5
Speed, r.p.m.	23	21	19	18
Number of compartments	3	3	3	3
Size of feed, in.	1.5	1.5	2	2.5
% through 200 mesh	88+	88+	88+	88+
Tons per hour on raw mix	15	18	28	40
Bbl. per hour clinker	50	60	95	135

Improvements have recently been made in the design of the Solo mill by using it in conjunction with a special air separator which increases the capacity and fineness when grinding cement clinker.

The *Allis-Chalmers Compeb* mill has two or more compartments; reduction from 1½ in. to finished product is made in one operation, as grinding cement clinker to a fineness of from 85 to 98 per cent through 200 mesh. It operates more efficiently in closed circuit. Standard mills are available in sizes up to 10 ft. in diameter and 50 ft. long. The preliminary or short compartment is charged with 2- to 5-in. forged steel balls, proportioned to give maximum efficiency; the finishing compartments have concavex grinding mediums, balls presenting both a concave and a convex surface, which have been found superior to spheroids.

The *Hardinge conical mill* is used extensively for wet and dry grinding, in open circuit and in closed circuit. A mixed load of large and small balls is used; the large balls assemble at the feed end where the diameter is largest, while the small balls arrange themselves in decreasing sizes toward the discharge with the material to be crushed arranged in corresponding manner. Several types of feeders are used. Scoop feeders work best for wet materials: for smaller mills, grinding both wet and dry, the conical feeder is satisfactory. Drum feeders are used primarily for large ball and pebble mills, grinding wet, where the material is fed directly into the mill. A combination drum and scoop feeder is used for coarse feed, at the same time picking up oversize from the classifier. This combination is capable of handling any quantity which the mill may be capable of grinding, in addition to a circulating load of 1000 per cent. For dry grinding in small diameter trunnion mills or where capacities are very high, a screw feeder must be used. Hardinge wet grinding mills are now supplied with discharge arrangements for high, medium, or low pulp levels, depending upon the characteristics of the ore and product desired. For fine or medium products the high pulp-level discharge spout or the standard discharge spout with cone grate is supplied. When a granular product or very high circulating load is desired, the low pulp-level discharge, which consists of a conical grate with discharge scoops, is used. A similar discharge arrangement is used in dry grinding mills for granular products such as glass, sand, or abrasives.

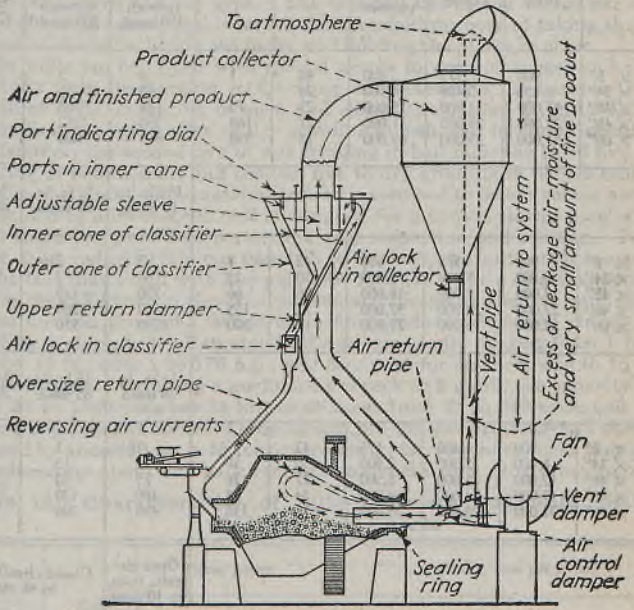


FIG. 2.—Hardinge conical mill with superfine air classifier.

The mill is fed by a *Hardinge constant-weight belt feeder*, which is 98 per cent accurate, thus making it possible for the operator to keep a record of the weight of the material fed. It is no longer necessary to use an enclosed type of feeder, as the feed end of the mill is under a partial vacuum and a slight suction is maintained at the feed chute to draw any dust into the mill.

Table 12. Operation Characteristics of Hardinge Ball Mills

Size of mill	Approx. weight, lb.			Speed, r.p.m.	Horse power to run	Capacity, tons per 24 hr.		
	Mill	Lining	Balls or pebbles			1½ in. to 90%, through 100 mesh	¾ in. to 90%, through 200 mesh	½ in. to 98%, through 325 mesh
2′ × 8″	900	375	400	40	1	4	3	1½
4½′ × 24″	8,100	5,400	4,500	28	25	48	36	18
6′ × 48″	17,000	12,000	15,000	25	70	144	108	54
8′ × 48″	27,000	23,000	31,000	21	160	360	252	126
10′ × 66″	51,000	35,000	65,000	18	350	840	600	300
						1½ in. to 10 mesh	¾ in. to 80 mesh	¼ in. to 200 mesh
2′ × 8″	900	375	500	42	1	5	2½	2
4½′ × 24″	8,100	5,400	5,000	30	23	60	2	23
6′ × 48″	17,000	12,000	18,000	26	66	230	110	90
8′ × 48″	27,000	23,000	37,000	21	155	550	260	215
10′ × 66″	51,000	35,000	75,000	17	320	1200	550	465
						½ in. to 28 mesh	½ in. to 65 mesh	8 mesh to 200 mesh
2′ × 8″	900	400	175	42	½	1½	1	⅔
4½″ × 24″	8,000	2,300	2,400	30	10	17	13	9
6′ × 48″	12,000	5,000	8,500	27	26	75	50	33
8′ × 48″	17,000	14,000	14,000	24	58	180	120	80
10′ × 66″	32,000	20,000	28,000	18	150	450	300	200
						Open circuit, ½-in. to 10 mesh	Closed circuit, ½ in. to 48 mesh	
2′ × 8″	900	400	175	42	½	2	1½	
4½′ × 24″	8,000	2,300	2,400	30	12	15	9	
6′ × 48″	12,000	5,000	8,500	27	30	54	36	
8′ × 48″	17,000	14,000	14,000	24	62	120	84	
10′ × 66″	32,000	20,000	28,000	18	160	336	216	

Hardinge air classifiers operate on the principle of reversed air currents, air being injected at the discharge end. The current reverses itself in the grinding zone and is drawn back to the discharge end carrying the product with it. Oversize drops out in the classifier above the mill and returns by gravity either to the feed end of the mill where it enters with the new feed or to the discharge end where it is pushed up by the incoming air current and blown back into the grinding zone. Two types are used, the superfine classifier for products from 60 to 325 mesh and the loop classifier for products from 10 to 60 mesh.

The superfine classifier is shown in Fig. 2 (see Air Separation). Air is blown into the mill through the central pipe. As the air is withdrawn around the pipe, the coarse oversize drops into the rotary classifier revolving with the mill. The classifier is elevated to return the oversize to the mill by gravity. In the superfine section the product is classified further as it passes up between the two cones within the classifier. The finished product is blown to a cyclone collector, and the air from the cyclone is returned to the fan, which completes the cycle.

All Hardinge mills are lined with metal or flint liners, either of the wedge-bar or the wave and rib type. The wearing bar of the wedge-bar type is made of chrome steel, and it serves the threefold purpose of taking the major impact of the balls, lifting the mass, and holding the plates in place.

The mills are equipped with special grates to prevent surges and to keep the balls from discharging. Operating characteristics for various types of Hardinge mills, operating in open and closed circuit, are given in Table 12.

The first five rows refer to dry grinding in ball mill in closed circuit with air classifier; the second five to wet grinding in ball mill; the third five to wet grinding in pebble mill; and the last five to dry grinding in pebble mill, open and closed circuit. The material ground is assumed to be of average hardness.

The *Marcy open-end rod mill* is designed for a heavy, slowly revolving rod mass and a low pulp line, permitting easy inspection and relining. It takes a 1-in. feed, reducing it in one pass to 8 to 20 mesh; the uniformity is explained by the fact that a mill with low pulp line does not make a displacement product, as the difference in elevation of feed and discharge ensures rapid removal of the finished product. Wherever a fine product is desired, closed-circuit grinding is employed. Ten standard sizes are made, ranging from 1 by 2 ft. to 7 by 15 ft., with 1 to 175 h.p., and 35 r.p.m. for smallest size to 15 r.p.m. for the largest. Grinding a medium-hard rock to 8 mesh, the capacity varies from 23 to 1280 tons per 24 hr., to 48 mesh from 12 to 640 tons, and to 100 mesh from 5 to 264 tons. These figures are for wet grinding and should be reduced by about 30 per cent for dry grinding; for very hard rocks they should be reduced by about one-half; for very soft rocks they should be doubled.

Table 13. Characteristics of Allis-Chalmers Rod Mill, Grinding Medium-hard Quartz

Size, ft.	Rod charge, lb.	Horse power		Capacity, tons per 24 hr.			
				1 in. to 20 mesh		½ in. to 60 mesh	
		Dry	Wet	Dry	Wet	Dry	Wet
3 × 6	4,900	...	15	...	95	...	45
3 × 8	6,500	...	20	...	125	...	60
4 × 8	12,000	45	36	140	240	60	110
4 × 10	15,000	55	45	175	300	75	140
5 × 10	26,500	100	80	300	500	130	225
6 × 12	47,500	160	135	550	800	250	350

The *Allis-Chalmers rod mill* is designed for both wet and dry grinding; the main difference between the two types is that mills for dry grinding have trunnion bearings cooled by water circulating through coils placed in the babbitt metal. Table 13 gives the operating characteristics for grinding medium-hard quartz to 20 and to 60 mesh.

The *Traylor rod mill* is built to stand up under hard duty and, with the exception of a few details, is of the same general design as the *Traylor ball mill*. It is built in sizes to handle 56 to 1500 tons per 24 hr. from minus 1-in. feed size to 20-mesh product size or 4 to 100 tons per 24 hr. from minus 1 in. to 200 mesh when wet-grinding ores of medium crushing resistance in closed circuit.

Jar mills, used extensively for small batches, are made of metal, porcelain, or glass, with grinding mediums suited to the material pulverized. They may be used for wet and dry grinding and should have a feed of about 8 mesh. Table 14 gives the sizes and capacities of the *Abbe porcelain jar mill*.

Table 14. Sizes and Capacities of Abbe Porcelain Jar Mill

Type	Outside diam., in.	Height, in.	Capacity		Content, gal.
			Dry charge based on sand	Liquid charge	
Specimen	5.20	5.71	⅔ pt.	1 pt.	0.25
Bacilli	5.75	6.50	1 pt.	1⅓ pt.	0.35
Assay	8.75	9.65	3 pt.	2⅓ pt.	1.25
No. 0	12.50	9.60	1 qt.	5 qt.	2.34
No. 1	13.00	12.50	1⅓ gal.	2¼ gal.	4.00
No. 2	14.75	16.50	2 gal.	3½ gal.	6.50
No. 3	17.72	10.00	2⅓ gal.	4½ gal.	7.67
No. 4	17.72	18.70	5¼ gal.	8½ gal.	15.50
No. 5	22.50	19.00	7¾ gal.	13 gal.	23.50
Domestic	11.13	13.68	1⅓ gal.	2¼ gal.	4.00

The *Day triturator* is air tight to prevent evaporation, and the charge does not come in contact with metal. It is generally used for triturating, grinding, pulverizing fine art colors, better grades of chewing gum, patent drugs, and similar materials. The inner jar is of heavy porcelain, the outer of heavy sheet iron. Operating characteristics of two simple units are given in Table 15.

Table 15. Operating Characteristics of the Day Triturator

Size of mill	Inside diam., in.	Inside depth, in.	Capacity, gal.	Horse power required
1 jar	10	13	3	¼
2 jar	10	13	6	½

The *Day pot mill* can, if required, be equipped with an inner bowl of one solid piece of vitreous material with grinding balls of the same material. Operating characteristics are given in Table 16.

Pan Mills. Edge-runner mills, chasers, Chilean mills, and dry and wet pans are included under the general head of pan mills (Fig. 1*h*). The pan mill consists of one or more rollers revolving in a pan; the pan may remain stationary and the rollers be driven, or the pan may be driven while the rolls revolve by friction. In some types the rolls (mullers) are made of stone; in others they are of stone equipped with steel tire or of iron with steel tire. Iron scrapers or plows at a proper angle feed the material under the rolls. In wet grinding the pulp is fed into the center of the pan and distributed by pipe in front of the runners. The height of the pan is varied by an adjustable curtain over which the material discharges. Fineness of product is regulated by height of curtain and volume of water.

Table 16. Operating Characteristics of the Day Pot Mill

Type	Capacity, gal.	Balls used	Horse power required	Pulley speed, r.p.m.
Vitreous	6	4– 4 in.	1	50
No. 1	6	3–10 lb.	1	50
No. 2	8.5	3–10 lb.	1	50
No. 3	24	3–10 lb.	2	50
No. 4	50	3–10 lb.	3	75

In the *Bonnot pan* the rolls are suspended by adjustable chains to permit regulation of the clearance between rolls and pan. The dry pan has a bottom fitted with screen or perforated plates along the outer edge with perforations varying from $\frac{1}{16}$ to $\frac{3}{8}$ in. The diameter varies from $5\frac{1}{2}$ to 10 ft., capacities from 1 to 6 tons per hr., and power consumption from 2 h.p. per ton for soft material, coarsely crushed, to about 8 h.p. per ton for medium-hard material when a comparatively fine product is made.

The *Ross pan* is used for dry and semidry materials such as enamel and glass which must be kept free from iron. In a standard machine the bed stone is 4 ft. 6 in. in diameter, rolls 36 in. in diameter by 14 in. face. Rolls are of buhr or other stones. If contamination is of no consequence, mills are built with iron bed and iron rolls. Operating characteristics of a chaser mill are given in Table 17.

Table 17. Operating Characteristics of a Chaser Mill

Size of pan	Capacity, lb. per hr.			Horse power
	Licorice	Dalmatian flowers	Copper oxide	
3′0″	50	35	200	3 – 7
4′8″	120	85	450	5.0– 7.5
7′8″	250	175	1200	10 –12

Attrition Mills (Fig. 1*j*). The term "attrition mill" is generally applied to a mill in which grinding takes place between two disks, of steel or stone, rotating in a horizontal or a vertical plane. In some machines each disk is on an independent shaft, separately driven and rotating in opposite directions. In another type one of the disks is stationary. Each disk has bolted to its face grinding plates which may be removed for regrinding or replacement. Mills of this type are used for grinding seeds, hops, casein, leather, beans, nuts, cereals, and wood pulp. Among the prominent types is the *Sprout-Waldron Monarch*, operating characteristics of which are given in Table 18.

Buhrstone Mills (Fig. 1*p*). This mill is an attrition mill, hard circular stones serving as grinding mediums, generally French, American, or esopus buhrstones; rock emery or combination of French buhr and esopus or of pebble grit and emery rock are also used. Buhrstone mills are used extensively in the milling industry for grinding cereals and grains. Other uses include grinding of minerals, colors, spices, and mineral pigments; reduction of cork and sawdust; wet grinding of enamels, mica, drop blocks, and polishing rouge. Feed enters the mill through a hole of one of the stones. It is distributed between the stone faces and ground while working its way to the periphery.

Table 18. Operating Characteristics of the Monarch Attrition Mill

	No. 2	No. 3	No. 4
Wheat to flour, bu. per hr.	4	6	8
Power required, h.p.	2	3	5
Wheat to meal, bu. per hr.	10	15	25
Power required, h.p.	4	5	7
Cork ground, lb. per hr.	50	75	100
Power required, h.p.	15	20	25
Sawdust ground, lb. per hr.	100	125	150
Power required, h.p.	15	20	25
Coffee ground, lb. per hr.	400	600	800
Power required, h.p.	5	7	10

The *Sturtevant rock-emery mill* is made in two types, a horizontal and a vertical; the former is made in one size and has 42-in. stones; it requires 15 to 20 h.p. and produces 3 to 4 tons per hr. of 20-mesh product, 2 to 2.5 tons 60 mesh, 1 to 1.5 tons 80 mesh, and 0.5 to 1 ton 100 mesh. Four standard sizes are made of the vertical mill, with 24-, 30-, 36-, and 42-in. stones, requiring 12 to 15, 18 to 20, 30 to 35, and 45 to 80 h.p. Hourly capacities range from 1.5 to 15 tons for 20 mesh, 1 to 12 tons for 60 mesh, 0.5 to 7 tons for 80 mesh, and 0.25 to 5 tons for 100 mesh. This mill is used for grinding coal and coke for foundry facing, shale, talc, siennas, ochers, and umbers. For very fine grinding they operate in closed circuit with a bolting reel or, in more recent installations, with an air separator.

The *Ross conical buhrstone mill* is especially adapted where the available head room is small. The material ground in this mill remains fairly cool, as the grinding is accomplished on a small circumference at high speed so that the material passes through the stones very quickly. The stones can be adjusted for fine or coarse grinding and granulating or for cracking materials as hominy and samp. Table 19 gives the operating characteristics for this mill.

Table 19. Operating Characteristics of Ross Conical Buhrstone Mill

Size, in.	Speed, r.p.m.	Max. horse power	Capacity, lb. per hr.				
			Bark	Casein	Cinnamon	Cork	Soy beans
16	2200	25	300	...	1200	300	1500
20	1800	30	550	400	....	500	2600
24	1700	50	1200	800			
30	1450	85					
36	1150	100	....	...	....	2000	

The *Robinson Unique* mill is equipped with arms slightly curved toward the closed head and forming a cup-shaped cavity back of the arms adjacent to the feed spout. The material passes down the spout and enters the cavity where the centrifugal force tends to move it through the eye and keep a continuous flow of material to the grinding plates. It is also equipped with a tramming device to overcome the possibility of the disks being thrown slightly out of tram by stalling the mill, or by some unusual strain. Operating characteristics are shown in Table 20.

Table 20. Operating Characteristics of Robinson Unique Attrition Mill

Size, in.	Speed, r.p.m.	Max. horse power	Capacity, lb. per hr.			
			Cassava root	Peanuts	Borax	Mustard
16	2400	15	800	1,800	....	850
18	2200	20	....	....	2800	
22	1900	30	....	3,500		
26	1700	40	2200	4,500	....	2500
32	1400	80	....	8,200		
36	1300	100	....	10,000		

The *Quaker City* drug and chemical mills are used for grinding strychnine seed, nux vomica, albumen, larkspur seed, sarsaparilla root, slippery-elm bark, gum benzoin, myrrh, gum guaiacum, vanilla beans, and the general run of seeds, herbs, spices, and chemicals. Operating characteristics of several sizes are given in Table 21.

Table 21. Operating Characteristics of Quaker City Mills

Type	Capacity, tons per hr.	Speed, r.p.m.	Weight, lb.	Horse power
G- 6½	50–100	175–250	125	1– 3
K- 7	200–350	200–300	174	1– 3
C- 8	300–450	200–400	250	3– 8
H-12	450–600	200–600	400	8–15

This type is also made for hand power with capacities from 20 to 75 lb. per hr. The laboratory mills are made with interchangeable grinding units and may be used for both wet and dry grinding. Changing of units may be done quickly, and it is advisable to keep several units on hand, one for each type of material handled. Operating characteristics are given in Table 22.

Table 22. Operating Characteristics of Quaker City Laboratory Mill

Size of pulverizer	Model 4-E
Power required (total)	¼
Weight, including motor, lb.	150
Length of base, in.	27
Width of base, in.	12
Height of mill, in.	16
Fineness of product, mesh	75–100

The *Bauer attrition mill* with suspended motor construction allows the rotors always to be in direct center and ensures uniform air gap or spacing. Plate diameters vary from 19 to 36 in. for the various size mills.

The *Frigidisc grinder* is a rugged heavy-duty attrition mill developed for the rubber reclaiming industry. The mill is especially suitable for products that must be ground with a minimum rise in temperature, such as reclaimed tire-tread scrap, synthetic rubber, and other products of a tough, resilient nature. Both the stationary and moving grinding disks are cooled with a circulating liquid so that a high pressure can be exerted on them. The mill can be driven with a 25-h.p. motor.

Hammer Mills (Fig. 1*c*). This type of mill consists of a horizontal or a vertical shaft, rotating at high speed, on which is mounted the crushing element, which may consist of hammers (fixed or pivoted), bars, or rings. Hammer mills may be used to produce a finished product or to produce a feed suitable for a tube mill. Most types reduce the material by impact, while the material is in suspension, driving it against a breaker plate where a great deal of reduction takes place. A cylindrical cage or screen is placed under the hammers. When the material is reduced to the proper fineness, it is discharged through the openings in the cage. The cage may consist of a perforated or slotted screen or of bars arranged in a manner similar to grate bars. The fineness of the finished product can be regulated by changing the spacing of the cage bars and also by lengthening or shortening the hammers. The character of the finished product is determined to a large extent by the shape of the cage bars and the shape of the hammers. The hammer mill is made in a great many types and sizes and can be used on a greater variety of materials than any other type of machine. It can be used as a crusher, taking run-of-mine coal or rock 48 in. or over in size and reducing it in one operation to 1½ in. and finer. On the other hand, this type of machine is used for producing a finely pulverized material, taking ½-in. feed and reducing it to a practically 200-mesh product. The type capable of producing an extremely fine product is usually equipped with air separation, particularly if the product to be reduced is hard.

Table 23. Operating Characteristics for Williams Hammer Mills

Type	Size No.	Throat opening, in.	Capacity, tons limestone per hr.				Horse power
			2 in.	1¼ in.	¾ in.	Asphaltic, ⅛ in. or finer	
Super-Jumbo	4	24 × 16	125	100	60	20	100
Super-Jumbo	6	30 × 20	210	165	100	35	175
Super-Jumbo	8	30 × 26	310	250	150	50–55	250
Super-Jumbo	10	30 × 30	400	330	200	65–70	350
Camel Back	4	20 × 24	15–20*	50–60	....	20–25	75
Camel Back	6	36 × 24	30*	100	....	30–35	150
Camel Back	8	60 × 24	55–60*	200	....	45–50	225
Jumbo Junior	1	17 × 24	6–10*	15–20	8†	8–10	25–35
Jumbo Junior	2	22 × 24	10–15*	20–30	12†	12–15	40–50
Jumbo Junior	4	42 × 24	20–25*	40–50	22†	20–25	75–90
Jumbo Junior	6	62 × 24	30–35*	60–75	32†	35–40	150

* Capacities based on producing the average grade of agstone.

† Capacities based on producing waste glass 1 in. and under.

The *Williams* machines are built in many types and sizes and may be equipped with various adjuncts permitting the use of wet, sticky feed. In the roll type, the breaker plate is replaced by a revolving roll which keeps the incoming material agitated. The roll may be heated by steam or electricity to prevent adhesion. For wet material no cage is used for regulation of the fineness, but the cages are replaced with corrugated grinding plates. Operating characteristics for several types of Williams hammer mills used for crushing large rock are given in Table 23.

With certain modifications the Williams mills may also be used for finer crushing, pulverizing, and shredding, the characteristics of the finished product being controlled by varying the size and shape of screen and cage bars and the size and shape of the hammers. Straight hammers and pointed hammers are well adapted for granulating work, production of a material with minimum fines. Twist hammers and chisel hammers are other types used; the latter are used for shredding and tearing. The Williams coal crusher is of the hammer mill type and is made in sizes from 2 to 3 tons up to 500 tons per hour. These machines may be equipped with dumping cages permitting dumping of clogged material if the pulverizer becomes choked.

The *Stedman* two-stage swing hammer mill has a series of disk plates separated by spacers; each disk has a series of holes for radial adjustment of worn hammers. Various types of hammers are used: double-end notched hammers for fibrous material where an excess of fines is required; a chisel-edged type which gives a shearing and splitting action; a heavy-duty yoke hammer for reducing large lumps of hard material. Rings are sometimes used instead of hammers especially for abrasive materials.

In addition to these types, the smaller Samson and Trojan series produce a high percentage of 100- to 300-mesh material. They are used for grinding lime and gypsum in conjunction with a finishing mill and screen or air separator.

Table 24. Operating Characteristics for Jeffrey Swing Hammer Pulverizer

Size of machine, in.	Capacities, tons per hr.								Horse power
	Limestone		Coal		Burned lime		Gypsum		
	⅛-in. bar opening	1-in. bar opening	⅛-in. bar opening	1-in. bar opening	⅛-in. bar opening	1-in. bar opening	⅛-in. bar opening	1-in. bar opening	
Type A:									
15 × 18	½– 1	1– 2	¾– 1	2– 3	1– 2	2– 3	1 – 1½	1½– 2	7– 10
20 × 12	1 – 2	3– 4	1½– 2½	4– 6	2– 4	4– 6	2 – 3	3 – 5	12– 15
24 × 18	2 – 3	4– 6	3 – 4	8– 10	4– 6	8– 10	3½– 5	6 – 8	20– 25
30 × 24	4 – 5	7– 8	5 – 6	15– 20	7– 8	12– 15	6 – 7	10 – 12	30– 40
36 × 24	8 –10	14– 16	10 –12	30– 35	14–16	25– 30	12 –15	20 – 25	50– 60
36 × 30	12 –15	20– 25	15 –18	40– 50	20–25	35– 40	18 –20	30 – 35	65– 75
Type B:									
20 × 12	2 – 2½	4– 6	2 – 3	6– 8	3– 4	6– 8	3 – 4	5 – 7	12– 20
36 × 24	10 –12	25– 35	12 –15	40– 40	15–20	35– 40	15 –18	30 – 35	50– 75
42 × 48	30 –40	125–175	40 –50	140–175	50–60	80–100	40 –50	75 –100	150–250
54 × 48	....	125–200	....	175–200	....	200–250	....	175 –225	200–350

The *Jeffrey swing hammer mill* is a ruggedly built machine. Type A is a general purpose machine; type B is of extra-heavy construction and well adapted for severe duty. Operating characteristics for the two types are shown in Table 24.

A screen analysis of the products made with bars spaced as indicated is given in Table 25.

The *Jeffrey Type A Shredder* is made in five standard sizes, from 20 by 12 to 36 by 30 in. and 15 to 90 h.p. Handling pine chips with a ¼-in. bar opening, these machines have an hourly capacity from 1 to 8 tons; with ⅜-in. bar opening, the corresponding capacity for tan bark is 1 to 7 tons per hr.

Table 25. Screen Analyses of Products from Jeffrey Swing Hammer Pulverizer

Material	Bar opening, in.	% through sieve								
		½ in.	⅜ in.	¼ in.	⅛ in.	10 m.	20 m.	35 m.	65 m.	100 m.
Limestone	⅛	..	..	..	99.8	95	81	61	46	36
	1	95	90	80	58	40	30	20	15	10
Burned lime	⅛	..	..	..	....	98	92	80	71	65
	1	97	93	88	72	60	51	41	36	32
Bituminous coal	⅛	..	..	..	....	94	74	44	26	17
	1	97	93	86	68	48	33	18	9	6
Gypsum	⅛	..	..	..	....	97	87	69	53	42
	1	99	98	93	75	58	48	35	25	18
Scrap fire brick	⅛	..	..	..	....	96	78	42	20	9

Table 26. Operating Characteristics of Jay Bee Pulverizer

Material	Type and size	Screen size, in.	Capacity, lb. per hr.	Horse power
Asbestos	No. 2 Standard	3/32	900	25
Bones, dry	No. 3 Junior	1/16	3400	15
Chicle	No. 2 Junior	¾	300	10
Chili pods	No. 2 Junior	3/32	300	10
Cocoa-bean shells	No. 3 Direct-connected	1/32	1850	50
Cocoa-bean shells	No. 3 Direct-connected	1/20	2700	50
Cocoa-bean shells	No. 3 Standard	1/20	4000	50
Copra	No. 1 Standard	3/16	1000	20
Copra	No. 2 Standard	3/16	2000	25
Glue	No. 1 Standard	1/32	1500	20
Malt	Midget mill	1/64	250	3
Leather scrap	No. 2 Direct-connected	7/16	200	35
Mace	Midget mill			
	First grinding	¼		
	Second grinding	1/16	175	3
Nutmeg	No. 3 Junior			
	First grinding	¾		
	Second grinding	¼	170	10
Sawdust	No. 1 Standard	3/16	1000	20
Tankage	No. 2 Standard	¼	4000	30
Tobacco stems	No. 2 Humdinger	5/16	1000	40

Type E shredder comes in two sizes, 80 to 90 h.p. and 120 to 130 h.p., capacity per hour 10 to 12 tons and 16 to 18 tons, respectively, with ¼-in. bar; and type B in six sizes, from 15 to 300 h.p., shredding wood for blower systems at a rate of about ½ ton per hr. for the smallest size, and 12 to 15 tons per hr. for the largest.

The *Jay Bee pulverizer* is of the screen-hammer-mill type. It has a horizontal shaft rotating at high peripheral velocity in ball bearings and has disks mounted on the rotor, between which are double-hinged hammers. It usually operates in connection with an air conveyor. Operating characteristics are given in Table 26.

The *Pennsylvania hammer mills* find use for size reduction of cement rock, limestone, and shale in the large tonnages required in cement-mill practice and in limestone and gypsum operations. They are of steel construction and have built-in tramp-iron separators.

American ring crushers and pulverizers are equipped with a horizontal shaft on which are mounted steel disks and spiders or cross arms. On the faces of the disks are inserted the ring shafts on which are mounted the grinding rings. The rings are held outwardly by centrifugal force and grind by impact and attrition. Operating characteristics of several sizes of type S ring crushers, operating on run-of-mine coal, are given in Table 27.

Table 27. Operating Characteristics of Type S American Ring Crushers and Pulverizers

Number of mill	Capacity, tons per hr.					Horse power
	1½ in.	1¼ in.	1 in.	¾ in.	½ in.	
15 S	80	50– 65	45– 50	35– 40	25– 30	20– 25
24 S	150	130–140	100–120	80–100	60– 70	30– 40
38 S	300	225–250	200–235	175–200	125–150	60– 75
48 S	400	350–375	330–350	300–325	225–250	100–125
60 S	500	425–450	400–425	350–375	275–300	150–175

The *Mikro Pulverizer* embodies several distinct features, such as a special feeding device, special hammers, cooled housing and cover, and closed or adjustable air inlets. This machine is used for grinding dyes, dry colors, and fiber-press products, and may also be used for materials which tend to ignite or explode. It is also well adapted for grinding crystalline organic materials, such as sugar or citric acid. It is available in the following rotor diameters: 8, 12, 18, and 24 in., as well as a Bantam for laboratory or small-scale commercial use. A 24-in. machine, 40 h.p., reduces about 4000 lb. sugar per hr. to XXXX grade and a 12-in., 7.5-h.p. machine, 700 to 750 lb.

The *Mead mill*, used for drugs, chemicals, colors, and dyestuffs, is of the high-speed disk type. It is made in five standard sizes, capacities ranging from 100 lb. to 8 tons per hr.

The *Raymond screen pulverizer* is adapted for filter-press products, chemicals, colors, and dyes. It is easily cleaned when changing from one color or chemical to another. A screw conveyor in the bottom of the hopper forces the feed into the grinding chamber uniformly against the tips of the hammers. This screw is designed to pack the material, making it act as a seal, thus ensuring dustless operation. Operating characteristics are given in Table 28.

Larger sizes, in combination with air separators, are used for asbestos, mica, and similar fibrous and flaky materials. They cannot be cleaned so easily as the smaller sizes and are therefore used where changes of material are less frequent.

The *Ball and Jewell rotary cutter* makes use of rotating knives set at an angle to obtain a shearing cut. Material is retained in the cutter by a screen until sufficient size reduction has taken place to allow its discharge. It is useful for granulating plastics to a uniform fineness and may be used wet or dry. Other applications are for cutting fibrous materials such as roots, shavings, and leather.

Table 28. Operating Characteristics of a Raymond Screen Pulverizer

Material	Capacity, lb. per hr.	Fineness	Horse power
Para red	800	97% through 200 mesh	5
Tartaric acid	600	99% through 60 mesh	5
Soap powder	2000	30 mesh	7.5
Ultramarine blue	1500	98% through 200 mesh	7
Boric acid	1000	92% through 100 mesh	6.5
Malted milk	1500	20 mesh	5
Bismarck brown	400	98% through 200 mesh	5
Sugar	650	90% through 200 mesh	5

The *Sprout-Waldron rotary knife cutter* is made in several different styles and sizes, but the principle of operation is the same. A revolving cylinder or rotor equipped with adjustable knives rotates within an enclosure consisting of stationary knives and perforated screens. The material to be cut is passed through the knives and is cut as it falls through the perforated screens. The top and bottom screen feature serves to give a large area for cutting capacity. It is used for granulating cork, copra, and plastic materials and for flocking sulfite pulp and fibrous materials.

The *Abbe rotary cutter* consists of a rotor with attached steel knives, moving in a casing with stationary knives. It is well adapted for fibrous material such as asbestos, cocoa shells, paper, and leather, and is often used to produce a feed for attrition mills and buhrstones. It is made in six standard sizes, 1 to 50 h.p., with capacities from 25 to 3000 lb. per hr.

The term "disintegrator" is often applied to mills consisting of one or more "concentric squirrel cages" in which adjacent cages revolve in opposite directions (Fig. 1*k*). Single-cage mills are generally used for disintegrating lump materials, for instance for tempering, blending, cutting, and aerating sand. A 5-h.p. mill, operating at 235 r.p.m., has an hourly capacity of about 15 tons, a 20-h.p. mill of 135 tons. The double-cage disintegrator is well adapted for breaking sandstone, clay, and marl. A 30-in. cage, operating at 500 r.p.m., breaks about 15 tons sandstone per hour, a 54-in. cage, 350 r.p.m., 70 tons per hr.

Table 29. Operating Characteristics of Stedman Disintegrator

Cage diam., in.	Rows of bars	Speed, r.p.m.	Capacity, lb. per hr.	Horse power
18	4	1800–2000	3,000– 3,500	15–20
30	4	1000–1500	6,000– 7,000	30–35
36	4	800–1000	8,000–10,000	40–50
42	6	700– 800	12,000–14,000	60–70
48	6	500– 600	14,000–16,000	70–75

The *Stedman disintegrator* consists of a number of rows of bars (up to six), although the four-bar machine is the standard. The two cages containing the multiple rows of bars rotate in opposite directions and grind by impact. It is used for clay, colors, lithopone, molding sand, salt cake, acid phosphate, white lead, and similar products. Operating characteristics are given in Table 29.

The *vertical-beater mill* has the beaters mounted on a vertical shaft which is driven from a horizontal shaft through a set of bevel gears. Figure 1*n* illustrates the *Whirlwind* or *Centropact pulverizer* which consists of three grinding stages: top, intermediate, and bottom. Beneath the bottom or third stage is the discharge chamber from which the finished product is removed by means of a two-blade fan mounted on the vertical shaft. When grinding clays or lime this machine has a capacity of from 1.5 to 3 tons per hr. and requires from 25 to 50 h.p.

The *Rietz disintegrator* is designed for size reduction of dry or wet material and for hot, cold, sticky, or gummy services. It is essentially a vertical beater or hammer mill in which the vertical rotor is direct connected for operation at 1800 or 3600 r.p.m. Rigid hammers on the rotor may have widely varying size, shape, number, and arrangement. The hammers are usually staggered with shorter ones at the feed end. Screens of various sizes may be used so as to give different vertical and peripheral exposures. The cutting screen is held in place by a backing screen. Wire cloth can be used for wet disintegration and homogenizing. Uses include disintegration of sugar, resins, oyster shells, vegetables, and fish.

The *Aero* and *Riley Attritor* mills are beater mills of the unit type for coal grinding. These are described on pp. 1964 and 1965.

Colloid Mills. *Theory.* The colloid mill was originally intended for the preparation of colloidal suspensions. Later on, it was developed for emulsifications, dispersions, and homogenizations. The *Plauson mill* was the first commercial type, although von Weimarn had built a laboratory unit twelve years prior to the introduction of the Plauson mill. Von Weimarn's original idea was that colloidal dispersion could be obtained by mechanical grinding or a shearing action in the presence of some stabilizer exerting a protective action on the particles produced. Plauson believed (A. W. Kenney, *Chem. Met.*, vol. 27, No. 22) that colloidal solutions would result if any massive material could be subdivided to particles between 1 and 0.01μ in diameter and suspended in a non-conducting medium. He therefore singled out the impact mill as the most promising type. His mill utilized two principles now fundamental for this type: (1) To obtain extremely fine disintegration, the crushing force must move at the highest possible velocity, and the mill, therefore, rotates at a very high speed. (2) It is essential to disintegrate in a liquid medium which serves to propagate the force of the blow. Accordingly, the charge feed to the mill consists either of two or more liquids or of a finely ground solid mixed with a large excess of the liquid used as dispersing medium.

Plauson claimed that if the peripheral velocity is less than 10 m. per sec., no better effect is obtained than by grinding in air (1μ in size under best condition); with 30 m. per sec., particles 0.1μ were produced and at 40 m. per sec., true colloidal solutions. Higher velocities were recommended to accelerate the action. The use of such high velocities caused mechanical difficulties, and the mill was therefore modified so that the moving parts would rotate in opposite directions in a manner similar to that employed in the ordinary attrition mill, thus maintaining the same relatively high peripheral speed with half the original velocity. Figure 3 shows a mill designed in accordance with this idea, known as the friction type or the English Plauson. The solid material to be ground, suspended in the liquid medium, is fed at *A* through the hollow rotating shaft under pressure into the space between the rotating disks 59 and 62; after being subjected to the rubbing action it is thrown into the chamber 64, from which it flows through the outlet 65. Cooling water is introduced through a hollow shaft 72 and is discharged at 73. The original Plauson mill was generally operated as a batch machine, whereas the so-called English Plauson is operated with continuous feed and discharge.

Modern colloid mills may be classified into three main groups: the beater type, the smooth-surface type, and the rough-surface type. Included in the first type is the original Plauson mill with its various modifications one of the most recent of which is the *Oderberg mill*. In the Plauson mill a large part of the power consumed is not available for actual work but is dissipated through the friction between the material and the housing. The Oderberg design was evolved to eliminate most of this friction in order to decrease the power consumption. In this mill, the material to be milled is conducted tangentially past the beaters; the mill chamber, with the exception of the beating point, is empty, so that unlike most other types there is no rotation of the material within the housing. It is claimed that under similar conditions the Oderberg mill requires only 15 per cent of the power consumed by the Plauson mill.

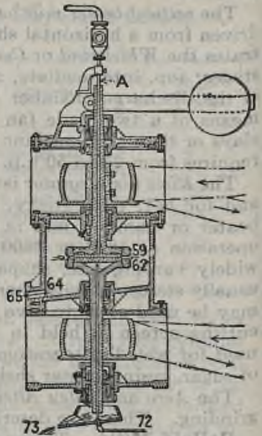


FIG. 3.—Vertical-friction-type colloid mill.

Smooth-surface mills depend only upon hydraulic shearing forces; machines of this type consist of a smooth moving surface which passes a smooth fixed surface at a high velocity (12,000 ft. per min. or higher). Films of fluid on both surfaces cause tremendous shearing stresses in the intermediate thin layer of liquid. These stresses disperse the particles suspended in the liquid. Operation is continuous, but the rate of flow between the working surfaces differs in the various types. One variation maintains a steady flow of material past the working surface by means of centrifugal force, while a second modification has its working surfaces so shaped that the movement of the passing material is delayed. Centrifugal smooth-surface mills have as their working surfaces either a truncated cone rotating within a fixed cone, or flat disks rotating in opposite directions.

In the *Premier colloid mill* (Fig. 1*m*) the rotor speed varies from 1000 to 20,000 r.p.m., with a minimum clearance of 0.002 in. The rotor acts as a centrifugal pump and draws the material continuously and with great force through and between the conical surfaces. The mills are furnished with a jacket that permits heating or cooling during operation. It is generally direct-connected to a 3600-r.p.m. motor and attains the following peripheral speeds: 5-in. mill, 5000 ft. per min.; 10-in., 10,000; 15-in., 15,000; and 20-in., 20,000 ft. per min. Operating characteristics are given in Table 30.

The *Hurrel mill* is of the delayed smooth-surface type and is used for dispersion of solids, plastics, and liquid masses. The No. 1, type B adjustable mill has a speed of 11,300 ft. per min., 3600 r.p.m., a clearance of 0.002 to 0.05 in., hourly capacity 300 to 600 gal., and 20 to 30 h.p. power consumption.

The rough-surface colloid mill differs from the smooth type only in one respect, the use of roughened working surfaces, which adds to the hydraulic shearing action a dispersing action of intense beating and turbulence, produced by eddy currents in the liquid between the irregular surfaces.

Table 30. Operating Characteristics of Premier Colloid Mill

Type	Size, in.	Rotor diameter, in.	Speed		Capacity, gal. per hr.	Horse power
			R.p.m.	Ft. per min.		
Liquid (*a*)	5	5	13,000	17,000	50- 300	2 - 10
Paste (*b*)	5	5	3,600	5,000	7- 50	0.5- 2
Paste (*c*)	10	10	1,800-3,600	5,000-10,000	50- 200	2 - 50
Liquid (*d*)	15	15	1,000-5,000	4,000-20,000	500-2000	25 -100

Some of the materials handled are (*a*) emulsions, inks, chocolate drinks, and milk of magnesia; (*b*) mayonnaise, heavy emulsions, toilet creams, dye pastes, and ointments; (*c*) asphalt, emulsions, ointments, paints, and tooth paste; (*d*) milk of magnesia, insecticide, emulsions, and inks.

Table 31. Operating Characteristics of Charlotte Colloid Mill

Size of mill No.	Capacity, gal. per hr.	Speed, r.p.m	Horse power
450	20- 100	3600	5- 10
650	100- 350	3600	15- 20
725	200- 400	3600	25- 30
800	300- 600	3600	35- 40
850	400-1000	3600	40- 50
1050	1000-5000	3600	85-100
Laboratory	5- 20	3600	1- 2
No. 7	20- 100	3600	4- 7
No. 20	100- 400	3600	18- 20
No. 50	400-1000	3600	35- 40
No. 75	1000-5000	....	60- 75

The *Charlotte mill*, shown in Fig. 4, consists of a grooved conical rotor *A* within a correspondingly grooved stator *B*. It has gravity feed, and the material is forced into the gap between rotor and stator by centrifugal force. In the illustration the inlet is *G*, the outlet *H*, the shaft *J*, and the water jacket *L*. This mill is used for reduction of solids carried in a fluid medium, for homogenization of immiscible liquids, and dispersion of plastic materials. Operating characteristics are given in Table 31.

Capacity ratings depend on the nature of the product and the result desired and not upon the amount of material forced through the mill. As the rough-surface mill does not depend entirely upon hydraulic shear, the Charlotte mill can operate at a slower speed than the smooth-surface type; it is usually connected directly to a 3600-r.p.m. motor.

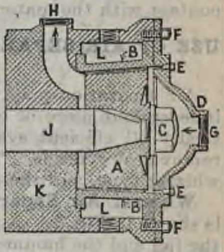


Fig. 4.—Charlotte colloid mill.

The *U.S. colloid mill* has a vertical and a horizontal design. One feature of this mill is that the material can be held in the grinding sphere for any length of time necessary to obtain the desired results. It is surrounded with a water jacket for heating or cooling. The vertical type is more suitable than the horizontal for grinding and mixing heavier pastes, creams, and pigments.

The chief applications of the colloid mill may be summarized as follows:

1. Emulsification of vegetable, animal, and mineral oils, resins, waxes, and similar materials.
2. Intensive mixing of liquids and solids and the blending of various chemical preparations, the constituents of which are difficult to mix.
3. Minute dispersions of solids in liquids.
4. Extraction and washing of oils.
5. Extraction of fibrous material with water or solvents.
6. Incorporation of pigments in the more fluid paints and enamels.
7. Grinding of color pastes, of oil or water with special application to the textile and leather industry.
8. Regeneration of milk and cream from sweet butter and milk powder.
9. Fine dispersion of water and solids in edible fats.
10. Extraction of oils, fats, resins, and juices from vegetable or animal tissue.
11. Manufacture of "water paints" and distempers.
12. Manufacture of pharmaceutical mixtures, ointments, and emulsions.
13. Manufacture of toilet creams, lotions, and shaving creams.
14. Manufacture of printing inks and some types of lithographic inks.
15. Dispersion of pigments, fillers, and oils, in rubber latex.
16. Manufacture of mayonnaise, creams, sauces, and other foodstuffs.
17. Manufacture of asphalt emulsions for paint, roofings, and floors.
18. Preparation of paper pulp and fiber for roofing felts and paper stock.
19. Recovery and deinking of pulp from paper.
20. Selective separation applying to the removal of clays and graphite and other materials.
21. Speeding up chemical reactions.
22. Pharmaceutical extractions such as tannin, ginger, and similar drugs; quinine from cinchona bark, cocaine from cocoa, caffeine from tea.

It should be borne in mind that the colloid mill must generally be used in conjunction with a pulverizer of some kind, particularly where the raw material contains particles of appreciable size. The feed should be about 80 mesh, or the largest particle should not be much larger than 175μ. The choice of pulverizer as a preliminary to a colloid mill is of great importance as it is evident that proper preparation of the feed reduces the amount of work to be performed by the colloid mill proper, with resulting increase in capacity, improved dispersion, and longer life of the parts which come in contact with the material.

USE OF AIR SEPARATION WITH CRUSHING AND PULVERIZING EQUIPMENT

Air separation may be applied in two different ways: extraneously as an independent piece of equipment, or as an integral part of the mill. Practically all efficient systems employ the principle of centrifugal action for removal of oversize. They are divided into two types: the counteraction or whizzer type, and the multi-vane or deflector type.

Whizzer Air Separation. The whizzer type, applied to a hammer mill, is shown in Fig. 5. A fan is placed on one end of the hammer shaft; between the fan and the hammers is the whizzer consisting of two or more thin blades with tips tapered to conform to the housing. Distance between blades and housing is regulated by moving the whizzer along the shaft. The action of the whizzer is that of a fan wheel opposing the action of the main fan. With minimum clearing between blades and housing, a maximum countercurrent is set up at the periphery in the direction indicated by the arrows. An air current accompanying the feed dropping into the pulverizing chamber carries the pulverized material through the clearing toward the fan intake. As the centrifugal force is greater on the coarser particles, their radial velocity will exceed the lateral; hence they are thrown to the periphery and deflected to the hammers by the countercurrent, while the finer particles discharge through the fan intake. As the whizzer is moved toward the hammers, a coarser product results.

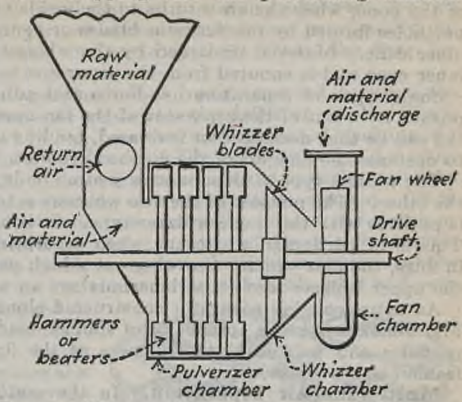


Fig. 5.—Whizzer air separation as applied to a Raymond hammer pulverizer.

The outline given presents only the fundamentals of whizzer air separation. A great deal depends upon the details in design, such as relative sizes of the main fan and whizzer, the exact taper of whizzer blade tips and of housing, diameter of whizzer, and number and width of blades. The all-important factor is the taper of the housing in the whizzer chamber, as it determines the effectiveness of the countercurrent of air and the deflection of the coarser particles to the hammers. This is of utmost importance when an extremely fine product is desired. The separated product passing through the whizzer enters the fan and is blown to a cyclone collector where it is discharged into bins or containers. The return air goes back to the pulverizer, completing the cycle.

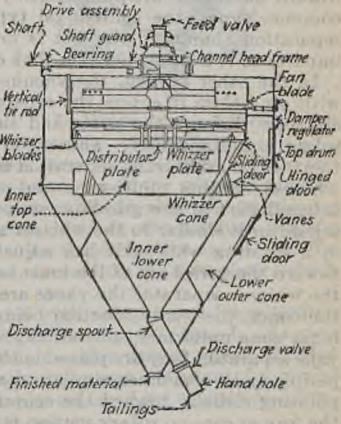


Fig. 6.—Centrifugal or mechanical air separator.

A cross section of an extraneous air separator with whizzer separation is given in Fig. 6. It consists primarily of two cones with an annular space between them. A hollow vertical shaft extends through the top of the separator, at the center, down into the inner cone. On the shaft, near the top, is mounted the main fan; below the fan is the whizzer, and below the whizzer the distributing plate. As this plate rotates at a high velocity, the centrifugal effect on the material throws it radially, in a uniform stream, in the space between the edge of the plate and the inner surface of the inner cone, into the path of an upblast of air. The dust-laden air passes up through the whizzer where the coarser particles are eliminated, and discharges through the fan into the annular space between the cones. The rotating fan blades throw the material to the inner surface of the outer cone, and it is discharged through a valve in the bottom of the cone, while the air returns to the inside of the inner cone through the portholes formed by the deflector blades or vanes, placed radially around the inner cone. Material discarded by the whizzer drops to the bottom of the inner cone and is spouted from the separator as tailings.

Some types of separators use horizontal adjusting plates which may be pushed in or out of the separator at the fan opening. The opening into the fan can be thus decreased or increased, tending to some extent to increase or to decrease the fineness of the finished product.

The whizzer type of air separator is often built with two whizzers one above the other. The purpose of the two whizzers is to obtain a finer product than is possible with the single-whizzer type. It is in fact a two-stage separator. This is of particular advantage when separating a material which is poor in fines, the first whizzer throwing out a high percentage of oversize, so that the upper whizzer receives a richer mixture on which to operate.

An extraneous air separator constructed along the same lines as shown in Fig. 6 makes use of a spinner or of spinner blades for precipitating oversize particles and to control the fineness of the finished product in the same fashion as the whizzer.

Multi-vane Air Separation. In the multi-vane (deflector) type, the objective is to spread the pulverized material on a revolving sheet of air; the coarser particles are thrown to the periphery of this sheet against the housing, fall down, and are discharged through a spout in the bottom. A rotating movement of the air is attained by compelling it to enter the chamber through a series of portholes guided by tangential deflector vanes. The fines, traveling toward the center, are pulled out by a fan and discharged. In one of the two common types, the centrifugal type, the material is fed directly into the separation chamber, where it is met with the upward traveling air which imparts to it a rotary motion that causes separation.

In the other system, the vacuum type, the pulverized material is mixed with air before it enters the separation chamber. The mixture enters the chamber through the ports and the rotary motion is imparted to air and solids simultaneously, whereupon separation takes place as described. Centrifugal multi-vane separation is applied only to an extraneous separator, whereas vacuum multi-vane and whizzer separation can be applied as an integral part of the grinding equipment. In general design the multi-vane separator is similar to the whizzer separator, with the exception that instead of a rotating whizzer it has adjustable vanes that can be set tangentially toward the periphery of the inner cone, giving the air a rotating motion. In the whizzer separator the vanes are fixed rigidly in a radial position between the cones, their only function being to permit the dust-free air to enter the inner cone uniformly and radially with minimum resistance. In the multi-vane separator, they are placed in the inner or the outer cone, pivoted near the periphery in a manner similar to a louver, so that they can be opened by pointing radially toward the center or closed by being set at an angle. In the former case no rotary motion is imparted to the incoming air, the centrifugal action is at a minimum, the least amount of separation takes place, and the finished product is the coarsest. As the vanes are closed, the centrifugal action increases, more material is thrown off, and the product becomes finer. To some extent the fineness may also be regulated by adjusting the opening into the fan.

The general design of a vacuum multi-vane separator is shown in Fig. 7. It consists of a double cone, the outer cone being shown open in the illustration. Suction is maintained through a sleeve at the top. In actual practice, the separator is usually placed on top of the pulverizer, or connected to it with a pipe, admitting the pulverized material to the space between the two cones by the upward blast of air, which enters the inner cone guided by the deflector vanes. The greater the angle of these vanes, the finer the separation.

The air containing the pulverized product travels upward between the two cones and enters the inner cone guided by the deflector vanes. These vanes may be in pairs, as shown in Fig. 7, or they may be placed continuously around the inner cone in the form of a louver. They are pivoted and may be rotated through a definite angle. This adjustment is made from the top of the separator by means of a pointer running over a calibrated segment. With the pointer at zero the deflector vanes point radially toward the sleeve or intake pipe at the center. There is no rotating motion of the air and it travels directly toward the center. As the air expands in the inner cone, some of the oversize drops to the bottom; the remainder enters the intake pipe and goes to a cyclone where it is discharged as finished product. This arrangement gives the coarsest separation. As the point is rotated, the deflector vanes also rotate, causing the material to enter the inner cone at an angle and imparting a rotating motion. Each particle is then acted upon by two forces, the centrifugal force of rotation and the force tending to move it radially toward the intake pipe, induced by the greater suction. Particles above a certain size, upon which the action of former force is greater than that of the latter, are thrown to the periphery and discharged through the bottom of the inner cone. The greater the angle of the deflector vanes, the farther the material is thrown from the intake pipe, and the nearer the periphery. More and more oversize is thrown out, until the deflectors reach the extreme angular position where the cyclonic action becomes so great that the maximum amount of material is thrown by centrifugal action; the product floating against this action is then extremely fine. As the deflectors are closed, the effective area of the portholes is diminished, decreasing the flow of air, which also tends to make a finer separation.

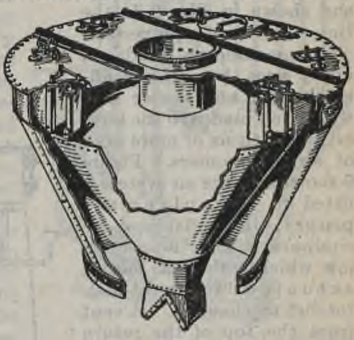


Fig. 7.—Multi-vane vacuum air separator.

The bottom of the inner cone is equipped with a flap valve for returning oversize. When the inner cone is empty the suction keeps it closed. As soon as the tailings accumulate to a definite height, their weight opens the flap valve, causing the tailings to flow without admitting any unseparated material or air. In some installations the flap valve is replaced by a positive-driven star roll which discharges the tailings continuously.

The most recent development in air separation has been the use of rotating-whizzer separators instead of the double-cone vacuum type. Its design and method of installation are shown in Fig. 8 as applied to the roller mill. The whizzer disks rotate in a horizontal plane. They may consist of one or two disks in number, each one fitted with multi-whizzer blades. The whizzer is driven through a variable drive and the fineness of the finished product is regulated by the speed of the whizzer. The faster the whizzer rotates the more particles are thrown out and the finer becomes the finished product. The raw material passes up from the grinding surfaces vertically and, before passing out of the machine at the top, is compelled to pass between the rotating-whizzer blades. The oversize is returned to the periphery and dropped back for further grinding while the desired fines pass up through the whizzer blades into the duct leading to the cyclone collector. This type of separation may be applied to any of the air-separation pulverizers wherein a current of air conveys the pulverized material.

The Imp Pulverizer. In the discussion of the whizzer type of air separation in connection with hammer mills, the mill described in the text and shown in Fig. 9 is the Imp type of pulverizer. This machine is made in many sizes, from the smallest having one beater disk with fixed blades to the largest size with six or more rows of swing hammers. Figure 9 shows how the air system is fitted to the pulverizer proper; the machines are equipped with a hopper below which is the star feeder, actuated by a pawl-and-ratchet mechanism. A vent from the top of the return air pipe passes through a tubular dust collector which makes the operation of the system dustless.

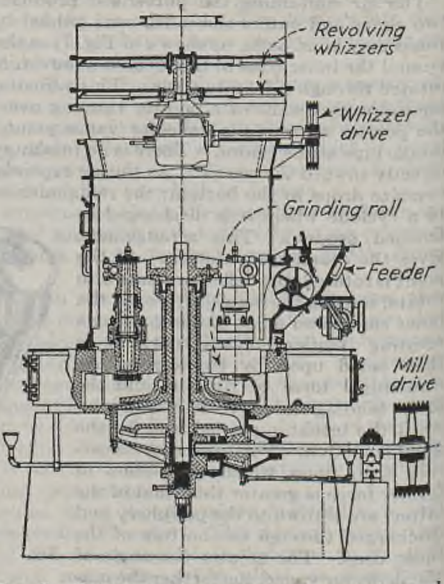


FIG. 8.—Rotating-whizzer separator, as an integral part of the Raymond roller mill.

The *Hardinge air-swept conical ball mill with superfine air classifier* is used for fine dry grinding of abrasives, minerals, ores, and chemicals. A description of the mill and classifier is given on p. 1905, followed by a table of operating characteristics.

Cylindrical ball mills or *tube mills* can be operated in closed circuit with extraneous separators with or without air sweeping, as shown in Fig. 14 for the same purposes as for the mills with integral classifiers.

The *Schutz-O'Neill Limited mill* consists of two or more beater plates on which are mounted beater blades or fixed hammers which pulverize the material against a shell which usually is corrugated. Sometimes a perforated shell, similar to a grater, is used. This is of advantage in pulverizing tough, fibrous material. Fineness of the finished product is regulated by a cone with attached blades or whizzers. The pulverized material travels longitudinally toward the cone and whizzer. When the cone plate is moved into the mill, a coarser product is obtained. With the plate in the extreme outer position, flush with the end of the mill, only the fines pass through, while the coarser particles are thrown into a tailings groove on top of the grinding chamber.

at the end of which a rotary valve returns it to the feed end of the mill. For very fine grinding and for extremely hard material, a perforated plate mounted on the shaft in front of the beater is employed. A row of holes in the plate, varying from ½ to ⅛ in. in diameter, provides an exit for the finely ground material and relieves the air pressure, thus permitting cooler operation. Table 32 gives the capacities and power requirement of the 20-in. limited mill, grinding a variety of materials.

Automatic Pulverizer. The *Raymond automatic pulverizer* is a beater-type machine equipped with air separator of the vacuum multi-vane type. It has a horizontal shaft on which may be mounted one or more disks fitted with beater blades. On the door of the pulverizing chamber is mounted an automatic throwout, the function of which is to remove refractory materials contained in the feed, such as sand and gravel from clay. A star feeder with pawl-and-ratchet mechanism receives the raw material from a stock bin and drops it into the pulverizing chamber, on top of which is mounted the air separator. The air enters the pulverizing chamber at the rear and removes the pulverized material. Particles of proper fineness are blown to the cyclone, which discharges to bins or containers, while oversize is returned to the pulverizer through the bottom valve of the inner cone. Impurities accumulate in the grinding chamber until they are picked up by the rapidly revolving beaters and thrown through the slot on the door into the throwout chamber, where they are finally rejected through the flap valve. The slide damper on the top of the throwout may be adjusted to admit air from the atmosphere which enters the pulverizer through the slot in the door. In its travel through the throwout the air cleans the rejects and blows fine particles back into the pulverizing chamber.

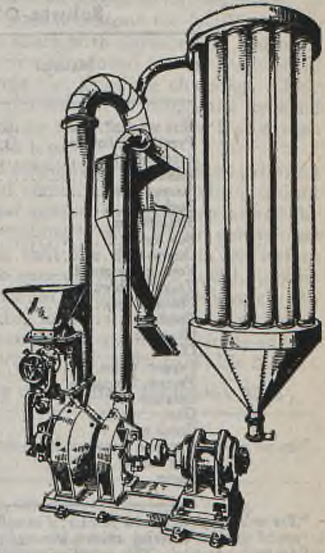


FIG. 9.—Raymond Imp pulverizer with whizzer air separation and dust conveying and collecting system.

The automatic throwout is often used for concentrating valuable constituents in an aggregate, such as phosphatic granules, which are separated from silt and clay. In this case the product discharged through the throwout is the valuable material. The automatic pulverizer can be made to function as an extraneous air separator, receiving pulverized material from another machine. The beaters are run at comparatively low speed, doing no pulverizing but stirring the material and throwing it into an air current. The air carries the material into the separator in the normal manner, the fines are removed, and the rejects eliminated through the throwout.

In the *Williams impact mill with air separator* actual grinding is accomplished with a hammer mill. The air separator extracts material of the desired size and returns the oversize to the mill for regrinding. A spinner separator may be used.

Ring Roll Mills. These mills are equipped with rolls or balls rolling around a grinding ring, pulverization being done between rolls or balls and ring. In one class, pressure is maintained with heavy springs; in another, by the centrifugal force of the rolls against the ring. In most mills, either the ring or the rolls are stationary; there is a type, however, where both rings and rolls rotate. The grinding ring may be placed horizontally or vertically. The following three classifications are made:

Table 32. Capacities and Power Consumption of 20-in. Schutz-O'Neill Limited Mill

Material	Production, lb. per hr.	Horse power
Nux vomica[*]	30– 40	15–20
Pyrethrum flowers	35– 45	15–20
Areca nuts[†]	40– 50	20–25
Quassia	25– 30	15–20
Senna leaves	100–125	20–25
Buchu	100–110	20–25
Stramonium	125–150	20–25
Bay leaves	125–150	20–25
Celery seed	125–150	15–20
Foenugreek seed[‡]	100–125	15–20
Tragacanth gum	50– 60	15–20
China twigs	75– 80	20–25
Cassia	90–100	20–25
Ginger	90–100	20–25
Cloves	125–150	20–25
Pepper, white	200–225	20–25
Pepper, Singapore	225–250	20–25
Gelatine	60– 70	15–20
Glue	100–125	20–25
Cocoa cake	400–500	25–30
Cayenne pepper	100–125	20–25
Dry colors	600–700	30–35
Sugar	400–500	30–35

[*] For ordinary commercial powder; if extrafine product is required, it should be passed through a buhr to rasp off the fiber covering which is screened out, after which the hard nux kernels are passed through the mill for final reduction. This operation applies to many seeds, beans, and nuts with tough fibrous covering.

[†] Areca nuts should be broken down in a disintegrator before they are fed to the limited mill.

[‡] Foenugreek seed should be passed through the mill quickly, sifted outside the mill, and the tailings mixed with the feed. This method is used for materials, part of which is powdered more easily than the remainder.

1. Mills without internal separation:
   - *a.* Sturtevant ring roll mill.
   - *b.* Kent ring roll mill.
2. Mills with screen separation:
   - *a.* Huntington type.
   - *b.* Bradley roller mills.
   - *c.* Griffin mills.
   - *d.* Fuller mills.
3. Mills with air separation:
   - *a.* Fuller mills.
   - *b.* Bethlehem or Whiting.
   - *c.* Raymond mills.
   - *d.* Bonnot mill.

The *Sturtevant mill* is of the first class. It has a concave, vertical grinding ring and is used for non-metallics, especially phosphate rock. A No. 1 mill with air separator grinds 2 to 4 tons limestone or phosphate rock per hour, 90 per cent through 80 mesh; a No. 2 mill has twice this capacity.

The *Kent Maxecon mill*, also of the first class, is used for bauxite, coke, limestone, magnesite, and phosphate rock. It requires 6- by 7-ft. floor space, weighs 6 tons, and consumes 25 to 50 h.p., grinding by 1000 lb. per sq. in. pressure. Hourly capacity, in closed circuit with screen or air separator, is 4 tons phosphate rock for acidulation or 10 tons limestone for fertilizer.

The *Bradley three-roll mill* has a vertical shaft with three roller journals. Grinding pressure is caused by centrifugal force produced by the rotation of the rolls. A screen of proper size surrounds the grinding chamber. This mill is also used in closed circuit with external screen, in which the internal screen is equipped with large openings. Hourly capacity is from 2 to 4 tons of limestone or phosphate rock, with 25 to 35 h.p.

The *Bradley-Hercules three-roll mill* is used extensively for semifine grinding of hard materials such as cement, rock, and cement clinker. The grinding medium consists of three rolls revolving in and against the inner surface of the horizontal annular ring against which the crushing is done by the centrifugal action of the rolls. The mill is so designed that the material is fed before each roll, allowing each grinding roll to take care of its proportional amount of feed; the mill discharges through an internal screen, down through the foundation. This machine is capable of taking material up to 2 in., reducing it to a fineness of from 50 to 65 per cent through 100 mesh in a single operation. Specifications and operating characteristics are given in Table 33.

Table 33. Operating Characteristics of Bradley-Hercules Mill

Feed	Size of feed, in.	Hourly capacity	Fineness of product	Screen on mill
Limestone..........	2½	56 tons	65% through 100 mesh; 48% through 200 mesh	8 mesh
Limestone..........	2½	54 tons	93.5% through 20 mesh; 61.2% through 100 mesh; 50.8% through 200 mesh	8 mesh
Limestone..........	2½	30– 35 tons	70% through 100 mesh	12 mesh
Limestone..........	2½	25 tons	99.3% through 20 mesh; 64.6% through 100 mesh; 50% through 200 mesh	16 mesh
Limestone..........	2½	42 tons*	32% through 200 mesh	5 mesh
Kiln-run clinker.....	...	135–150 bbl.	36% through 200 mesh	Number 1567 Ton Cap
Kiln-run clinker.....	...	150 bbl.	67.7% through 100 mesh; 42.9% through 200 mesh	8 mesh
Kiln-run clinker.....	...	210 bbl.	62%–65% through 100 mesh	8 mesh
Kiln-run clinker.....	...	175 bbl.	32% through 200 mesh	5 mesh
Kiln-run clinker.....	...	159 bbl.	42% through 100 mesh; 30% through 200 mesh	Number 595 Ton Cap

Weight of mill, lb.	60,000
Speed of vertical shaft, r.p.m.	125–130
Speed of horizontal shaft, r.p.m.	360–400
Diameter of roll heads, in.	22
Diameter of grinding ring, in.	66
Crushing effect of each roll against ring, lb.	13,000
Weight of roll head, lb.	1,000

* This output was obtained in closed circuit with Hummer screens; circulating load about 80 tons.

A 300-h.p. motor is used for grinding limestone and raw material in cement plants; for maximum capacity on cement clinker a 350-h.p. motor is preferable. On the raw end it is often advisable to pass the discharge to a screen or an air separator, removing a finished product of the required fineness, the tailings going to a tube mill the discharge of which is sent to a second separator or back to the first separator.

The *Griffin mill* has a chilled iron roll revolving against and around a grinding ring. The ground material is discharged through an internal screen by the aid of an air current resulting from the speedily revolving roll. Operating characteristics for the Giant and the Junior Giant mills are given in Table 34.

Table 34. Operating Characteristics of the Giant and Junior Giant Griffin Mills

Material	Output per hr.	Fineness of product	Screen on mill, mesh
Giant:			
Limestone	4.5–5 tons	83.2% through 200 mesh	30 and 35
Cement clinker preliminary to tube mill	26 bbl.	62%–64% through 200 mesh	6
White cement clinker	35 bbl.	72% through 100 mesh; 54% through 200 mesh	6 and 8
Coal screenings	3.5–4 tons	95% through 100 mesh	30
Limestone	3–3.5 tons	96.5% through 100 mesh; 82.5–200 mesh	35
Cement clinker	12–14 bbl.	96.6% through 100 mesh; 82.8–200 mesh	35
Air-cooled slag	7 tons	67.5% through 100 mesh; 49.8–200 mesh; 35.9% through 300 mesh	3 and 4
Junior Giant:			
Coal	1.5–2 tons	88%–90% through 100 mesh	30 and 35
Clinker	5.5–6 bbl.	80%–82% through 200 mesh	35
Firebrick	2.25 tons	91% through 100 mesh; 69% through 200 mesh	30 and 32
Phosphate rock	3.5–4 tons	35% through 200 mesh	16
Feldspar	1.5 tons	97.8% through 100 mesh; 87.6–200 mesh	30 and 35
Coke	0.5–0.75 tons	90% through 100 mesh	35
Limestone	2 tons	82%–85% through 100 mesh	16

	Giant	Junior giant
Diameter of ring, in.	40	30
Diameter of roll, in.	24	18
Crushing effect of roll against ring, lb.	15,000	7,000
Speed of pulley, r.p.m.	175	210
Weight of mill, lb.	26,000	13,000

The *Fuller-Lehigh mill* is of the vertical-ball type. It is made in two designs, with screens and with air separation. The screen type is equipped with two sets of fans rigidly attached to the yoke on the shaft of the mill, above the grinding elements. As these fans rotate with the shaft, the pulverized material is lifted from the grinding zone by the first set of fans, the upper set acting as an exhauster. It draws the fine material from the separating chamber and forces it through the screen, which completely encircles this chamber. The finished product drops through annular openings in the mill base into a lower compartment containing a fan sweep which aids in withdrawing the discharge. In the air-separation type the pulverized material is sucked from the mill by an exhauster and blown into a cyclone. Table 35 gives the capacities and power consumption of this mill, grinding a normal soft 1¼-in. limestone with maximum 4 per cent moisture for the air-separation type and 2 per cent for the screen type; the fineness of the finished material is about 80 per cent through 200 mesh.

Table 35. Operating Characteristics of Fuller-Lehigh Mill

Number of mill	Capacity, lb. per hr.		Horse power		
	Screen type	Air-separated type	Screen type	Air type	
				Mill	Fan
33	5,000	......	40		
36	......	6,000	...	60	30
42	9,000	10,000	75	75	40
48	14,000	15,000	100	125	50
54	......	20,000	...	150	60
57	18,000	......	150		
70	24,000	30,000	250	300	100

A modification of the standard Fuller-Lehigh air-separation mill illustrated in Fig. 1*l* has two grinding tracks, one above the other, with two sets of balls. The manner in which grinding takes place is essentially similar to that in the Babcock and Wilcox type B pulverizer described below. Instead of discharging the product to extraneous screens or air separators it is equipped with an integral air separator. This pulverizer is built in sizes up to 40 tons per hr.

The *Raymond roller mill* is of the integral air-separation type. The base of the mill carries the grinding ring, rigidly fixed in the base and lying in the horizontal plane. Underneath the grinding ring are tangential air ports through which the air enters the grinding chamber. A vertical shaft with a bevel gear near the bottom and resting on a thrust bearing is driven by a horizontal shaft through a pinion. Keyed rigidly to the shaft near the top is a spider which carries the roller journals. These journals have rolls on the bottom rotating on their own bearings while traveling around the ring. Two or more journals are pivotally suspended by trunnions fastened at the top of the journal housing and supported in the arms of the spider. They hang almost vertically, so that when the mill is at rest the rolls press only lightly against the grinding ring.

A recent development in connecting the air-separation system to a Raymond mill is shown in Fig. 17. Here the location of the main exhauster is such that it blows the air into the mill instead of sucking from the mill, so that the pulverized material does not pass through it.

The roller mill with air separation is used extensively for the grinding of coal, coke, limestone barytes, gypsum and practically all of the non-metallics except the siliceous materials of an abrasive character. It is also used to a large extent in the grinding of the chemical and mineral pigments, particularly where a product is required of a minute particle size.

The method of separation used with Raymond mills depends on the fineness desired. If a medium-fine product is required, up to 85 or 90 per cent through 100 mesh, a single-cone air separator is used (see Fig. 10). This consists of a housing surrounding the grinding elements with an outlet on top through which the finished product is discharged. This is known as the low-side mill. For a finer product and where frequent changes in fineness are required, the vacuum or whizzer type of separation is used. Its mode of operation is described on p. 1923. This type of mill is known as the high-side mill.

The Bethlehem (Whiting) is a mill of the roller type in which the journals carrying the rolls remain stationary in a horizontal position. The grinding element or table rotates in a horizontal plane. The grinding pressure is obtained by means of adjustable springs operating on the roller journals. The material is fed to the center of the table and owing to the centrifugal force travels to the periphery between the rolls and the rotating grinding rings. After being pulverized it is thrown into a current of air, passes up to a classifier, and is discharged in a manner generally similar to that of other types of roller mills equipped with air separation.

The *Babcock and Wilcox type B pulverizer* is a modification of the Fuller type B. It is used in closed circuit with an extraneous air separator. Raw material is led directly into the pulverizer through a feed spout and is deposited into the center of the mill on a table fastened to the rotating intermediate grinding ring. Centrifugal action of the table throws the material outward into the path of the inner row of grinding balls; it then passes to the outer row in which the second stage of pulverization takes place. From this stage the material is discharged outside the outer top row of grinding balls and passes through an adjustable ledge gap into the space below, where it enters the path of the lower row of balls, the third stage.

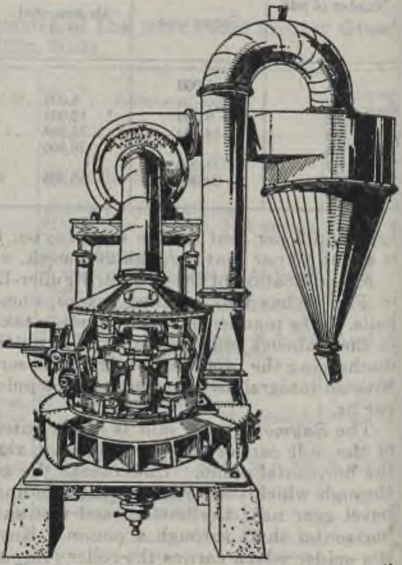


FIG. 10.—Raymond low-side five-roller mill with low-side separator and dust conveying and collecting system.

Upon leaving the lower row, the material is discharged into an annular compartment containing a number of sweeps, by the action of which it is rapidly discharged from the mill, whereupon it is elevated to the mechanical air separator.

The Jet Pulverizer. The jet pulverizer is based on a different principle from the pulverizers previously described. Figure 11 illustrates a pulverizer of this type known as the Micronizer. It consists of a shallow circular grinding chamber wherein the material to be pulverized is acted upon by a number of gaseous fluid jets issuing through orifices spaced around the periphery of the chamber. The gaseous jets introduced are generally superheated steam at pressures from 100 to 500 lb. per sq. in. in a temperature up to 500°F. superheat. Compressed air at 100 lb. pressure may be utilized instead of steam. The pulverizing chamber varies from 12 to 36 in. in diameter and from 1 to $2\frac{1}{2}$ in. in axial height.

Jet orifices are drilled through the peripheral wall of the grinding chamber and vary in number from 6 to 16, equally spaced around the chamber; the orifices vary in diameter from ⅛ to ¼ in. They are generally placed in a tangential position so that entering steam or air will promote the rotation of the material to be pulverized in the same direction.

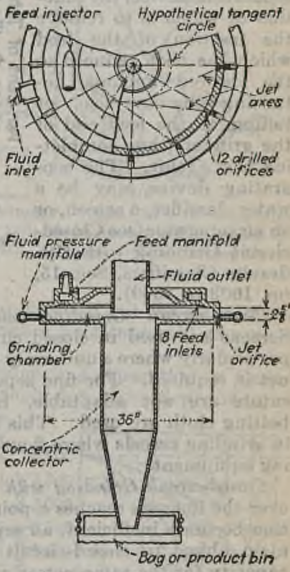


FIG. 11.—The Micronizer.

The operation of the Micronizer depends on the fact that, when a gas of high energy content is introduced into the small pulverizing chamber, the applied fluid pressure is transferred into velocity head by expansion to substantially atmospheric pressure. This causes a high-speed rotation of the contents of the pulverizer chamber. The centrifugal force of rotation causes the material to concentrate at the periphery where the jets are introduced. Since most of the energy of the fluid jets is dissipated near the point of entry, there are set up intense local velocity gradients and intense interactions within the circulating material. A great deal of reduction of the material is thus caused by the impinging of the particles upon themselves which tends to reduce the wear on the pulverizer housing. For abrasive materials the pulverizer may be lined with rubber to reduce the wear.

The gaseous fluid supplying the grinding energy is withdrawn at an inward point, tending to cause the dust-laden gaseous fluid to travel spirally. The smaller particles are thus withdrawn and the coarser particles thrown to the periphery are subjected to further reduction.

The outlet from the grinding chamber leads directly into a concentric centrifugal collector. This collector receives the material as it is traveling in a high-velocity rotary motion which is conducive to the separation of the material from the fluid so that about 85 to 95 per cent of the product is collected in the concentric collector. The remainder of the material may be filtered out from the discharged fluid with a standard air filter. An injector operating on the principle of the aspirator is utilized to introduce the feed into the outer portion of the grinding chamber. The main application of the Micronizer is for the production of an extremely fine product from a top size of 20μ down to 5μ. Certain materials may be produced where the average particle size is under 2μ. It has been found particularly applicable in the production of face powder and other cosmetics, in insecticides and in many precipitated products. It has been used for the reduction of barites, chryolite, sulfur, pyrolusite, and other non-metallic minerals where extremely small particle sizes were required and the cost of production was of secondary consideration.

The steam consumption varies with the nature of the material and the fineness of reduction. For the softer materials 1 to 10 lb. of steam are used for 1 lb. of finished product, while for the harder minerals the steam consumption may average from 20 to 30 lb. and for the harder materials, particularly where a product is required with a top size of 5μ, the steam consumption may go up to 60 to 75 lb. per lb. of finished product.

Closed-circuit Grinding. In closed-circuit grinding the grinding unit does not deliver a finished product but discharges to a separating device, the function of which is to remove the fraction of the feed which has been reduced to the desired fineness, returning the oversize, or tailing, to the feed end of the grinder, thus completing the circuit. The separating device may be a water classifier, a screen, or an air separator (see Closed-circuit Grinding with Hydraulic Classifiers, Sec. 15, pp. 1603 to 1609).

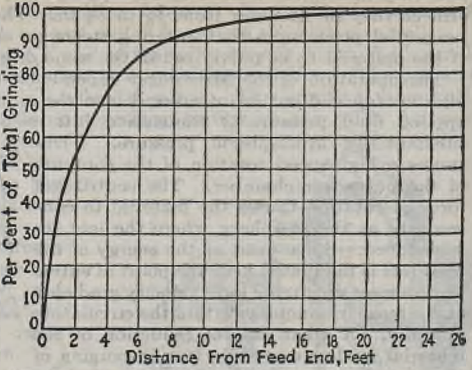


FIG. 12.—Percentage of total grinding accomplished at various distances from the feed end of a tube mill.

Closed-circuit Grinding with Screen Separation. Screens are used in closed circuit with a pulverizer, particularly where a uniformly coarse or granular product is required. For fine separation where air separators are not adaptable, bolting reels with silk bolting cloth are used. This is particularly the case in grinding cereals where flour rolls are used for grinding equipment.

Closed-circuit Grinding with Air Separation. Whenever the fineness reaches a point where screen separation becomes inefficient, air separators are used. The main object in closed-circuit grinding is to increase capacity for the same power consumption, as the mill does not have to furnish a product which is entirely within the size requirements but is allowed to discharge a product coarser than ultimately desired.

As the raw material travels through a tube mill, the greater portion is reduced in the first few feet of travel. A test was made in a 6- by 26-ft. tube mill (*Rock Products*, June 8, 1921), in which samples for screen analysis taken throughout the length showed that about 80 per cent of the total grinding is accomplished within the first 5 ft. and that the remaining 20 per cent of the grinding consumes about 80 per cent of the power consumed. The results are illustrated by the curve in Fig. 12; the results show that if a 5-ft. mill were used, the product would be within 20 per cent of the specified fineness. This product could have been fed to an air separator in closed circuit with the mill, the fines removed, and the oversize returned to the mill.

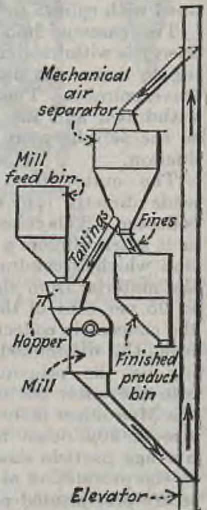


FIG. 13.—A hammer mill in closed circuit with an air separator.

It is debatable whether closed-circuit grinding does not give a product with less impalpable powder than when the mill is operated as a finishing mill. This all depends on the meaning given the term "impalpable powder." When grinding certain fillers or pigments, or even cement clinker, a tube mill, if properly operated, undoubtedly produces a higher percentage of particle under 20 to 10μ when operated in open circuit. It may not be so economical as closed circuit, but economy must sometimes be sacrificed to obtain the requisite quality. For many purposes, fineness is not the only criterion, as particle shape or density may be more important. Certain fillers seem to possess more desirable properties when they are retained in the mill until they are in their finished form. Open-circuit grinding will often give best results in grinding flaky or tough fibrous materials. Air separation may be used for such materials, but the separator is used only after the material has passed through a long tube mill or even a series of one to four mills.

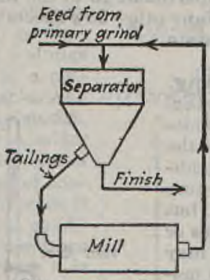


FIG. 14.—Primary grinder and tube mill in closed circuit with an air separator.

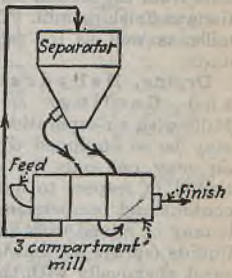


FIG. 15.—A three-compartment mill in closed circuit with an air separator.

There are many possible flow sheets for connecting separators in closed circuit with grinding equipment; the most suitable flow sheet depends upon type and number of grinding units and the nature of the finished product. A simple flow sheet of a hammer mill with air separator is shown in Fig. 13. A similar arrangement could be made with a tube mill and separator.

Figure 14 gives a flow sheet where the product from a primary grinder, instead of going directly to the tube mill, is first passed to a separator together with the tube-mill discharge. The separator removes the fines from both products, returning the oversize to the tube mill. A similar arrangement can be made for a two-compartment mill, both compartments discharging to the separator. A little different arrangement may be used for a ball mill and a tube mill served by one separator. The ball-mill discharge goes to the separator, the tailings from which are returned to the feed end of the ball mill, while the fines go to the tube mill. In this case the ball mill operates in closed circuit with the classifier, while the tube mill functions as a finishing mill in open circuit. This hook-up may also be used for a two-compartment mill, the second compartment of which may serve as finishing mill.

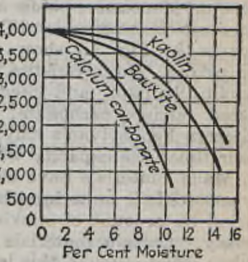


FIG. 16.—The effect of various moisture content in different materials on the capacity obtained from a pulverizer.

Figure 15 shows an arrangement used for a three-compartment mill with single separator. Tailings and fines from the separator may be returned to any compartment, depending upon the amount of fines required in the finished product. For a three-compartment mill with two separators the following arrangement may be used: The first separator receives the discharge from the first compartment and the second separator the discharge from the second compartment and the fines from the first separator; tailings from the two separators are the feed for the second compartment, while the fines from the second compartment go to the third compartment which functions as finishing mill. Many other flow sheets may be used for compartment mills as well as for separate units.

Drying, Dehydrating, and Cooling. *Drying.* Mills with air-separation units may be so equipped that the air may not only be conditioned in respect to moisture content and temperature, but it may be mixed with gases or liquids (spray) so that it may react chemically with the material pulverized. Practically all materials can be ground to better advantage when dry; in drying finely divided products especially, the capacity decreases very rapidly with increasing moisture content. The rate of decrease is not the same for all materials; some may contain an appreciable amount of moisture before any decrease is noticeable. The curves in Fig. 16 show the grinding characteristics of kaolin, bauxite, and precipitated calcium carbonate, all ground to 99.9 per cent through 200 mesh. Eastern Pennsylvania clay, ground to 100 per cent through 200 mesh, showed four times the capacity when bone dry than with 10 per cent moisture.

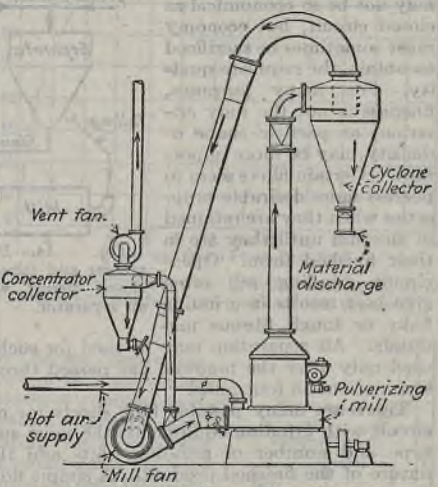


FIG. 17.—A kiln mill equipped with accessories for drying, grinding, air separating, conveying, and collecting.

Many materials have also been found easier to pulverize when hot than when cold; a material which, although perfectly dry, may be difficult to grind on account of sticking, will if heated tend to disperse and flow more freely. Certain materials are reduced faster when dried in the mill than when dried outside; this is probably a result of the rapid, almost explosive, removal of the moisture when the wet particles come in contact with the hot gas. This action tends to shatter the particles. The system used for conditioning the air is the same whether the mill is of the roller or the swing hammer type, and whether the system is used for drying, dehydrating, or cooling. A Raymond kiln mill, used for these purposes, is shown in Fig. 17. A duct carrying hot air is connected to the return-air line of the separation system. The pipe for venting the surplus air and the moisture extends from the fan discharge to a concentrator, the object of which is to collect most of the dust in the surplus air and return it to the mill, the surplus air going through a vent fan, and a vent pipe to the atmosphere. The vent fan overcomes the pressure drop in the air system. Waste gases from industrial furnaces may be used when available, except where discoloration or contamination of the product must be avoided, in which case hot air must be used. The system of pulverizing and drying simultaneously is generally known as flash drying. This system is used for the drying and grinding, not only of raw materials containing a moderate amount of moisture, but also precipitated products in the form of a sludge or those coming from a filter or a centrifuge. Figure 18 shows a flash-drying installation for handling a wet sludge. The raw material is fed into a mixer which receives some of the dry material being produced for the purpose of conditioning the feed to the flash dryer so it will be in the proper mechanical condition. The material from the mixer drops into the flash-drying system where it is dispersed by means of hammers or cage bars and thrown into the hot-air current. The dust-laden air passes up to a cyclone collector and is discharged in the usual manner. The vent containing the surplus air and water vapor is discharged to the atmosphere. Where this is not feasible, the vented gases pass through a standard air filter. In Fig. 18 we show the cyclone collector discharging the dried product into an air separator which is only used when it is desirable to obtain a much finer product than is produced by the flash-drying system. In this case the tailings or oversize from the air separator is discharged to the mixer and mixed with the raw feed. Where the air separator is not used, a dry divider is placed on the bottom of the cyclone collector so that any amount of finished product may be bypassed to the raw mixer. In Table 36 is given the cost of pulverizing and drying precipitated calcium carbonate.

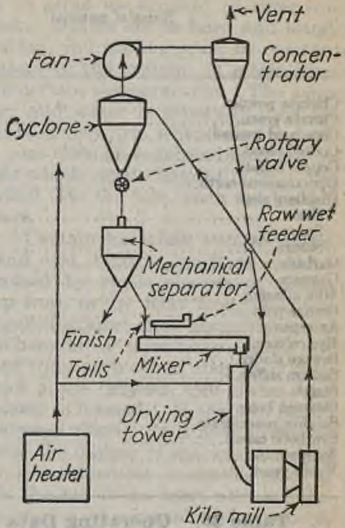


FIG. 18.—Flash-drying, pulverizing, and air-separating system.

Table 36. Cost of Grinding and Drying Precipitated Calcium Carbonate

Material	Calcium carbonate
Initial moisture	55 %
Final moisture of the dried product	½ %
Tons of finish per 24 hr	20
Heat used, B.t.u	4,000,000
Inlet temperature	600°F.
Kilowatt hour	43

The cost of operation is at the rate of $1.86 per ton of finished product. This is based on a power rate of 1 ct. per kw.-hr., on natural gas containing 1000 B.t.u. per cu. ft. at 25 cts. per thousand and labor at 60 cts. per hr.

Table 37 gives a list of products dried and pulverized on the flash-drying system. Table 38 gives data obtaining in grinding and drying bauxite.

Dehydrating. The term "dehydration" is used to denote the removal of combined water as well as free moisture. Often the term "calcination" is

Table 37. Partial List of Dried Products

Name of material	Dewatering equipment	Dryer moisture, % net basis	
		In	Out
Chlorine powder	Filter		
Chrome green	Leaf filter	25	0.1
Clay (acid treated)	Filter	60	8
Cooked blood	Press	80	5
Copper sulfate	Centrifugal	..	Monohydrate
Diatomaceous earth		45–60	2–6
Distillery slop:			
Grain	Screen		
Syrup	Evaporator	57 avg.	7
Sludge	Centrifugal		
Garbage		85	7
Gypsum		20–22	4–5
Milk albumin		50–60	5–6
Orange pulp	Press	82	4–10
An organic	Centrifugal	56–58	14
Rye culture		35	8
Sewage sludge	Filter	82	4–10
Sodium sulfite	Centrifugal	50–52	0.01
Starch	Filter	42	7
Steamed bone	Press	40	5
Sulphur concentrate	Filter	25	1.5
Synthetic casein	Filter	80	7
Tannins	Filter	60	3
Waste yeast	Leaf filter	60	5

Table 38. Operating Data on Grinding and Drying Bauxite

Initial moisture, %	9.75
Final moisture, %	0.75
Feed, lb. per hr	12,560
Product, lb. per hr	11,420
Moisture evaporated, lb	1,140
Temperature of gases entering mill, °F	700
Temperature of gases leaving mill, °F	170
Temperature of feed, °F	70
Temperature of material leaving mill, °F	150
Oil consumed, gal	15.3
Heating value of oil, B.t.u. per gal	142,000
Thermal efficiency, %	68.5
Total power for drying and pulverizing, h.p	105
Power for drying, h.p	10
Final product, % through 100 mesh	90

Table 39. Operating Data of a Kiln Mill Dehydrating and Grinding Copper Sulfate

Type of mill	No. 40 Imp
Feed, lb. $CuSO_4.5H_2O$ per hr	1500
Product, lb. $CuSO_4.H_2O$ per hr	1000
Temperature of gases entering mill, °F	700
Temperature of surplus air and moisture to filter, °F	280
Water removed, lb. per hr	500
Product, % through 300 mesh	95
Oil consumption, gal. per hr	11
Power required by pulverizer, h.p	35
Power required by dehydrating system, h.p	20

applied to this process, although this term really involves the use of higher temperatures than used in dehydration. Table 39 gives the operating data obtained in a kiln dehydrating copper sulfate ($CuSO_4.5H_2O$).

Cooling. Many materials are difficult to grind on account of their low softening point, which causes them to stick. Others are so hard and tough that excessive heat is evolved during grinding, and cooling must be resorted to in order to maintain constant temperature in the system. Certain dyes and pigments change color if heated above certain temperatures. The same system is used for cooling as for heating; cold water is generally used for cooling the air, except where the temperature is very low, when brine is used. In handling hygroscopic materials the air goes through a dehumidifier before it is passed through the pulverizer. Gases which react chemically with the pulverized material are sometimes introduced into the kiln, such as chlorine gas in the manufacture of bleaching powders.

Grinding of Combustible Materials. Certain materials, such as sulfur, starch, wood flour, cereal dust, dextrin, and coal, tend to ignite or explode when pulverized. Explosions may be caused by static electricity, sparks from furnaces or sparks caused by tramp iron, or by heat from chemical reactions taking place in the charge. To guard against such occurrences the pulverizer should be well grounded. Wherever a possibility exists for two parts of the system to become insulated from each other, they should be connected electrically. Sometimes a narrow space between two plates may become packed with material, and a condenser is formed. If the two plates become charged, a spark may discharge between them. It is now advisable to use bronze hammers and liners to decrease the danger of sparks caused by tramp iron. The best means for handling combustible materials and to reduce explosion hazards to a minimum is the use of an inert atmosphere, such as carbon dioxide. The safe concentration of oxygen varies widely. Brown and Clement [*Ind. Eng. Chem.*, **9**, 347 (1917)] give the oxygen limiting concentration as 12 per cent for dextrin, wheat starch, and flour; for oat, wheat, and corn-elevator dust, 14 to 14.5 per cent; and for standard Pittsburgh coal dust, 16 per cent. The critical oxygen content for sulfur mixture has been found by laboratory tests at about 9 per cent and for hard rubber at about 13 per cent. Under actual operating conditions it is easier to designate the safe percentage of carbon dioxide, which may be registered easily by recording instruments. In general, if the carbon dioxide concentration does not go below 8 per cent, almost all combustible materials may be ground safely.

Use of inert gas is particularly adapted to pulverizers equipped with air separation; flue gas is used for this purpose and enough is added to give a predetermined content of carbon dioxide, or of oxygen. The system, shown in Fig. 17 is generally used for grinding sulfur. Instead of hot air, inert gases are supplied. Table 40 gives the cost of installing and of operating a unit for grinding sulfur in inert atmosphere.

Cost includes the actual grinding of the sulfur from the time it enters the mill until it is delivered into the bin above the bagging system but does not include the cost of bags or the expense of bagging.

The installation used for sulfur can with certain modifications be used for starch, dextrin, and other combustible material. When beater mills or hammer mills are employed, it is advisable to make the beaters, hammers, and liners of bronze to minimize the possibility of generating sparks.

Hard rubber is one of the few combustible materials which is generally ground on heavy steam-heated rolls. The raw material passes to a series of

Table 40. Operating Characteristics of a No. 5057 Roller Mill with Double-whizzer Air Separation

Capacity and power			Cost of grinding per ton					
Fineness	Tons per hr.	Kw.-hr. per ton	Fixed charges per ton	Maintenance per ton	Power at 2 cts. kw.-hr.	Labor 60 cts. per hr.	Inert gas	Total cost
60% 200 mesh	13.50	6.6	$0.030	$0.005	$0.135	$0.010	$0.06	$0.26
70% 200 mesh	11.50	7.7	.035	.005	.155	.010	.07	.30
80% 200 mesh	9.00	9.8	.040	.005	.196	.015	.08	.36
90% 200 mesh	6.50	13.0	.055	.010	.260	.020	.11	.48
90% 325 mesh	4.50	17.0	.080	.010	.340	.030	.16	.65
95% 325 mesh	3.75	19.2	.095	.015	.385	.035	.19	.75
97% 325 mesh	3.50	20.6	.10	.020	.415	.040	.21	.82
99% 325 mesh	3.00	24.0	.115	.025	.480	.050	.24	.93
99.5% 325 mesh	2.25	32.5	.16	.030	.650	.060	.30	1.25
99.7% 325 mesh	1.75	39.0	.20	.035	.780	.070	.34	1.45
99.9% 325 mesh	1.50	45.5	.23	.040	.920	.080	.39	1.70
99.95% 325 mesh	1.25	55.0	.28	.050	1.100	.100	.45	2.05
99.99% 325 mesh	1.00	67.0	.35	.060	1.340	.160	.58	2.57
99% under 20μ	0.65	100.0	.53	.090	2.000	.220	.75	2.70
99% under 10μ	.37	170.0	.95	.130	2.400	.350	1.00	5.00
99% under 5μ	.20	300.0	1.60	.20	6.000	.600	1.40	9.80

The micron sizes were determined by microscopic count.

The fixed charges include depreciation at about 7% per year and interest on the investment at 6% less the interest earned on the depreciation reserve.

The inert gas is based on using coke as a fuel at $18 per ton.

The total cost covers the itemized costs given above and in addition includes lubrication and various other miscellaneous costs.

Cost of a Sulfur Grinding Installation

Price of mill f.o.b. Chicago, Ill.	$11,000
Price of preliminary crusher	400
Price of elevator	375
Price of raw bin, 10-ton capacity	825
Price of finished product bin, 5-ton capacity	525
Price of equipment to produce inert gas	1,500
Price of motors	2,000
Freight to Eastern port	675
Foundations and erection	700
Total cost	$18,000

The price of the mill includes the following:

1 No. 5057 high-side roller mill
1 Pneumatic feed control
1 Main exhaust fan
1 Double-whizzer separator
1 Variable drive including V-belts for driving whizzer
1 Cyclone dust collector
1 Rotary valve at bottom of cyclone
1 Fan for introducing inert gas to mill
Sufficient piping to connect the above parts so that the bottom of the cyclone may discharge the finished product 35 ft. above the base of the mill

The above price does not include motors.

Motors Required for Mill

1 60-h.p., 1200 r.p.m. motor for mill
1 40-h.p., 1200 r.p.m. motor for fan
1 10-h.p., 1200 r.p.m. motor for whizzer separator
1 1-h.p., 1200 r.p.m. motor for feeder
1 1-h.p., geared-in-head 120 r.p.m. output for rotary valve on cyclone
1 2-h.p., 1800 r.p.m. motor for CO_2 fan.

rolls in closed circuit with screens and air separators. Farrel-Birmingham rolls are used extensively for this work. There is a differential in the roll diameters, and the particular size best suited for the average hard rubber is one having rolls with 13-in. and 17-in. diameter and 20-in. face. The motor should be separated from the grinder by a fire wall. It is also desirable to run these machines at rather low speed and low differential between the rolls because it is very easy to overheat hard rubber in grinding, making it smoulder, which necessitates the shutting down of the grinder until it cools off before clearing out the charged material. The performance of a series of rolls grinding hard rubber, producing a finished product, through an air separator is given in Table 41.

Table 41. Operating Characteristics of Rolls Grinding Hard Rubber

Number of roller mills	3
Size of each mill, in.	13 and 17 × 20
Motor on each mill, h.p.	50
Size of vacuum air separator, ft.	4.5
Size of motor on separator fan, h.p.	20
Fineness of feed to separator, % through 100 mesh	32
Fineness of product from separator, % through 100 mesh	95
Production, lb. per hr	250

A larger production could probably be attained, but operation at the lower rate is advisable to prevent generation of an excessive amount of heat.

It is always advisable to equip systems for grinding explosive materials with relief vents to take care of an emergency should the inert gas cease to function properly. Recently, several installations have been made for grinding hard rubber in mills of the Bonnot roller type and in Fuller ball mills. The inert gas is introduced hot at a temperature between 110° and 130°F. Certain grades of rubber are more easily reduced when ground comparatively hot.

Production of Granules. See page 1996. It is often necessary to produce a granule or a granular product of definite size limits, such as a granulated monocalcium phosphate that will all pass through a 50-mesh screen and, within a few per cent, remain on a 200-mesh screen. This is not a simple problem of crushing and grinding but involves a complete process including the selection of a proper type of crusher or pulverizer, the best method of feeding, optimum number of stages, and the most suitable method of screening, bolting, or air separation. The type of equipment to be used depends upon the nature of the material and the nature of granule to be produced. Table 42 gives

Table 42. Variations in the Fineness of Product Obtained with the Same Material in Six Different Types of Pulverizers

Percentage on	A	B	C	D	E	F
20 mesh	0	0	0	0	0	0
50 mesh	35	30	10	3	0	0.5
80 mesh	20	18	12	5	1	1
100 mesh	12	10	14	7	2	3.5
200 mesh	10	9	13	15	12	10
250 mesh	9	9	12	18	5	4
300 mesh	7	6	11	15	10	5
325 mesh	4	6	10	10	11	12
Through 325 mesh	3	12	18	27	59	64

the analysis of products obtained by pulverizing the same material in six different types of pulverizers.

None of these mills gave a product satisfying the requirements of the granular monocalcium phosphate referred to (−50, +200 mesh). No pulverizer on the market will produce such a material in one operation. For this purpose it is necessary to use a mill that will produce the largest percentage of particles between 50 and 200 mesh, with a minimum −200 mesh. The pulverized material may then be passed over a screen to remove particles above 50 mesh, the undersize being bolted or air separated to remove all below 200 mesh. Oversize is usually returned to the pulverizer. In Table 43 some of the data have been rearranged to show which mill will give the best results in producing the granular product required.

Table 43. Selection of Best Pulverizer for Producing a Granular Product, −50, +200 Mesh

Character of product	A	B	C	D	E	F
% remaining on 50 mesh	35	30	10	3	0	0.5
% between 50 and 200 mesh	42	37	39	27	15	14.5
% through 200 mesh	23	33	51	70	85	85

Soft materials, such as clays, fuller's earth, diatomaceous earth, sodium bicarbonate, and monocalcium phosphate, are generally ground in regular flour mills to obtain a finely divided product. A ratio of reduction of 2.5:1 or even 1.5:1 is generally used; the material passes through a series of rolls with bolting reels. Sometimes air separators are used in the circuit for removal of fines. Table 44 gives the results obtained in granulating soft materials to various particle sizes.

Table 44. Granulation of Soft Materials to Various Sizes

	Diatomaceous earth	Fuller's earth	Sodium carbonate
Size of product required, mesh	−60 +100	−80 +150	−120 +200
Type of mill used	two roller	two roller	two roller
Number of mills in series	6	12	5
Size of rolls, in.	7 × 16	7 × 16	7 × 20
Capacity, lb. per hr.	1800	3500	2500
Recovery from original feed, %	75	80	55
Horse power required, total	35	80	30

Some clays, fuller's earth, and diatomaceous earth are often calcined to obtain certain desirable physical properties. Such heat treatment hardens the material and makes possible granulation with a smaller percentage of fines.

Cleaning and Concentrating. Many materials may be cleaned by removing impurities in a pulverizer equipped with air-separation apparatus or followed by air-separation equipment. Sand may be removed from clay, and refractory impurities separated from hydrated lime. Phosphate rock may be freed from such impurities as clays and silt.

The various oxides of lead, copper, and other metals often contain a certain amount of unoxidized material. When such materials are fed to an automatic pulverizer with air-separation equipment, the relatively soft oxide is reduced to fine powder and separated from the metal. Results obtained with litharge and copper oxide are given in Table 45.

Table 45. Removal of Metal from Litharge and Copper Oxide by Pulverizing and Air Separation

	Litharge	Copper oxide
Type and size of automatic pulverizer	No. 1	No. 00
Feed, lb. per hr.	3000	800
Metal in feed, %	0.5	8.5
Production, lb. per hr	2980	675
Tailings produced, lb. per hr	20	100
Metal in product, %	0.025	0.3
Metal in tailings, %	70	66
Oxide in tailings, %	30	34
Power required, h.p.	70	30
Product, % through 200 mesh	99.8	99

To separate lead from dross, skimmings, and scrap battery plates, the material is first coarsely crushed in a high-speed hammer mill discharging to a screen. Oversize, almost all metallic lead, goes to the melting pot; undersize, a mixture of metal and oxide, goes to the pulverizer.

In the cleaning and concentrating of clays, chalks, and marls, water flotation is frequently used. The product is then dried and disintegrated in hammer or cage mills. If further cleaning is desired, the washed product may be processed in a series of air separators, the tailings from one separator being fed into the next. Tailings from the last separator are either discarded or fed to the water flotation system. Some clays give a sufficiently fine product by dry pulverization and air separation.

Lime, after hydration, usually contains all the impurities of the original lime, such as sand, gravel, core, and clinker. The mixture from the hydrator is processed in an automatic pulverizer with a throwout, or in a series of

Table 46. Cleaning and Concentrating Hydrated Lime

Size of separator, diam., in.	12
Size of automatic pulverizer	No. 1
Power for separator, h.p.	25
Power for pulverizer, h.p.	7.5
Power for pulverizer fan, h.p.	40
Feed to separating system, tons per hr.	8
Production of hydrated lime, tons per hr.	7.2
Tailings, tons per hr.	0.8

Chemical Analysis

	Feed	Finished product	Throwout
Calcium and magnesium hydrate, %	88.95	98.80	5.12
Silica, %	3.16	0.15	28.61
Iron and aluminum oxides, %	2.26	0.12	20.42
Calcium and magnesium carbonates, %	2.63	0.26	23.15
Calcium and magnesium oxides, %	3.00	0.67	22.70
Fineness, % through 200 mesh	90	99.5	4.5

separators, the tailings from the last separator being used for agricultural purposes. The results obtained in processing hydrated lime are given in Table 46.

Certain phosphate sands are cleaned and concentrated to remove sand and silt. Table 47 gives the results obtained in processing dry phosphatic sand in an automatic pulverizer with throwout.

Table 47. Cleaning and Concentrating Phosphate Sand

Size of automatic pulverizer	No. 3
Feed, tons per hr	8.5
Capacity (of throwout), tons per hr	5.25
Fines, tons per hr	3.25
Power required by pulverizer, h.p.	15
Power required by fan, h.p.	40

Chemical Analysis

	Feed	Throwout product	Fines
$Ca_3(PO_4)_2$ content, %	52	68	26
$Fe_2O_3 + Al_2O_3$ content, %	12.5	6	23

The throwout product contains most of the valuable material and is the product required.

INDUSTRIAL APPLICATIONS OF CRUSHERS AND PULVERIZERS

Milling of Cereals and Other Vegetable Products into Flour and Granules. The roller mill is that most widely used for grinding wheat and rye into high-grade flour. A typical mill used for this purpose is fitted with two pairs of rolls, capable of making two separate reductions. After each reduction the product is taken to a bolting machine to separate the fine flour, the coarse product being returned for further reduction. The rolls run toward each other at different speeds in order to produce a rubbing action. Grooved rolls are used for crushing the grain, cleaning up the bran, and for grinding corn. For grinding to a finished product, smooth rolls are employed. Feed is supplied at the top where a vibratory shaker spreads it out in a thin stream across the full width of the rolls. For best results the feed should be regular, continuous, and even from one end of the roll to the other.

Rolls are made with various types of corrugation, special corrugations being used where certain results are desired. Two standard types are most generally used; the dull and the sharp, the former mainly on wheat and rye, and the latter for corn and feed. Under ordinary conditions, a sharp roll is used against a sharp roll for very tough wheat; a sharp fast roll against a dull slow roll for moderately tough wheat; a dull fast roll against a sharp slow roll for slightly brittle wheat; and a dull roll against a dull roll for very brittle wheat. Table 48 shows the operating characteristics for several sizes of roller mills. Speeds given refer to the pulley on the fast side of the mill; the speed on the slow side depends upon the differential in speed required, which usually is 2½:1 for corrugated rolls and 1¼:1 for smooth rolls.

Milling of wheat is not only a question of grinding and sifting, but it involves proper preparation of the wheat prior to grinding. As the grain arrives in the mill, it contains sticks, straw, string, sand, and other materials which must be removed. It is first passed through a receiving separator with three superimposed screens where cleaning is aided by suction. A cleaner separation is obtained with the milling separator which is used in larger mills. When the grain is very dirty, a washer and drier is used. A wheat steamer is often used advantageously. Uniform heating mellows the wheat berry and so conditions it that moisture can be added in whatever quantities desired in the milling. The grinding rolls become effective, and further granulation is secured; bolting is also made easier. A product can thus be maintained at a uniform standard, irrespective of season or the condition of the original wheat. A mill, having a capacity of 16 to 20 bbl. of flour (196 lb. each), equipped with a 6- by 14-in. double roller mill (corrugated), a 6- by 16-in. double roller mill (smooth), with accessories, requires 10 to 12 h.p.

Table 48. Operating Characteristics for Roller Mills

Size of rolls, in.	Speed of rolls, r.p.m.	Horse power
6 × 12	700	2
6 × 14	700	2.5
6 × 16	700	3
6 × 20	700	4
7 × 16	600	3.5
7 × 20	600	4
7 × 24	600	5
9 × 18	500	5
9 × 24	500	6
9 × 36	500	7
10 × 30	450	7
10 × 36	450	8
10 × 42	450	9

The cereal milling industry produces a great variety of products besides flour, such as meal, corn flour, graham flour, barley, oats, rye, buckwheat, hominy grits, and whole-wheat flour. In addition to roller mills, buhrstone and attrition mills are used extensively, particularly when the whole grain is to be ground. Besides the double-roller mill used for flour, single-roller mills are used, principally for cracking corn and rolling oats. Two-pair high and three-pair high roller mills are used to a great extent in the feed industries, the former principally for coarse feed, such as screenings; the latter for all cereal grains for table use and also for various grains for feed. This mill can be used in combination, the top pair for cracking corn, the middle pair for finishing into coarse feed or corn meal, and the bottom pair exclusively for rolling oats. The various feed stuffs are also ground on swing hammer mills. Operating characteristics for single- and double-runner attrition mills are given in Tables 49 and 50.

Table 49. Operating Characteristics of a Double-runner Attrition Mill with Two Rotors Running to Opposite Directions

Size of mill, in.	Speed, r.p.m.	Capacity feed, lb. per hr.	Horse power
16	2400	1,800	10- 15
20	2000	3,000	15- 20
24	1800	5,500	25- 40
30	1500	6,600	50- 75
36	1300	10,000	60-100

Soy Beans, Soy-bean Cake, and Other Pressed Cakes. Soy beans are ground in about the same manner as the various grains, depending on the nature of the product desired. Roller mills, dryers, and bolting reels may be used. After granulation on rolls the granules are generally treated in presses to remove the oil. The product from the presses goes to attrition mills or flour rolls and then to bolters, depending upon whether the finished product is to be a feed meal or a flour. If the whole cake is to be pulverized without removal of fibrous particles, it may be ground in a hammer mill, with or without air separation. A 20-h.p. hammer mill with air separator, grinding pressed cake, had a capacity of 300 lb. per hr., 90 per cent through 200 mesh; a 20-h.p. screen-hammer mill grinding to 1/16-in. screen produced 1000 lb. per hr.

Table 50. Operating Characteristics of a Single-runner Robinson Attrition Mill, Grinding Grain

	Size of mill							
	16 in.	18 in.	20 in.	24 in.	26 in.	30 in.	32 in.	36 in.
Speed, r.p.m.[1]	2500	2250	2200	1800	1600	1400	1300	1200
Speed, r.p.m.[2]	1000	950	900	800	750			
Capacity[3]	1200	1600	2000	3300	4000	5000	5300	6300
Capacity[4]	1200	1300	1500	1900	1900	2200		
Capacity[5]	65	80	100	150	200			
H.p.[6]	9–12	10–15	12–18	20–30	22–32	25–35	28–38	30–50
H.p.[7]	5– 8	6– 9	8–10	9–12	10–15	12–18		

[1] R.p.m. when grinding feed or corn meal.
[2] R.p.m. when cracking corn.
[3] Grinding feed, pound per hour.
[4] Grinding corn meal, pound per hour.
[5] Cracking corn, bushels per hour.
[6] Power when grinding feed or corn meal.
[7] Power for cracking corn.

Table 51. Operating Results with Hammer Mills, Pulverizing Various Seed Cakes

Material	Pulverizer	Capacity, tons per hr.			Horse power
		Pea meal	Pea and finer	Extra fine	
Cottonseed cake	Williams	1	¾–1	½	8–12
Expeller cake	Williams	2½–3	2½–3	2	25–30
Linseed cake	Williams	6 –8	5 –6	4–5	50–60

The method used for grinding pressed cakes depends upon the nature of the cake, its purity, residual oil, and moisture content. Many of these materials are treated in hammer mills, especially where no fine reduction is required. In many cases the hammer mill is used merely as a preliminary pulverizer, followed by an attrition mill. A finer product may be obtained in a hammer mill in closed circuit with an extraneous screen or air separator. Table 51 gives the results obtained with hammer mills grinding linseed cake, cottonseed cake, castor cake, corn cake, and other expeller cakes.

Cocoa Powder. Pulverization of cocoa powder involves processes for heating and cooling the powder in its travel through the equipment. Besides producing a powder of uniform fineness the proper color must be obtained, generally a dark tan or brown, approaching a chocolate color. This color is obtained by heating the cocoa moderately during the grinding, whereby it darkens to the desired color; if the powder is then chilled suddenly, the dark color remains permanently. The cake from the presses usually contains 20 to 25 per cent butterfat, although the present tendency is to lower this amount to 15 per cent or even less for the cheaper grades. The material from the pulverizer is chilled in a screw conveyor and taken to a bolter or air separator. In recent years, bolters have been largely replaced by air separators, and pulverizers of the hammer-mill type with internal screen separation.

Starch and Other Flours. Grinding of starch is not particularly difficult, but precautions must be taken against explosions; starches must not come in contact with hot surfaces, sparks, or flame when suspended in air. Where a product of medium fineness is required, the average hammer or beater mill of the screen type is employed. For finer products the air-separating pulverizer of the beater type, or a screen-hammer mill in closed circuit with an extraneous air separator, is used. Potato flour, tapioca, banana, and similar flours are handled in this manner. Table 52 gives the capacity of different types of pulverizers.

Table 52. Operating Results with Various Pulverizers, Grinding Flour

	Corn starch	Potato starch	Tapioca flour
	Screen-mill pulverizer	Screen-mill pulverizer	Beater-mill (with air separation)
Capacity, lb. per hr.	3000	2000	2000
H.p.	15	7.5	40
% through 200 mesh	75	92	98
	Beater-mill (with air separator) pulverizer	Beater-mill (with air separator) pulverizer	Beater-mill (with air separator) pulverizer
Capacity, lb. per hr.	4000	1000	1500
% through 200 mesh	95	98	90
H.p.	45	7.5	25
	Screen-mill (extraneous air separator) pulverizer		
Capacity, lb. per hr.	3500		
% through 200 mesh	90		
Horse power required	30		

Hay and Other Herbage. Loose hay, velvet beans, snap corn, alfalfa, corn fodder, bean straw, and other herbage are ground in hammer mills equipped with perforated screens. The Williams hay cutter is equipped with special feed, and a belt conveyor carries it under a draper which compresses it and forces it into the cutting mechanism. The material is cut between high-speed chisel-shaped hammers and a stationary cutting knife. The fineness is regulated by the size of the openings in the screen or cage. A mill equipped with a 0.5-in. cage when grinding hay at the rate of 6 to 8 tons per hr. requires 125 to 150 h.p. When coarse alfalfa is further reduced to a fine meal, a hammer mill producing 2 to 2.5 tons per hr. requires 100 to 125 h.p.

Dried Fruits and Vegetables. Pulverization of dried fruits and vegetables is greatly affected by the residual moisture and by the drying process used. Certain drying processes embrittle the material, giving a product that is easily ground. Swing hammer mills, with slotted or perforated screens or cage bars, are most generally used. A recent development is the use of an air-separation pulverizer for dehydrating and pulverizing wet waste fruit pulp. Hot air is introduced into the machine, and the moisture is removed while the material is being pulverized and conveyed to the bagging machine.

Metalliferous Ores. Ores are ground both wet and dry in open and closed circuit, although in later years the tendency has been toward wet grinding in closed circuit with a classifier. Reduction varies from a 16- to 35-mesh product for table concentration up to 100 to 200 mesh for cyanidation. The mills used in the majority of cases are ball mills, rod and tube mills, and stamp mills.

Non-metallic Minerals. Dry grinding is most generally used; wet grinding is resorted to where certain impurities have to be removed, such as iron oxide or fine grit, and where washing imparts certain desirable properties to the finished product. The system of wet grinding and classification chosen depends upon the nature of the material. Very hard materials are ground in ball mills with hydraulic classifiers, the finished product going to a filter press and dryer. After drying, the cake generally has to be broken up in some type of disintegrator.

It is often possible, particularly with a soft material like clay, to employ dry grinding with a series of air separators, at the same time eliminating impurities such as grit and sand. Nearly every type of pulverizer is used in the grinding of non-metallics in open or closed circuit. The objective is often to obtain many grades of the same material by tying air separators and screens into the grinding system to remove various-sized products. Choice of equipment generally depends on (1) hardness and (2) contaminations. Most of the refractory silicates are ground in ball mills and tube mills in continuous or batch operation. Silex or flint lining and balls of flint, porcelain, and similar materials are used where iron must be kept out. Capacity of any system decreases rapidly with increasing fineness of the material; this applies particularly to non-metallics, where extreme fineness is usually required.

Silica and *feldspar* are ground in silex-lined mills with flint balls. Feldspar for ceramic and chemical industries is ground finer than for the glass industry. The following description of a feldspar mill is abstracted from *U.S. Bur. Mines, Information Circ.* 6488. The flow sheet shown in Fig. 19 may be modified to the treatment of many other non-metallics. The fine-grinding department consists of three silex-lined Hardinge mill units with flint pebbles, 27 to 28 r.p.m., each fed from a 50-ton surge bin by James automatic belt feeders. Unit 1 has an 8- by 4-ft. Hardinge mill, discharging by an elevator to a Gayco air separator or a James vibrating screen, according to what kind of product is desired, a fine (120 to 250 mesh) or a coarse (20 mesh). An intermediate product (40–100 mesh) is made by removing fines in the Gayco and screening the coarse. A tube mill is used in connection with unit 1 for

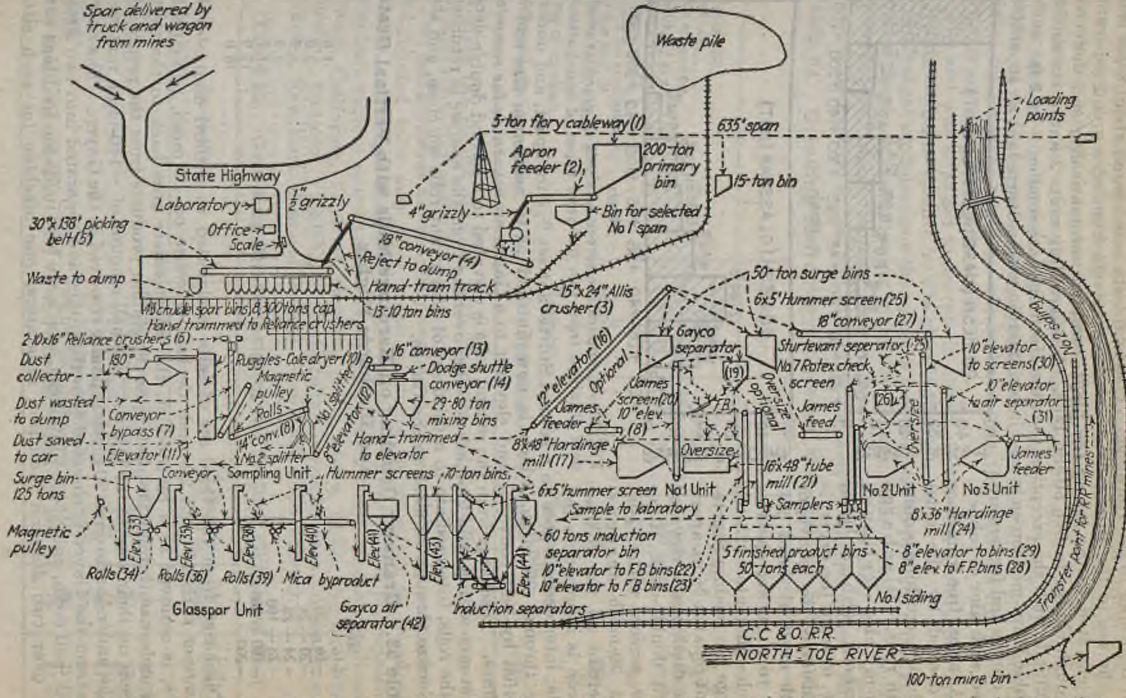


FIG. 19.—Flow sheet of a typical non-metallic plant for crushing, granulating, pulverizing, screening, and air separating.

fine grinding; it receives about half the oversize from the Gayco, the other half returning to the Hardinge. Plant capacity is increased about 8 per cent by the use of the tube mill. Finished products from screen and separator are elevated to the top of the plant, sampled automatically, and delivered through chutes to five 50-ton bins. Fine-grinding units 2 and 3 are similar to unit 1, with the exception that the Hardinges are smaller, 8 by 3 ft., and a double Hum-mer screen is used in conjunction with a Sturtevant air separator. All chutes and feeders are lined with silex or rubber.

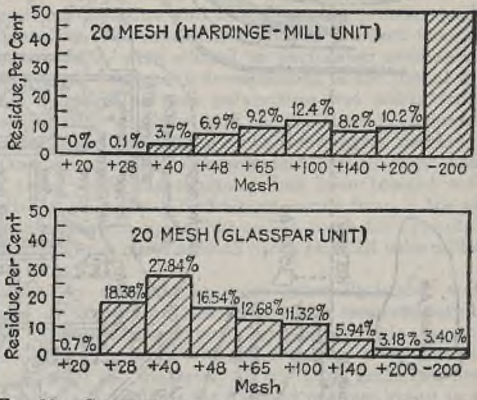


FIG. 20.—Comparative screen analysis of granular products obtained from two different types of installations.

As a spar free from fines is desired by the glass industry, crushing is done with rolls. Discharge from the Reliance crushers, about ½-in. size, or from the dryer, is transferred by conveyor with magnetic-head pulley to a bucket elevator discharging into a 125-ton surge bin feeding Sturtevant rolls set ⅜ in. apart. The discharge is elevated to a double-deck Hummer screen with ¼-in. and 20-mesh screens. Oversize on the coarser screen is returned to the rolls; intermediate goes to a second Sturtevant set ³⁄₁₆ in. apart, and undersize through 20 mesh, which is now finished as to maximum size and is carried by belt conveyor and elevator to a Gayco air separator for removal of fines. Discharge from the second rolls is elevated to a second double-deck Hum-mer with 8- and 20-mesh screens; oversize from the coarser is returned to the rolls, intermediate to a third set of Sturtevant rolls set close, and the undersize through 20 mesh to the Gayco.

Table 53. Granular Grinding on Hardinge Mills with Vertical Grate and Discharge Flights

Mesh	
+16	1.0
−16 + 20	5.0
−20 + 24	8.9
−24 + 28	9.5
−28 + 100	51.7
−100 + 200	13.9
−200	10.0

Discharge from the third set of rolls is elevated to a third double-deck Hum-mer with 10- and 20-mesh decks. Oversize on 10 mesh, principally mica, is removed as a by-product, the intermediate goes back to the rolls, and undersize through 20-mesh joins the similar products from the preceding screens on their way to the Gayco. A small amount of −140-mesh material in the undersize through 20 mesh is finally removed in the Gayco. The oversize from the Gayco is elevated to a Hum-mer and separated into two sizes, −20 +40 mesh, and −40 +140 mesh, which are held in 70-ton feed bins and run over two Johnson induction separators, one machine for each size.

Figure 20 gives the comparative screen analysis of the two units as installed. The Hardinge mill, just discussed, had not been designed for granulating. It was a very fine grinding mill arranged only for trunnion overflow. It was therefore not possible to pass the spar through the mill fast enough to prevent overgrinding. As a result the Hardinge product shows a much higher percentage of the undesirable −200-mesh material. Table 53 gives the results of a Hardinge mill equipped with a vertical grate and discharge flights for granular grinding, which shows that the products from the two systems are more nearly the same.

The mill was arranged to operate in closed circuit with a vibrating screen grinding nepheline syenite for the glass industry.

With slight modifications the systems may be used to produce a fine grade or granules from the following materials: quartz, slate, marble, corundum, carborundum, tripoli, pumice, and volcanic ash. Practically all abrasive silicates are handled in ball and tube mills followed by air separators. Table 54 gives the results obtained with Hardinge pebble mills, grinding several siliceous refractory materials.

Table 54. Grinding Refractory Siliceous Materials in Pebble Mills

	Feldspar	Silica sand	Enamel frit	Grog
Size of mill	8′ × 60″	8′ × 48″	4′ × 16″	5′ × 22″
Feed size	2″	20 mesh	⅛″	1½″
Size of product	99% through 200 mesh	98% through 325 mesh	97% through 100 mesh	95% through 10 mesh
Capacity, tons per hr	1.75	1.25	0.225	5
Power for mill, h.p.	68	58	8.5	28
Power for auxiliaries, h.p.	21	20		
Pebble load, lb.	10,000	12,000	2000	2800
Speed of mill, r.p.m.	22	18	30	30
Moisture, %	1	1	0	1
Type of classifier	Hardinge	Air	Trommel screen on mill	
Lining and grinding mediums	Flint blocks and flint pebbles			Steel

Talc and soapstone are generally easily pulverized, although certain fibrous and foliated talcs may offer great resistance to reduction to impalpable powder. Arranged according to the resistance offered to fine pulverization, talcs from various sources may be listed as follows: Italy, Manchuria, Russia, France, California, Vermont, Georgia, South Africa, India, Quebec, North Carolina, Ontario, Virginia, North Carolina (pyrophyllite), Massachusetts (foliate), New York (Natural Bridge), New York (Gouverneur).

Table 55. Grinding Characteristics for Various Grades of Talc

Source of talc	Fineness	Capacity, lb. per hr.	Horse power	Cost of grinding per ton
Italy	99.5% through 200 mesh	6000	85	$0.98
Manchuria	99.0% through 200 mesh	6200	85	$0.99
Russia	99.5% through 325 mesh	3500	75	$1.35
California	99.5% through 200 mesh	5500	80	$1.09
Vermont	99.5% through 200 mesh	5000	80	$1.16
Virginia	99.0% through 200 mesh	4200	80	$1.28
North Carolina (pyrophyllite)	99.0% through 300 mesh	3000	75	$1.68

Table 55 gives data obtained in crushing various talcs to impalpable powder in a five-roller high-side Raymond mill, the feed measuring 1 in. or under.

Costs given do not include primary crushing or bagging. Power is figured at 3 cts. per kw.-hr. and labor at 50 cts. per hr. Depreciation, interest, maintenance, and lubrication are included.

U.S. Bureau of Mines, Bull. 213 gives an excellent description of mining and processing talc and soapstone, which may be applied to many of the other non-metallics. Tube mills, 150 h.p. each, lined with silex or porcelain, are set in two lines of four each in tandem. Each line has a capacity of 2 tons per hr. Closed-circuit grinding with air separators is frequently employed. Batch grinding is also used. Batch pebble mills are operated for a certain length of time, up to 8 hr. for the finer grades, before dumping. In Talcville, N.Y., 6- by 8-ft. pebble mills are used, 30 to 35 h.p., charged with 1 ton talc and 3 tons flint pebbles and rotated for 4 to 7 hr. at 22½ to 23 r.p.m.

Clays and Kaolins. A large percentage of clays and kaolins are washed and water-floated, after which they are filter-pressed, dried, and disintegrated in hammer mills or disintegrators, followed by a series of air separators. This is of special advantage when the clay contains impurities such as fine mica flakes which are not removed by water flotation. Wet and dry pans and swing hammer mills are used in most brick, tile, or pottery plants where comparatively coarse pulverization is required.

Clays used in the chemical industries as a constituent of paint and as a filler for paper and rubber are both wet-ground and dry-ground. When it is necessary to treat the clay chemically, it is generally wet-ground and treated as above. The tendency at present is to dry-grind most clays which do not require chemical treatment. The roller mill with air separation is used nearly exclusively, particularly on the southern clays. The raw material, after being crushed, passes through a rotary dryer and the moisture content reduced to about 8 or 10 per cent. The material from the dryer goes to a roller mill with whizzer separation. Hot gases are introduced to the mill which complete the drying while the material is being pulverized to the necessary fineness. Even on washed clays there is a tendency to partial drying of the clay in an extraneous dryer and to finishing the drying and pulverizing in an air-separation mill which may be of the roller-mill type or of the hammer-mill type. These mills are equipped with automatic throwouts, so that a high percentage of impurities such as silica sand may be eliminated from the system. On an average grade of unwashed clay a No. 5 roller mill equipped with whizzer separation will grind from 3 to 3½ tons per hr. to a fineness of about 99.95 per cent through a 325-mesh screen, while on washed clay the capacity will be from 30 to 40 per cent higher. When grinding 3½ tons an hr. of a raw clay, the power consumption will be about 100 h.p., and it takes about 750 cu. ft. of natural gas containing 1000 B.t.u. per cu. ft. to dry the clay from 10 per cent moisture down to about 1 per cent.

Mineral pigments, such as ochers, umbers, siennas, and red oxides of iron, were for many years ground on stone mills and bolted. When the materials were wet-ground and classified, they were filtered-pressed, dried, and disintegrated in a cage mill or a hammer mill with screen separation. This is still the process when the material is water-floated. When the oxides are ground dry, roller and hammer mills with air separation have replaced a great many stone mills. Even when buhrstones are used, bolting has generally been replaced by air separation. Some of the red pigments are still ground on buhrstones as the idea prevails in some quarters that the stones make a smoother and richer colored material. A 36-in. buhrstone in closed circuit with an 8-ft. air separator produced 300 lb. per hr. of ground hematite, 99.2 per cent through 325 mesh, with 15 h.p. on the stone. Table 56 gives operating characteristics of a Raymond No. 5 high-side mill grinding various oxides of iron.

Table 56. Grinding Iron Oxides in No. 5 Raymond Roller Mill

Material	Fineness	Capacity, lb. per hr.	Total horse power per ton
Raw sienna	99% through 200 mesh	5950	23.5
Burned sienna	99.5% through 200 mesh	5800	22.1
Raw umber	99% through 200 mesh	5200	26.9
Burned umber	99.5% through 300 mesh	5400	25.9
Natural ochre	99.9% through 200 mesh	4500	31.0
Iron oxide (ore)	99% through 325 mesh	3100	45.0
Iron oxide (precipitated)	99.9% through 325 mesh	1800	72.5

Power consumption includes power required for grinding, separating, and conveying product to bins above the baggers. A 4.5-ft. by 16-in. Hardinge conical mill in closed circuit with classifier, grinding 50-mesh iron oxide with 33 per cent moisture for the paint trade showed a capacity of 25 tons per 24 hr., 100 per cent through 200 mesh. Power consumption was 20 h.p., mill speed 30 r.p.m., ball load 4000 lb.

Non-metallic Carbonates and Sulfates. Non-metallic carbonates include limestone, calcite, marble, marls, chalk, dolomite, and magnesite; the most important sulfates are barite, celestite, anhydrite, and gypsum; these are used as fillers in paint, paper, and rubber. Gypsum and anhydrite are treated under the heading Cement, Lime, and Gypsum.

Table 57. Capacity of Raymond Roller Miller, Grinding Limestone

Fineness	No. 2 roller mill	No. 3 roller mill	No. 4 roller mill	No. 5 roller mill
	Lb. per hr.			
75% through 200 mesh	4200–4500	6500–7000	8500–9500	11,000–12,000
85% through 200 mesh	3700–4000	5700–6200	8000–8500	10,000–11,000
95% through 200 mesh	2600–3000	4000–4500	5500–6000	7,000– 7,500
99% through 200 mesh	2000–2300	3000–3500	4000–4700	5,000– 6,000
99% through 300 mesh	1200–1300	1900–2000	2400–2700	3,200– 3,500
99% through 325 mesh	1000–1100	1600–1800	2000–2200	2,800– 3,000
99.5% through 325 mesh	900–1000	1400–1600	1800–2000	2,500– 2,800
	Total h.p., and cost in dollars per ton			
75% through 200 mesh	55–65, $0.79	70–75, $0.69	90–95, $0.62	100–105, $0.58
85% through 200 mesh	53–55, 0.88	65–70, 0.77	88–90, 0.75	98–100, 0.70
95% through 200 mesh	50–53, 1.18	60–65, 1.04	85–88, 0.93	95– 98, 0.87
99% through 200 mesh	47–50, 1.62	55–60, 1.41	80–85, 1.25	90– 95, 1.10
99% through 300 mesh	45–47, 2.35	53–55, 2.06	75–80, 1.88	85– 90, 1.68
99% through 325 mesh	42–45	50–53	70–75	80– 85
99.5% through 325 mesh	40–42, 2.62	47–50, 2.30	65–70, 2.05	75– 80, 1.92

Table 57 indicates the results obtained with several sizes of Raymond mills grinding typical limestone; these results may be supplied to practically the entire group. When a material is very soft, such as high-grade barite, the capacities may be about 25 to 35 per cent higher.

Of the carbonates, magnesite is generally the hardest to pulverize. This material is often calcined and pulverized in the same manner as lime. Dead-burned magnesite is treated as cement clinker.

Fluorspar. A No. 3 Raymond roller mill grinding fluorspar had a capacity of 3700 lb. per hr., 95 per cent through 200 mesh, with a power consumption of 32 h.p. on the mill and 23 h.p. on the fan, a total of 22.3 h.p. per ton. Magnesite, generally the most difficult of the carbonates to pulverize, grinds similarly to fluorspar. When calcined it is ground in the same manner as quick lime. Dead-burned magnesite is handled like cement clinker. Table 58 gives the results obtained in a Hardinge mill, wet grinding barytes and limestone to be used as a paint filler.

Table 58. Wet Grinding of Barytes and Limestone in Hardinge Mill

	Limestone	Barytes
Size of mill	8′ × 48″	7′ × 36″
Size of feed	1½″	1½″
Fineness of product, mesh	350	300
Capacity, tons per hr	¾	2
Mill speed, r.p.m.	18	22
Power for mill, h.p.	40	25
Classifier system	Cone	Drag
Moisture in mill, %	30	28
Type of mill lining	Flint	Flint
Grinding mediums	Coarse limestone	Lump barytes

Asbestos and Mica. The choice of crusher for asbestos depends on whether a long or a short fiber is desired. Crushing is done in slow stages to preserve as much as possible of the fiber length. Primary crushers employed are usually of the jaw type with secondary crushers of the smaller jaw type. Small gyratories and corrugated rolls are also used. With some grades a third reduction may be required. After drying and crushing to about 2 in., the asbestos rock goes to the so-called fiberizing machines, which reduce the rock, liberate the fiber, and split it into fine and coarse fiber. There are different types of fiberizers, the swing hammer mill, the Jumbo, and the Laurie and Pharo cyclones. The *Jumbo* consists of a cylindrical shell surrounding a shaft with six pairs of arms placed at 6-in. intervals and disposed crosswise to each other. The arms are of heavy steel bars with chilled iron beaters, the faces of which are constructed on an angle. The *Laurie* cyclone consists of two beaters of the screw-propeller type, driven in opposite directions at 1700 to 2000 r.p.m., in a cast-iron chamber. The *Pharo* cyclone was designed to overcome the tearing effect on the fiber, one of the objections to the Laurie. It is of the same general type, but the hood above the discharging end is cut off immediately above the latter, and the crushing blades, or beaters, of which the paddles are one right and the other left, rotate in the same direction.

As the material reaching the mill contains a large amount of freed asbestos, separation of the fiber begins immediately. It is first put through a screening trommel, the fines are discharged on a shaking screen, and the overflow—all above 1½ in.—falls into one of the fiberizing devices which discharges on the same screen. The latter is slightly inclined and is made from wire or perforated plates. It has an oscillating movement which, apart from the sizing of the rock and eliminating the sand, causes the fiberized asbestos to rise to the top. The liberated fiber is taken up by a fan, while the overflow falls into a second fiberizing machine, which discharges, like the first, on a screen, where the asbestos is again lifted by a fan, and so on until the rock is practically entirely pulverized. Tailings free from asbestos go to the dump.

Asbestos is often pulverized. This is the case when it is used for molded products. The pulverizing is usually accomplished by passing the material through a series of buhrstones or by using a high-speed screen mill with air-transport system. A mill with a $\frac{1}{64}$-in. screen pulverized 400 lb. per hr. with 13-h.p. power consumption. Certain impurities, such as sand, gravel, and hard fiber, may be removed by using an air-separation pulverizer with automatic throwout, as described under Hammer Mills. Operating characteristics for a mill of this type are given in Table 59.

Table 59. Pulverizing Impure Asbestos in Screen Mill

Type of automatic pulverizer	No. 1
H.p. required by pulverizer	15
H.p. required by fan	20
Capacity, lb. fine asbestos per hr	1250
Tailings, lb. per hr	25

Screen Analysis

	Fines	Tailings
% on 10 mesh	Trace	15
% on 20 mesh	8	42.5
% on 48 mesh	4	17.5
% on 65 mesh	3	13
% on 80 mesh	1.5	10
% on 100 mesh	1	2.5
Through 100 mesh	82.5	0.5

The cost of pulverizing this impure asbestos fiber is about $2.52 per ton material fed. This price is based on a power cost of 2 cts. per kw.-hr., 10 per cent depreciation, 6 per cent interest, and labor at 50 cts per hr.

The micas, as a class, are difficult to grind to a fine powder; one exception is disintegrated schist in which the mica occurs in minute flakes. The material pulverized is generally the waste from production of sheets and scrap from punching and trimming. Arranged in order of increasing resistance to grinding to a fine powder, micas from various sources may be classified as follows: Madagascar, Ontario, Quebec, Manchuria, India, New Hampshire, North Carolina, South Africa, Russia, Brazil.

Mica is pulverized wet or dry; the wet-ground product is the more desirable, as it retains its luster to a high degree. When ground wet, it is first passed through revolving screens with a constant stream of water; it is then ground in wooden chaser mills at a slow rate and graded after drying, by passing through a series of bolting reels, the finest reel being about 200 mesh. A modification of this process is used in certain European countries, where the mica is ground in chaser mills and buhrstone mills. The water with the ground mica is passed over screens and thus graded. After pressing out the water, the solids are dried and disintegrated in a double-cage mill. Table 60 gives data obtained in wet-grinding a Manchurian mica.

For dry grinding, hammer mills equipped with an air transport system are generally used. The material, after dropping through a perforated screen into the intake of an exhauster, is collected in a cyclone followed by bolting

Table 60. Wet Grinding Manchurian Mica

Amount passed through mill, lb. per hr.	400
Total power consumption, including pumps, screens, mill elevators, and conveyor, kw.	60
Power consumption, kw.-hr. per ton product	300
Screen analysis, feed to bolting reels:	
% on 20 mesh	5
% on 50 mesh	18
% on 80 mesh	17
% on 100 mesh	12
% on 200 mesh	19
Through 200 mesh	29

reels. Table 61 gives the operating characteristics grinding North Carolina mica in a high-speed hammer mill with flat beater blades firmly bolted to the beater disks and surrounded by a 16-mesh screen.

Table 61. Grinding Mica in High-speed Hammer Mill

Size of mill	No. 1 Screen
Size of motor (direct-connected), h.p.	60
Size of feed	Scrap
Production, lb. per hr.	950

Screen analysis of discharge, % on:

20 mesh	40 mesh	60 mesh	80 mesh	100 mesh	150 mesh	through 150 mesh
1%	15%	22%	16%	10%	11%	25%

For coarser grades, a 10- to 12-mesh screen is used.

The Fertilizer Industry. Many of the materials used in the fertilizer industry are pulverized, such as those serving as sources for calcium, phosphorus, potassium, and nitrogen. The most commonly used for their lime content are limestone, oyster shells, marls, lime, and, to a small extent, gypsum. Limestone is generally ground in hammer mills, ring roll mills, and ball mills. Fineness required varies greatly from −10 mesh to 75 per cent through 100 mesh.

Table 62. Operating Data Grinding Oyster Shells and Burned Lime in Hammer Mills

Type of mill	Size, in.	Capacity, tons per hr.	Horse power
Jeffrey	15 × 8	0.5–0.75	8
	20 × 12	1 –1.5	12
	24 × 18	2 – 3	20
	30 × 24	4 – 5	30
	36 × 24	8 –10	40
Stedman	12 × 9	1.5	8
	20 × 12	4	20
	24 × 20	8	40
	30 × 30	12	60
	36 × 36	20	100

Oyster Shells and Lime Rock. Operating characteristics for hammer mills grinding oyster shells and burned lime for agricultural purposes are given in Table 62.

The data for the Jeffrey mill refer to the pulverization of oyster shells and the data for the Stedman mill refer to the pulverization of burned or quicklime through a $\frac{1}{8}$-in. bar opening.

Phosphates. Phosphate rock is generally pulverized for one of two major purposes: for direct application to the soil, or for acidulation with sulfuric acid in the manufacture of acid or superphosphate, phosphoric acid, and the various phosphates. Table 63 gives the data for pulverizing phosphate materials in a Raymond five-roller mill equipped with integral air separation, and in ring roll mills equipped with extraneous air-separation units.

Table 63. Capacities, Grinding Phosphate Materials

	Production, tons per hr.		H.p.		Cost, dollars per ton	
	(*a*) 95–95 % through 60 mesh; 50–55 % through 200 mesh	(*b*) 90–95 % through 200 mesh	*a*	*b*	*a*	*b*
Algiers	7.5	...	13			
Arkansas block rock	6.5	4	15	19	0.49	0.70
Belgium	8.5	...	12			
Bohemia (apatite)	6.5	...	17			
Canada (apatite)	6	3	18	26	0.61	0.85
Egypt	6.5	...	17			
Florida (pebble)	6.5	3.5	17	24	0.51	0.73
Florida (hard rock)	7	4	17	24	0.48	0.68
Florida (soft rock)	...	5.5	..	15	0.42	0.59
Idaho	7	4	14	19	0.46	0.68
Kentucky	7.5	5	12	17	0.42	0.60
Morocco	8	...	12			
Pacific and Indian oceans:						
Angaur Island	8.5	...	12			
Christmas Island	8	...	12			
Marshall Islands	8.5	...	12			
Makatea Island	7	...	13			
Nauru Island	7	...	13			
Ocean Island	7.5	...	13			
Russia (Podolian)	6.5	...	13			
Tennessee (blue rock)	7	4.5	14	18	0.46	0.65
Tennessee (brown rock)	7.5	5	12	17	0.42	0.60
Tennessee (gray rock)	8	5	12	17	0.42	0.60
Tennessee (phosphatic limestone)	7	4.5	14	18	0.46	0.65
South Carolina	6.5	3.5	17	24	0.52	0.73
Tunis	8	...	12			

The coarse material (*a*) is used for acidulation, and the fine material (*b*) for direct application to the soil. Cost data are based on power at 3 cts. per kw.-hr. and labor at 50 cts. per hr.

Some average results obtained in grinding various organic and inorganic raw materials for fertilizers are given in Table 64.

Inorganic salts seldom require fine pulverization, but they frequently become lumpy. In such a case they are passed through a double-cage mill

Table 64. Results Obtained in Grinding Raw Materials for Fertilizers

Material	Pulverizer	Type or size	Bar opening, in.	Capacity, lb. per hr.	Horse power
Acid phosphate	Stedman	36 in.	..	12,000	40
Steamed bone	Jeffrey	A-30 × 24 in.	1/8	10,000	40
Dry kelp	Williams	Shredder 2	1/4	12,000	35
Guano (Peruvian)	Jeffrey	A-24 × 18 in.	1/8	7,000	40

or a hammer mill with the screen or cage bars removed. This is done with ammonium sulfate from by-product ovens and sodium nitrate. When used as an ingredient fertilizer the latter is generally mixed with other raw materials, and the mixture is later disintegrated. The various potassium salts used in fertilizers are generally shipped ready for use, but if they have become caked in transit they are broken in an ordinary disintegrator.

Basic slag is often used as a source of phosphorus. Its grinding resistance depends largely upon the way it has been cooled, slowly cooled slag generally being more easily pulverized. The most common method for grinding basic slag is in a ball mill followed by a tube mill or a compartment mill. Both systems are in closed circuit with the air separator. A 7- by 5-ft. mill, requiring 125 h.p., operating with a 14-ft., 30-h.p. separator, gave a capacity of 5 tons per hr. from the separator, 95 per cent through 200 mesh. Mill product was 68 per cent through 200 mesh, and circulating load 100 per cent.

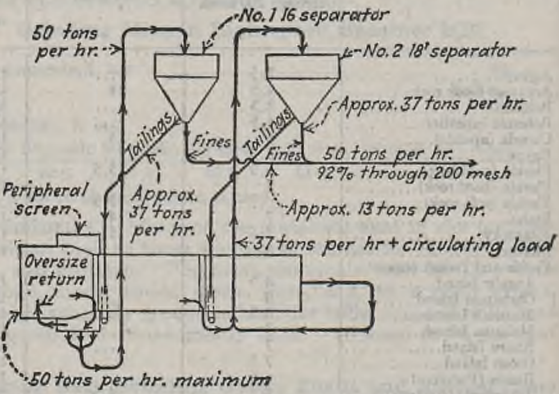


FIG. 21.—Three-compartment compeb mill (with peripheral screen on first compartment) in closed circuit with two air separators grinding and separating the raw mix in a cement plant.

The Cement, Lime, and Gypsum Industries. Portland cement is manufactured by wet and dry processes. In the latter the material is dried in rotary dryers prior to grinding. It is then burned and the clinker pulverized. In the wet process the materials are mixed and ground, a certain amount of water is added, giving a mill discharge with up to 40 per cent water. The slurry is calcined in the usual manner and the clinker ground as in the dry process.

Crushing in the dry process is done in gyratories, jaw crushers, hammer mills, or roller mills. Pulverizing the raw mix is done in mills of the ring roll type with or without tube mills, in ball mills or kominutors followed by tube mills, and in compartment mills. Air separation is standard practice; Hercules ball and tube mills are now operated in closed circuit with air separators. Production for a 250-h.p. Bradley-Hercules, operating on a 2-in. feed, was about 40 tons per hr., 51.8 per cent through 200 mesh, 60.2 per cent through 100 mesh, 69.9 through 50, 74.6 through 40, 81.4 through 30, and 90.1 per cent through 20 mesh. A 5- by 22-ft. Allis-Chalmers two-compartment compeb mill, 200 h.p., 26 r.p.m., operating on 1-in. feed, had an hourly capacity of 9 tons, 78 per cent through 200 mesh. An 8-ft. by 60-in., 345-h.p. Hardinge conical mill, 18 r.p.m., operating on 3/8-in. feed with 2 per cent moisture had an hourly capacity of 26 tons, 91 per cent through 200 mesh; type of air classifier, 12 ft. superfine, with 145-h.p. fan; lining and balls, steel.

The present tendency is to connect the mills in closed circuit with air separators, increasing output 50 to 60 per cent with 25 to 35 per cent reduction in power consumption. A flow sheet for a 9½- by 8- by 40-ft. compeb mill and two separators is shown in Fig. 21. Fifty tons of raw material enter the mill. At the first compartment a peripheral screen removes the large oversize material and returns it to the front end of the mill. The final product from this compartment (38 per cent through 200 mesh) passes to an air separator which divides the initial 50 tons into fines and tailings. Fines are considered to be about 13 tons testing 92 per cent through 200 mesh, tailings about 37 tons testing 18 per cent through 200 mesh. The tailings enter the second compartment of the mill; the product from this compartment enters the second separator which removes the fines, the tailings entering the third compartment, the discharge from which is likewise fed to the second separator.[1]

Wet Grinding. An 8- by 4-ft., 150-h.p. Hardinge mill, 17 r.p.m., with trommel screen, metal lining, and steel balls, grinding an 8-mesh feed, showed an hourly capacity of 32 tons, 100 per cent through 10 mesh; ball consumption 0.24 lb. per ton. A 7- by 26-ft. Allis-Chalmers two-compartment compeb mill, 500 h.p., 20 r.p.m., with Dorr rake in closed circuit with the primary compartment and Dorr bowl in closed circuit with the secondary compartment, gave the operating characteristics shown in Table 65.

Table 65. Operating Data for Mill with Hydraulic Classifier

	Open circuit	Closed circuit
Capacity per hr., bbl.	70.1	167
Power consumption, kw.-hr. per bbl.	5.5	3.01
Loss of grinding mediums, lb. per bbl.		
In primary compartment	0.086	0.057
In secondary compartment	0.321	0.112
Fineness of product, % through 200 mesh	8	97
% through 100 mesh	95	
Charge in primary compartment	13 tons balls, 2¼–4 in.	
Charge in secondary compartment	39 tons balls, ¾–⅞ in.	

Clinker is ground by the same equipment in both processes; ball mills, tube mills, and compartment mills are used. Mills of the ring roll type, such as the Bradley-Hercules, are sometimes used as preliminary to a tube mill. In recent years closed-circuit systems have been favored. A 7- by 24-ft. Allis-Chalmers mill with 185-kw. power consumption in open circuit gave a capacity of 46 bbl. cement per hour, 97.9 per cent through 200 mesh; clinker feed 115°F., discharge 174°F. An 8- by 7- by 40-ft. three-compartment mill, in open circuit, operating on 2½-in. clinker, produced 2000 bbl. per 24 hr. Percentages through 200 mesh at the end of first, second, and third compartment were 34, 60, and 88 to 90, respectively; the first compartment using 2- to 4-in. balls, the second, 1¼- to 2-in. balls or cylpebs, and the third ⅝-in. cylpebs.

Data on grinding charges in a 7-ft., 3-in. by 42-ft., 8-in. three-compartment mill are given in Table 66.

The results obtained in grinding clinker in a Bradley-Hercules as preliminary mill is given in Table 67. Discharge from the Bradley-Hercules

---

[1] The various hook-ups of pulverizing machinery with air separation in the cement industry were furnished by E. L. Davis.

goes to intermediate mills, and the discharge of the three mills goes to a fourth mill for final grinding.

Table 66. Operating Data, Grinding Cement in Open Circuit

First compartment, grinding charge,	4-in. balls, lb.	12,000
	3.5-in. balls, lb.	7,000
	3-in. balls, lb.	3,500
	2.5-in. balls, lb.	1,750
	2-in. balls, lb.	875
	Total, lb.	25,125
Second compartment, grinding charge,	1.5 -in. balls, lb.	10,000
	1.25-in. balls, lb.	10,000
	1-in. balls, lb.	4,000
	Total, lb.	24,000
Third compartment, grinding charge,	⅞-in. balls, lb.	10,000
	¾-in. balls, lb.	45,000
	Total, lb.	55,000
Capacity, barrels per hr.		90.2
Fineness of product, % through 200 mesh		91.7
Power required, h.p.		570
Maintenance cost per barrel of cement:		
Labor		$0.003
Material		0.005
Total		$0.008

Closed-circuit Grinding of Cement Clinker. Figure 22 shows a compeb mill with two separators. Undersize from the peripheral screen goes to the first separator, which makes fines going to the fourth compartment and tailings going to the second compartment, the latter discharging to separator two. Here two products are made, fines for the fourth compartment, tailings for the third. Table 68 gives the results obtained when operating this system.

Table 67. Operating Data for Hercules Mill, Grinding Cement Clinker*

Preliminary mill	Bradley-Hercules with 9-mesh screen
Fineness of product	97% through 20 mesh, 65.5% through 100 mesh, 51% through 200 mesh
Production per hr., bbl.	75.5
Horse power per bbl. cement	4.64
Intermediate mills	6 × 22 ft. with 15% load of grinding mediums
Fineness of output	95.6% through 200 mesh
Temperature of clinker, °F	240
Size of fourth mill for finishing	6′-6″ × 22′
Grinding mediums, %	15
Fineness of finished product	97.6% through 200 mesh 96% through 300 mesh
Temperature of mill, °F	257
Horse power per bbl. cement	3.97

* Most of the data on open-circuit grinding of raw mix and clinker with Bradley Mills and tube mills were furnished by Mr. H. H. Leh, General Manager of the Keystone Portland Cement Co.

It has been found that the time of setting and the strength of many cements vary with the fineness to which the cement has been ground. A fineness between 94 and 98 per cent through 325 mesh is often required to obtain a cement having the desired properties. An air separator properly connected

Table 68. Operating Data, Grinding Cement Clinker in Closed Circuit

Size of mill, ft.	9.5 × 8 × 40
Size of mill motor, h.p.	1100
Capacity, kiln-run clinker fed to mill, bbl.* per hr.	180
Temperature of kiln-run clinker, °F.	280

Fineness of feed to the two separators:

	Separator 1, %	Separator 2, %
10 mesh	95.2	
20 mesh	82.0	98.4
28 mesh	73.2	97.6
48 mesh	56.8	91.2
100 mesh	42.4	68.4
200 mesh	32.8	46.4

Fineness of finished product, through 200 mesh, %	90.2
Temperature of finished cement, °F.	250

* The capacity at times was 250 bbl. per hr., with a fineness of 88 per cent through 200 mesh.

Table 69. Operating Data, Grinding Cement to a Very Fine Product

Type and size of mill, ft.	Three-compartment, 7.5 × 43
Diam. separator, ft.	18
Hourly capacity, bbl.	46
Feed to separator, % through 200 mesh	98.7
Tailings from separator, % through 200 mesh	83.2
Finished product, % through 200 mesh	99.8
through 325 mesh	97.9
-30μ	92.1
-25μ	83.8
-20μ	72.9
Total power required, h.p.	746
Horse power per barrel of product	12.1

with the mills will often decrease the power required. Table 69 gives the results obtained when grinding cement to a fine state of division in order to produce a superior product.

When this cement was ground in open circuit to produce a comparable material, the power required was 19.7 h.p. per bbl. cement. A 10- by 6.5-ft. Hardinge mill with air classifier, grinding a ½-in. clinker to 82 per cent through 200 mesh, showed a power consumption of 4.95 h.p. per bbl. with an hourly capacity of 102 bbl. (19.2 tons). Grinding to 88 per cent through 200 mesh, the consumption was 5.65 h.p. per bbl. and the ca-

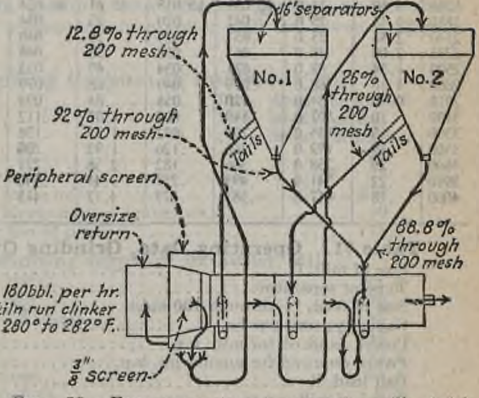


FIG. 22.—Four-compartment compeb mill (with peripheral screen on first compartment) in closed circuit with two air separators grinding and separating cement clinker.

pacity 88 bbl. (16.6 tons) per hr. Ball load in both cases was 60,000 lb. and mill speed 18 r.p.m.

At present the distribution of particle sizes in cement is determined by a turbidimeter, generally of the Wagner type. By determining the distribution of particle sizes, it is possible to calculate the surface area of the pulverized product in terms of square centimeters per gram. A product having a high surface area indicates that it contains a higher percentage of impalpable powder than the same material when ground to a lower surface area. The use of the turbidimeter has become standard in the cement industry. The time of set and strength of cement varies with the surface area of the pulverized cement. A high strength and early set are generally obtained by grinding and separating the cement to a very fine state of division. The surface area of the average cement may run anywhere from 1700 to 2000 sq. cm. per g., while the surface area of the high early cement may run anywhere from 2700 to 3200 sq. cm. per g.

The surface area of a pulverized product is being determined more and more in connection with the pulverization of many other minerals. A product should, of course, not be ground to a surface much higher than is absolutely necessary since the cost of pulverization and separation increases at a very rapid rate as the surface area increases. This is indicated in Table 70, which shows the cost of grinding slate used as a filler for rubber or as an ingredient in asphalt. The cost ratio will generally be the same for most non-metallics.

Table 70. Cost Per Ton of Pulverizing to Different Surface Areas

Surface area	Tons per hr.	Kw.-hr. per ton	Depreciation	Interest on investment	Power at 1 ct. per kw.-hr.	Maintenance	Labor at 60 cts. per hr.	Lubrication and miscellaneous costs	Total cost of pulverizing per ton	Cost ratio
1700	4.30	21.0	$0.031	$0.015	$0.21	$0.024	$0.065	$0.015	$0.36	1.00
1850	3.40	25.0	.042	.020	.25	.034	.075	.018	.44	1.22
2140	2.38	33.0	.051	.025	.33	.040	.100	.020	.57	1.56
2275	2.00	38.0	.061	.030	.38	.048	.123	.023	.67	1.86
2500	1.54	49.0	.071	.034	.49	.055	.165	.035	.85	2.36
2800	1.10	68.0	.099	.049	.68	.079	.184	.042	1.13	3.14
2910	0.90	84.0	.120	.058	.84	.094	.192	.055	1.36	3.88
3100	.70	107.0	.149	.069	1.07	.112	.210	.068	1.68	4.67
3300	.55	135.0	.200	.092	1.35	.158	.233	.074	2.11	5.85
3500	.39	192.0	.266	.126	1.92	.200	.296	.086	2.90	8.06
3660	.29	258.0	.366	.182	2.58	.271	.315	.090	3.81	10.6
3910	.22	341.0	.493	.237	3.41	.340	.356	.095	4.93	13.7
4060	.18	417.0	.585	.279	4.17	.415	.394	.100	5.95	16.5

Table 71. Operating Data, Grinding Quicklime in Ball Mill

Size of mill, ft.	7 × 3
Type of separator	Hardinge air classifier
Size of feed, % through 100 mesh	90
Capacity, tons per hr.	12.5
Power required for mill, h.p.	100
Power required for auxiliaries, h.p.	35
Ball load, lb.	20,000
Speed of mill, r.p.m.	20

Lime for agricultural purposes is generally ground in hammer mills. Where a fine product is desired, as in the building trade and for chemical manufacture, ring roll mills, ball mills, and certain types of hammer mills are used. Table 71 gives the operating data for a Hardinge mill grinding quicklime.

For certain chemical purposes quicklime is ground very fine. The beet-sugar industry takes a product with 99 to 99.5 per cent through 200 mesh. Table 72 gives capacities for various sizes of Raymond mills equipped with air separation.

Table 72. Operating Data on Raymond Mills, Grinding Lime

	Mill No. 4	Mill No. 5	Mill No. 10
Fineness of product, % through 200 mesh	99.5	99	99
Analysis of lime, % CaO	98.5	96.2	95
Type of kiln	Shaft	Shaft	Rotary
Tons per hr	2.5	3	5.4
Power required, total h.p.	70	75	110
h.p. per ton	28	25	20.3

Lime coming from the hydrator is often pulverized without separating out the impurities, by the use of Bonnot mills, Raymond roller mills, or ball mills. As a rule it is really not pulverized but air-separated to remove impurities such as sand, overburned, underburned, and core. The hydrate is passed through an automatic pulverizer with air separation and throwout. A recent tendency is to handle this material with air separators in series, the tailings from the final separator being discarded or sold for agricultural use. A modification of this system is to feed the tailings to an automatic pulverizer with throwout for final cleaning. Very clean tailings are thus obtained, cleaner than in an air-separation unit.

Gypsum is usually calcined in kettles or rotary calciners, after reduction to a fineness varying from 75 to 95 to 98 per cent through 100 mesh. Mills of the ring roll type, equipped with air separation, and, to a lesser extent, ball mills are used, although buhrstones are still in use in many of the older plants. Results obtained in grinding Nova Scotia gypsum or buhrstone mills in closed circuit with screen and in a roller mill are given in Tables 73 and 74.

Table 73. Grinding Gypsum in Buhrstone Mill

Number of buhrstones	Two 42-in. horizontal Two 36-in. vertical Three 30-in. horizontal
Capacity, tons per hr	20
Fineness, % through 100 mesh	73
Horse power required by stone mills	210
Horse power required by elevators and conveyors	80
Horse power required per ton of product	14.5

Table 74. Grinding Gypsum in Roller Mill

Size of mill	No. 5
Horse power required by mill	70
Horse power required by exhauster	45
Total h.p.	115
Capacity, tons per hr	12.5
Product, % through 100 mesh	84.5
Horse power per ton of product	9.2

In the following list gypsum from various sources has been arranged in order of increasing resistance to grinding: Iowa, New York, Nova Scotia, Kansas, Michigan, Ohio, Wyoming, Virginia, Texas, Nevada, and Montana. Gypsum calcined in a rotary kiln is usually pulverized after calcination. Ring roll mills and tube mills with air separation are sometimes employed, but more often intermittently operating tube mills are used.

Coal, Coke, and Other Carbon Products. *Bituminous Coal.* The grinding characteristics of bituminous coal are affected by impurities contained, such as inherent ash, slate, gravel, sand, and sulfur balls. The grindability of a coal is determined by grinding it in a standard laboratory mill and comparing the results with the results obtained under identical conditions on a coal selected as a standard.

This standard coal is a low volatile coal from Jerome Mines, Upper Kittaning bed, Somerset Co., Penn., and is assumed to have a grindability of 100. Thus a coal with a grindability of 125 could be pulverized easier than the standard, while a coal with a grindability of 70 would be more difficult to grind. In general we assume that the capacity obtained on a mill will be proportional to grindability. Rittinger's law states that the energy required for grinding is proportional to the new surface area produced. For example, in the following table we show how the total surface increases with the fineness of grinding. For simplicity we have assumed cubical particles:

Table 75. Surface in Relation to Particle Size

Size, in.	Number particles	Total surface, sq. in.
1.	1	6
0.1	1,000	60
.01	1,000,000	600
.005	8,000,000	1200
.004	15,625,000	1500
.002	125,000,000	3000

In the first line we show one 1-in. cube. It has a surface area of 6 sq. in. The second line shows that the 1-in. cube has been pulverized so as to obtain 1000 small cubes each 0.1 in. in diameter with a total surface area of 60. There are 600 sq. in. of surface produced when grinding, so that each particle is 0.01 in. in size, and so on, until, when we grind to a particle size of 0.002, we produce a total surface area of 3000 sq. in. It will also be noted that the surface area produced varies as the reciprocal of the particle size. This is really another way of stating Rittinger's law.

If we now pulverize a sample of material then grade it on various screens to determine the different sizes produced, we can calculate the new surface produced, and, as stated above, by comparing this new surface produced with the new surface produced on a standard sample under similar testing conditions, we can determine the grindability. There are two general methods recommended for determining the grindability of coal: one is the Hardgrove method and the other the U.S. Bureau of Mines or ball mill method.

The Hardgrove Method. A special machine is used which is manufactured by the Babcock & Wilcox Co. Essentially this pulverizer consists of a stationary bowl containing eight 1-in. balls, each 1 in. in diameter, which are driven by a rotating top ring. The pressure on the balls is produced by the shaft, top grinding ring, gear, and weights added to give a definite pressure on the balls that shall total to 64 lb. A predetermining counter is used to stop the machine after it has made exactly 60 revolutions.

In the following table we assume that certain results were obtained on a sample of coal tested as outlined above and finally show how the grindability factor is determined:

Table 76. Grindability Data Obtained on a Coal Sample

Micron designation	U.S. Standard sieve series numbers	Grams between the 2 sieves	Surface-unit factor	Final surface units
1190–590	16– 30	14.3	1.35	19.3
590–250	30– 60	17.3	2.86	49.5
250–149	60–100	6.4	6.02	38.6
149–105	100–140	3.2	9.45	30.3
105– 74	140–200	2.2	13.40	29.5
74– 62	200–230	1.4	17.70	24.8
62– 44	230–325	0.6	22.60	13.6
44	Passing 325	4.6	47.50	219.0
		50.0		424.6

In column 4 we list the surface-unit factor for the average particle size between the two screens.

The surface units are equivalent to specific area in square centimeters per gram with a sp. gr. of 1. We might also define surface units as the square centimeters per cubic centimeter of a solid material.

The surface-unit factor may be obtained from the following formula:

$$S \text{ (surface-unit factor)} = \frac{1200}{\dfrac{Y_1 + Y_2}{2}}$$

where Y_1 and Y_2 are the openings of the two screens. For example in the first line of the table $Y_1 = 1190$ and $Y_2 = 590$.

$$\therefore S = \frac{1200}{\dfrac{590 + 1190}{2}} = \frac{1200}{890} = 1.35$$

The result 1.35 is the surface-unit factor per gram. Therefore, multiplying this unit factor by 14.3, the grams between the two screens, we get 19.3 as the total surface units for that fraction.

The remaining fractions are calculated in a similar manner. One assumption had to be made for determining the average particle size of the -325-mesh particles. Here Y_1 equals 44 and Y_2, the lower limit of the finest particle, is about 6.6μ, or the average particle size of the -325-mesh screen was thus assumed to be about 25.3μ. We therefore have the following results:

Final surface unit of sample	424.6
Original surface unit of sample	67.5
New surface units produced	357.1

New surface units of the standard or coal with a grindability of 100 has been found to be 656. Therefore the grindability of the sample tested is

$$\frac{357.1}{656} = 54.9$$

The original surface units obtained by multiplying the original 50 g. by the surface-unit factor of the material passing the No. 16 and remaining on the No. 30 screens.

Bureau of Mines or Ball Method. In the above method the relative amount of energy necessary to pulverize different coal is determined by placing a sample of coal in a ball mill and determining the number of revolutions required to grind it so that 80 per cent of the sample passes a 74μ (No. 200-mesh) sieve. The average number of revolutions necessary to grind the sample so that 80 per cent passes the 74μ (200-mesh) sieve is expressed in the following two ways:

$$\text{Grindability (lb. per hr.)} = \frac{2645}{\text{average number of revolutions}} \qquad (1)$$

$$\text{Grindability (per cent)} = \frac{72{,}000}{\text{average number of revolutions}} \qquad (2)$$

In the second formula we note that, if we make the average number of revolutions 720, we obtain the per cent of grindability as 100, and, if we substitute 720 for the revolutions in the first formula, we obtain 3.67 on the grindability in pounds per hour.

We therefore take as a standard a coal which requires 720 revolutions on the testing mill to grind it so that 80 per cent will pass through a 74μ (200-mesh) sieve. This standard is then assumed to have a grindability of 100. All other coals will then have a grindability inversely proportional to the number of revolutions it takes in the test for the coals in comparison with the number of revolutions required by the standard. For example, if it takes 1080 revolutions for a certain coal, its grindability will be $720/1080 = 0.66\frac{2}{3}$ or $66\frac{2}{3}$ per cent. We get the same results, of course, by substituting 1080 in formula (2) for the number of revolutions.

There are two general methods of burning pulverized coal, the unit system, in which the coal is blown directly into the furnace as it is pulverized; and the central grinding system, in which the pulverized coal is stored in a bin and transported to hoppers at the furnaces. Mills used in the storage system are described under the various types of mills. Mills used in the unit system were formerly high-speed hammer mills with air separation; at present, tube mills, ball mills, and roller mills are also used. The Aero and Riley attritor are inherently unit pulverizers.

The Riley attritor is available in three single types and two duplex types. Capacities vary from 2500 lb. for the smallest unit to 15,000 lb. for the largest duplex unit. This type of crusher utilizes a series of swing hammers pivoted to the rotor hub, around which is a stationary grid, cut away at one section so that foreign material is thrown out. After passing through this first effect, the coal is carried in a current of air into the second effect, alternate rows of moving and stationary pegs, where most of the pulverizing is done. Leaving the second effect, the coal is passed through a rejector, a number of scoop-like blades on the main shaft, where the heaviest particles are thrown back into the pulverizing compartment, permitting the passage of the finer particles only, which enter the fan inlet and are carried into the furnace. Hot air can be introduced into the machine for drying the coal. Air at 300°F. dries coal with 8 per cent moisture down to about 1 per cent.

The Whiting hammer mills are of the Vertial type and are greatly used for pulverizing coal and direct-firing industrial furnaces.

The Strong Scott pulverizer is of the beater type, equipped with an integral air classifier, and is extensively used for pulverizing coal for direct-firing boilers and rotary cement kilns.

The Aero unit pulverizer is used for coal, pitch, and coke, blowing the ground material directly into the furnace. It is shown in cross section in Fig. 23. It has a shell made in the form of a cylinder split horizontally; the upper part of the casing is hinged and may be opened for inspection, and the housing is divided by steel diaphragms into several pulverizing chambers. Two openings through the shell, between the fan chamber and the last of the pulverizing stages, admit primary air. The rotor consists of a built-up unit assembled on a heavy steel shaft, which carries steel-plate disks upon which alloy-steel paddles are fixed in position, one set for each pulverizing chamber. Near one end is the fan for the feed mechanism, a table-type feeder. Flow of coal into the machine is regulated by the speed of the revolving table and a vertically mounted sleeve moving in the coal intake pipe directly above the table.

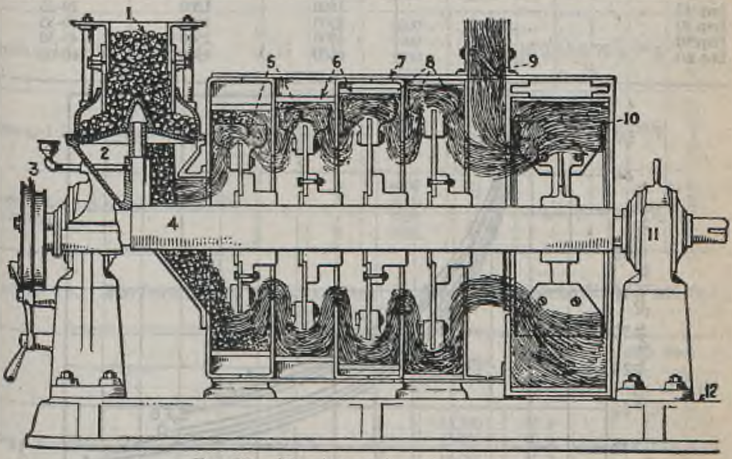


FIG. 23.—The Aero unit coal pulverizer.

Refractory material such as tramp iron is removed in the first pulverizing chamber and eliminated through a tramp-iron pocket. The lumps of coal are shattered by impact with the paddles and hurled by centrifugal force against the liners of the chamber; the coal is held in the first stage by the diaphragm until it is fine enough to be lifted by the draft of air and carried over the diaphragm against the centrifugal force of rotation. An adjustable air damper is incorporated with the feed-mechanism end, through which a small amount of conditioned air is admitted. This flow of air lifts the fine particles of coal over the baffles between the pulverizing stages and affords a means for controlling the fineness of the finished product.

As the successive stages increase in diameter, the centrifugal force becomes higher as the coal travels toward the fan. With the larger cross section, the air velocity becomes lower, causing the coal particles to remain in the successive stages until reduced to a size which will enable the pull of the air current toward the fan to exceed the centrifugal force, thus pulling it out of the machine. The carrier air port at the feed mechanism can be arranged for connection to a hot-air or gas supply for introducing hot gases into the pulverizer. This will dry the material, thus increasing the capacity and decreasing power consumption. The warm, dry particles tend to flow freely through the duct system and to give better and more efficient combustion.

Capacities of the Imp mill (Raymond hammer mill) grinding an average-grade bituminous coal are given in Table 77.

Table 77. Capacities of Raymond Hammer Mills Grinding Coal

Type	Lb. per hr.		Horse power
	65–70% through 200 mesh	80–85% through 200 mesh	
Imp 32	1800	1200	20–25
Imp 40	2500	1600	30–35
Imp 50	4000	2500	45–50
Imp 60	6500	4500	60–65

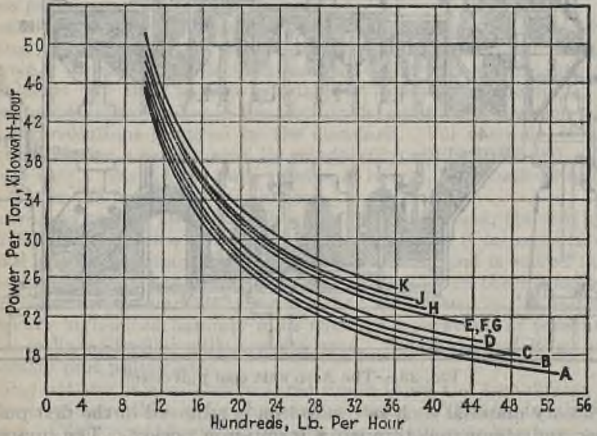


FIG. 24.—The variation in power relative to capacity of a Raymond impact pulverizer when grinding various grades of bituminous coal.

In the Hardinge conical mill for pulverizing coal the air enters the mill through the annular space between the mill trunnion and the central outlet pipe. A double-acting control valve regulates the amount of air going through the mill as well as the air passing through the fan to the burner. The regulation of the air controls the amount of coal leaving the mill, hence the output. Adjustments for fineness are made by regulating the position of the central fuel-discharge pipe. The feed is regulated automatically through a link mechanism to correspond to the amount withdrawn.

A 7- by 48-in. Hardinge mill, 127 h.p., 23 r.p.m., with steel ball and liners and with central-discharge air classifier, grinding a 1-in. coal with 4 per cent moisture, gave a capacity of 5.5 tons per hr., 78 per cent through 200 mesh.

The Kennedy air-swept tube mill is of relatively short length. No screens are used either inside or out; the fine coal is mixed thoroughly with air and

Table 78. Performance of Raymond Imp Hammer Mill Grinding Various Grades of Bituminous Coal

Type	Ash, %	Moisture, %	Capacity, lb. per hr.	Kw.-hr. per ton	Through mesh				Type of coal
					200	100	60	40	
New River, Pocahontas, and similar types	8	3	5300	16.0	65%	85%	95%	99%	A
		5	5100	16.9					B
		7	4800	18.0					C
		9	4500	19.3					D
Pittsburgh, Ohio, Alabama, and eastern West Virginia	12	4	4300	20.0	65%	85%	95%	99%	E
		6	4100	21.5					F
		8	3900	22.8					H
		10	3600	25.0					K
Illinois and Indiana	15	10	4000	22.2	65%	85%	95%	99%	G
		12	3900	22.8					H
		14	3800	23.5					J
		16	3600	25.0					K
Texas and Dakota lignites	14	30	4000	22.2	65%	85%	95%	99%	G
		32	3900	22.8					H
		34	3800	23.5					J
		36	3600	25.0					K
Eastern Kentucky and western West Virginia	9	3	4000	22.2	65%	85%	95%	99%	G
		5	3900	22.8					H
		7	3800	23.5					J
		9	3600	25.0					K

Table 79. Performance of Raymond Roller Mill, Grinding Various Types of Coal

Type	Ash, %	Moisture, %	Capacity, lb. per hr.	Kw.-hr. per ton	Through mesh			
					200	100	60	40
New River, Pocahontas, and similar types	8	3	13,500	10.4	70%	90%	96%	99%
		5	13,000	10.8				
		7	12,300	11.4				
		9	11,500	12.2				
Pittsburgh, Ohio, Alabama, and eastern West Virginia	12	4	9,000	15.5	65%	85%	95%	99%
		6	8,600	16.3				
		8	8,000	17.5				
		10	7,200	19.5				
Illinois and Indiana	15	8	9,000	15.5	65%	85%	95%	99%
		10	9,000	15.5				
		12	8,600	16.3				
		14	8,000	17.5				
		16	7,200	19.5				
Texas and Dakota lignites	14	28	9,000	15.5	65%	85%	95%	99%
		30	9,000	15.5				
		32	8,600	16.3				
		34	8,000	17.5				
		36	7,200	19.5				
Eastern Kentucky and eastern West Virginia	9	3	8,500	17.6	65%	85%	95%	99%
		5	8,300	18.0				
		7	7,800	19.2				
		9	7,000	21.4				

air-floated to the burners. The mill rotates slowly at a speed of 19 to 50 r.p.m., depending on the size. Hot air is used if drying is desired.

The curves shown in Fig. 24 give the performance of a Raymond Imp hammer mill when used for direct firing and grinding various grades of bituminous coal. Letters A, B, etc., refer to the type of coal pulverized. The same letter is used for different coals where they have the same grinding characteristics. Table 78 designates the type of coal referred to and also gives more detailed data. Table 79 gives the performance of a Raymond roller mill when used as a unit mill.[1]

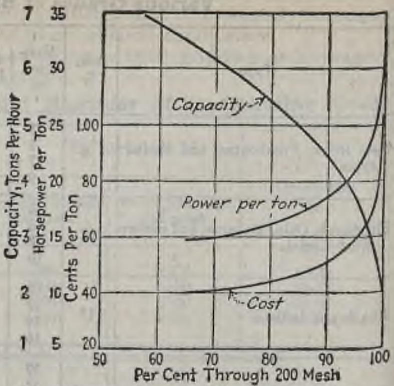


FIG. 25.—The variation in capacity, power, and cost of grinding relative to the fineness of grind.

Mills generally used in the central grinding or storage systems are the Babcock & Wilcox, Raymond, and Hardinge mills. Cost of installation of a storage system is considerably higher than for a unit system using high-speed hammer mills, but cost of operation is less. Since ball mills and roller mills with air separation have been adopted for direct firing, the disparity between the two systems has been somewhat lessened.

The graphs in Fig. 25 show how the capacity, power, and cost of grinding vary with the fineness of the product. These data apply to a moderate-size mill grinding at a rate of 200 tons per 24 hr. Power cost is based on a rate of 2 cts. per kw.-hr. and labor at 50 cts. per hr. The coal was a Pennsylvania bituminous coal, from Cambria county, lower Kittanning seam, with a moisture content under 2 per cent. Where the power rate is about 2 cts., the cost of power may be from 50 to 60 per cent of the total grinding cost. Total cost shown in the graphs includes interest on investment, depreciation, and maintenance.

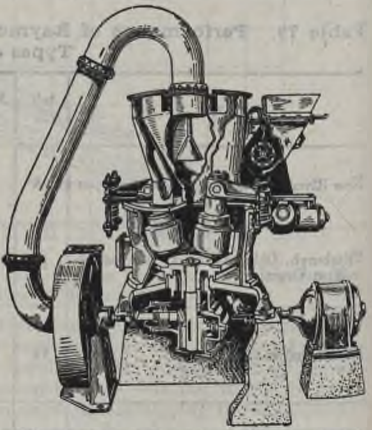


FIG. 26.—Inside construction of the bowl mill.

Direct firing of boilers and rotary kilns has become more prevalent during the past few years and is generally replacing the storage system. As previ-

---

[1] The author is obligated to Joe Crites, chief engineer of the Raymond Pulverizer Division, Combustion Engineering Company, Inc., for furnishing the graphs shown in Fig. 25 and the data contained in Tables 78, 79, and 80.

ously described, the hammer mill was generally used for direct firing, but, when this method of firing became more prevalent and was used in connection with large installations, it was found that the cost of maintenance was too high; hence new mills of the roller- and bowl-mill types have been developed and are gradually replacing all hammer-mill types.

The bowl mill is a departure in the design of the standard roller mill previously described. In this style of mill the journals that carry the grinding rolls are stationary while the grinding ring rotates. The grinding pressure is produced by means of springs, which may be adjusted to give the required pressure, and the distance between the rolls and the ring may be set to any predetermined clearance. The rolls do not touch the ring there being no metal-to-metal contact between the grinding surfaces. Figure 26 shows the inside construction of the bowl mill. The grinding ring is carried on the lip of a rotary bowl. The raw material from the feeder drops on the bowl where, owing to the centrifugal force of rotation, it is forced to the periphery and, owing to the angle of the ring, it is forced upward between the ring and the rolls where it is pulverized. The action of the tapered rolls on the tapered ring causes the pulverized material to work upward and out of the grinding chamber into an upblast of air. The air with the pulverized material passes up into a classifier of the double-cone vacuum type. Here the required fines are removed and the oversize dropped back to the bowl where it is mixed with the raw feed. This type of mill is primarily used for pulverizing coal and blowing it directly into industrial furnaces or rotary kilns. Hot gases may be used for drying the coal while it is being pulverized. Figure 27 shows a direct-fired unit blowing coal into a kiln. Hot gases are taken from the hood of the kiln and, after passing through a cinder trap for removing ash and dust, pass directly into the mill. Thermostatic control through a tempering device may be used to maintain a constant temperature in the mill irrespective of the moisture in the coal. Tramp iron and other extraneous hard materials are automatically thrown out from the mill through a tailings spout.

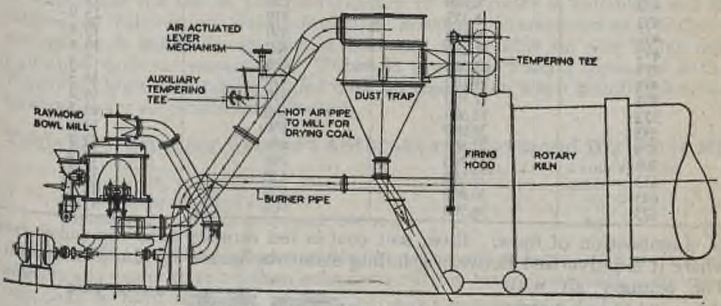


FIG. 27.—Bowl mill used for direct-firing a rotary kiln.

The following table gives pertinent data on the operation of a bowl mill when grinding a typical grade of coal having a grindability of from 55 to 60. The data are based on grinding coal to a fineness of about 80 per cent through 200 mesh.

The Babcock and Wilcox Pulverizer. Figure 28 illustrates the design of the B. & W. type E pulverizer. The type E pulverizer is a modification of the type B described on p. 1930. It is developed particularly for the direct firing of industrial furnaces. This unit requires no external equipment for drying or classification of fines. Raw, wet coal is fed directly into the pulverizer where it is pulverized between grinding elements heated by the primary air. The primary air with the material passes through a vacuum-type air classifier in the top of the pulverizer. The oversize is rejected and returned for further grinding, and the particles of proper fineness pass through the classifier into the burner pipes. The type E pulverizer is made in 15 sizes and in capacities up to 12 to 13 tons per hr.

Table 80. Operating Characteristics of Bowl Mills When Grinding 80 % through 200 Mesh

Mill size	Capacity, lb. per hr.	Lb. air per min.	Kw.-hr. per ton
271	2,520	130	18.4
312	2,980	150	18.0
312A	3,360	160	18.0
352	3,740	170	17.5
352A	4,450	180	17.5
372	5,200	200	17.0
372A	5,960	225	16.0
412	6,720	250	15.9
452	8,125	310	15.9
453	9,660	350	15.4
492	11,150	400	15.4
473	13,000	425	15.4
493	14,100	450	14.9
532	17,100	550	14.2
572	18,600	600	13.8
533	20,000	600	13.3
573	22,300	660	12.7
593	23,700	700	12.7
613	27,500	820	12.7
633	30,000	900	12.7
673	34,900	960	12.7

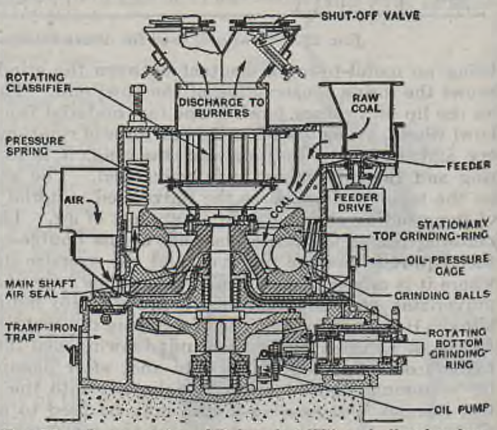


FIG. 28.—Cross section of Babcock & Wilcox ball pulverizer.

A 7- by 36-in. Hardinge conical mill equipped with air classifier, grinding 1-in. coal with 3 per cent moisture for the storage system to 72 per cent through 200 mesh, had a capacity of 8½ tons per hr. The power required for mill was 90 h.p.; for auxiliary equipment, 50 h.p.; speed, 21 r.p.m.; ball load, 19,000 lb.

Anthracite coal is generally harder to reduce than bituminous coal. It is pulverized for foundry facing mixtures, generally on buhrstones or in ball mills, followed by bolting reels or air separators. Only to a lesser extent is it used for fuel. Anthracite used for manufacture of electrodes is calcined, followed by grinding in ball mills and tube mills, or ring roller mills equipped with air separation.

A 10- by 66-in. Hardinge mill was superfine air classifier, grinding 4-mesh anthracite with 3.5 per cent moisture produced 12 tons per hr., 82 per cent through 200 mesh. The power required for mill was 370 h.p.; for auxiliaries, 70 h.p.; speed of mill, 19 r.p.m.; ball load, 57,000 lb.; cost of ball consumption, 0.7 ct. per ton.

An 8- by 10-ft. tube mill, in closed circuit with a 14-ft. centrifugal air separator, operating on a feed with 90 per cent through 40 mesh, pulverized 2 tons per hr., 99 per cent through 200 mesh, with 100 h.p. for the mill and 30 h.p. for the separator.

Anthracite for use in the manufacture of electrodes is calcined, and the degree of calcination determines the grinding characteristics. Calcined anthracite is generally ground in ball and tube mills or ring roller mills equipped with air separation. Table 81 gives the results obtained with a Raymond high-side mill equipped with air separation when grinding calcined anthracite for electrode manufacture.

Table 81. Grinding Calcined Anthracite in Raymond High-side Mill

Size of mill	4 roller, 42 in.
Capacity, lb. per hr	4600
Horse power required	70
Fineness of product, % through 200 mesh	76

Coke. The grinding characteristics of coke vary widely. Petroleum coke is generally easier to grind than coke derived from bituminous coal. By-product coke is hard and abrasive, while certain foundry and retort coke is extremely hard to grind. For certain purposes it may be necessary to produce a uniform granule with minimum fines. This is best accomplished in rod or ball mills in closed circuit with screens. Hourly capacity of a 4- by 10-ft. rod mill with screens, operating on by-product-coke breeze was 9 tons, 100 per cent through 10 mesh, and 73 per cent on 200 mesh; power requirement 40 h.p.

Table 82. Grinding Pitch in Raymond Pulverizers

	No. 0000	No. 00
Melting point of pitch, °C	100	135
Capacity, lb. per hr	550	900
Fineness of finish, % through 200 mesh	85	92
Power required, h.p	10	20

Petroleum coke is generally pulverized for manufacture of electrodes; roll mills with air separation and tube mills are generally used. A No. 5 Raymond roller mill gave an hourly output of 3.8 tons, 78.5 per cent through 200 mesh, with 90 h.p.

Pitch may be pulverized as a fuel or for other commercial purposes; in the former case the unit system of burning is generally employed, and the same equipment is used as described for coal. The grinding characteristics vary with the melting point and may be anywhere from 50 to 175° C. Table 82 gives the results obtained with several sizes of Raymond pulverizers, grinding pitch.

Natural graphite may be divided into three grades in respect to grinding characteristics: flake, crystalline, and amorphous. Flake is generally most difficult to reduce to fine powder and the crystalline variety is the most abrasive. The following is an arrangement of graphites according to origin, in order of increasing resistance to grinding: Mexico (Sonora), Michigan, Rhode Island, Korea, Ontario, East Siberia, New York, Quebec, Alabama, Madagascar, and Ceylon. Graphite is generally ground in ball mills, tube mills, on buhrstones, and in ring roll mills, with or without air separation. For large capacities, ball and tube mills are used, particularly on the flake and crystalline varieties. Operating characteristics for a tube mill with vacuum air separator in closed circuit is given in Table 83.

Table 83. Operating Characteristics of Tube Mill Grinding Graphite

	Ceylon	Madagascar	Korea
Size of mill, ft.	6 × 18	6 × 24	4 × 8
Feed, mesh	20	16	25
Discharge	40% through 250 mesh	60% through 220 mesh	50% through 100 mesh
Fineness of discharge from separator	95% through 250 mesh	97% through 220 mesh	98% through 100 mesh
Capacity, lb. per hr.	725	1500	800
Total horse power	90	125	70

A No. 4 Raymond roller mill, grinding Mexican graphite, gave a capacity of 2500 lb. per hr., 99.1 per cent through 200 mesh, with a power consumption of 75 h.p.

Artificial graphite has recently been ground in ball mills in closed circuit with air separators. For lubricants the graphite is ground wet in a paste where water is eventually replaced by oil. The colloid mill is used with marked success for production of graphite paint.

Mineral black, a shale sometimes erroneously called "rotten stone," contains a large amount of carbon, and is used as a filler for paints and other chemical operations. It is pulverized and separated with the same equipment as shale, limestone, and barytes.

Bone black is sometimes ground very fine, for paint, ink, or for chemical uses. A tube mill or a Griffin mill is often used, the mill discharging to a fan which blows the material to a series of cyclone collectors in tandem. The discharge from the first cyclone is usually returned to the mill for further grinding; the discharge from the last goes to an air filter where the finest grades are obtained. The number of cyclones used depends on the grades required. A system using a single-roll Griffin mill may produce 100 to 125 lb. per hr., using 1500 to 1600 h.p. per ton of product. A 6- by 22-ft. tube mill, followed by a vacuum air separator and a filter, would use about 140 to 150 h.p., producing about 200 lb. per hr.

Decolorizing carbons of vegetable origin should not be ground too fine. Standard fineness varies from 100 per cent through 30 mesh to 100 per cent through 50, with 50 to 70 per cent on 200 mesh as the upper limit. Ball mills, hammer mills, and rolls, followed by screens, are used. Where the material is used for filtering, a product of uniform size must be used.

Charcoal is usually ground in hammer mills with screen or air separation. For absorption of gases it is usually crushed and graded to about 16 mesh. Care should be taken to prevent it from igniting during grinding.

Gilsonite is often used in place of asphalt or pitch. It is easily pulverized and is generally reduced on hammer mills with screen or air separation. A No. 00 Raymond beater mill with vacuum air separation produced 950 lb. per hr., 90 per cent through 200 mesh; 650 lb. per hr., 95 per cent through 200 mesh; and 300 lb. per hr., 99.5 per cent through 200 mesh; the pulverizer was driven by a 20-h.p. motor.

Carbon black or gas black is already of extreme fineness when manufactured. The material may, however, contain a small amount of impurities which are removed by bolting. A more efficient operation is to pass the material through an air separator and the tailings from the separator to the bolting reel for removal of fines.

Carbon butts are hard and difficult to grind. The same type of equipment is used as for calcined-anthracite roller mills with air separation, or ball and tube mills. Compartment mills have also recently been used with marked success.

Carbon mixtures (green mix) are generally made from flour of petroleum coke, graphite, and lampblack, mixed with a binder such as pitch; solvents such as benzol are incorporated in the mix. After cooling, the mixture is caked and therefore generally reground. Table 84 gives the results of several grinding mills used in the carbon brush industry for grinding such mixtures.

Table 84. Grinding Carbon Mixtures

	No. 00 Raymond	Midget roller mill
Material in mix	Graphite	Lampblack, pitch, coke
Capacity, lb. per hr.	700	1350
Fineness, % through 200 mesh	65	62.5
Power required, h.p.	20	30
Type of separation	Vacuum, air	Vacuum, air

Lampblack when manufactured is usually very fine. The gas is passed through baffled chambers or ducts in which the various grades are precipitated; the coarser grades are often pulverized for the carbon brush industry. Grinding may be done in ball mills, on buhrstones, hammer mills, or roller mills, with or without air separation. Where an extremely fine product is required, the same system as described for bone black may be used. A hammer mill equipped with air separation ground about 200 lb. per hr. to a fineness of 95 per cent through 200 mesh, with a power consumption of about 20 h.p.

Chemicals, Dry Colors, and Dyes. Most chemicals, dry colors, and dyes offer little resistance to disintegration, but other difficulties incidental to grinding arise. Chief among these is the agglomeration, or balling, of the pulverized material. This occurs particularly with certain precipitates or filter-press products.

Dry colors and dyestuffs are generally pulverized in small units, such as hammer mills or jar and pebble mills. For very small productions the jar mill is probably the most practicable, as it is easily cleaned when changing from one color to another. A train of any number of such jar mills can be operated from one drive, each producing a different material. These mills are particularly adapted for wet grinding, although they are also frequently used for dry grinding. However, in the latter case, there is a tendency for the material to coat the balls and to adhere to the lining, thus decreasing the efficiency of the grinding operation.

All other types of mills, such as hammer mills with screen and air separation, are constructed so they can be readily cleaned when changing from one material to another.

Most colors are not ground very finely in a dry state, inasmuch as at a later stage they are ground wet in pebble mills or in rolls with the proper vehicle. Some dyes, however, are ground very fine; if very hard and crystalline, they are reduced in an air-separation pulverizer or in a screen hammer mill in closed circuit with a bolting reel or an air separator.

A few of the basic chemical pigments used in the manufacture of paints are, however, pulverized in large quantities, and, as no necessity exists for cleaning the pulverizing equipment, they are often ground in large roller mills of the ring roll type, equipped with air separation. Some operating data for grinding several of the basic pigments are given in Table 85.

Table 85. Operating Data, Grinding Pigments in Ring Roller Mills

	Capacity, lb. per hr.	Horse power per ton	Fineness of product
Barium sulfate	4500	28	1% on 325 mesh
Zinc sulfate	5000	32	1% on 325 mesh
Barium carbonate	4600	29	0.9% on 325 mesh
Lithopone	5000	25	0.1% on 325 mesh
Titanium with barium base	2000	63	0.1% on 325 mesh
Titanium with calcium base	2200	57	0.08% on 325 mesh
Titanium dioxide	1800	70	0.05% on 325 mesh
Titanium with aluminum silicate	2800	45	0.08% on 325 mesh
White lead	5500	23	0.5% on 325 mesh

The properties of the pigments depend greatly upon the type of mill used and upon the conditions under which they are ground. These factors affect both particle size and shape. In grinding certain *titanium pigments* it is often desirable first to grind in a roller mill with air separation; the product from this first grinding may then be processed through a high-speed beater mill of the Imp type. A material increase in tinting strength often results from such treatment.

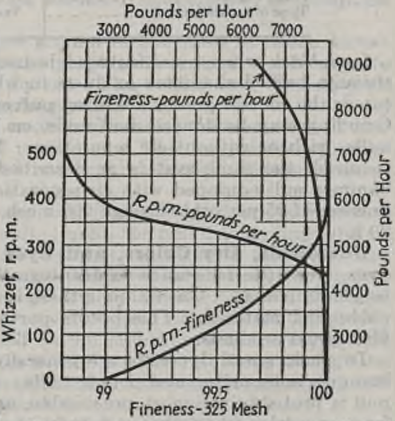


Fig. 29.—Operating characteristics of mill equipped with integral whizzer separator when grinding a titanium pigment.

Figure 29 shows the grinding characteristics of a mill equipped with whizzer separation when grinding a titanium pigment to different finenesses.

Lead oxides are generally first ground in high-speed automatic pulverizers with air separation and automatic throwout. This method is particularly adapted to grinding of incompletely oxidized materials and those containing an appreciable amount of metallic lead, which is eliminated through the throwout. The objective in the production of certain lead oxides is to obtain a product of lowest possible apparent density. Different types of mills produce oxides of varying densities, even though the screen analysis may be quite similar. How the type of mill used and the grinding method applied affect the apparent density of the product may be seen from the following test: The lead oxide was first passed through a pulverizer with throwout, for removal of the metallic lead. It next went to a roller mill equipped with air separation and was subjected to extremely fine grinding. Finally, the product from the roller mill went to an Imp pulverizer equipped with whizzer separation. This pulverizer both changed the particle shaped and reduced the apparent density of the material. The results are given in Table 86.

Calcium arsenate is very sticky and has a tendency to adhere to the grinding equipment. A high-speed pulverizer, with air separation and air conveying, gives a product possessing proper physical properties. Table 87 gives the operating characteristics for a Raymond No. 00 automatic pulverizer, grinding calcium arsenate.

Table 86. Data Showing the Influence of the Type of Mill upon Apparent Density of Lead Oxide

Type of automatic pulverizer with throwout	No. 1
Power (total) required by pulverizer system, h.p.	75
Production, lb. per hr.	4000
Apparent density, g. per cu. in.	38
Size of roller mill	No. 3
Power required by mill, h.p.	75
Capacity, lb. per hr.	4000
Apparent density, g. per cu. in.	27
Size of Imp pulverizer	No. 50
Power required by pulverizer, h.p.	75
Production, lb. per hr.	1900
Apparent density, g. per cu. in.	18

Table 87. Operating Characteristics for a No. 00 Raymond, Grinding Calcium Arsenate

% through 200 mesh	Apparent density, cu. in. per lb.	Production, lb. per hr.	Horse power
90	81.5	1300	28
94	86.0	1100	28
96	90.6	950	27
98	94.2	750	27
99	98.7	625	25
99.5	104.8	550	24
99.9	110.0	475	22
99.99	124.8	400	20

A great many insecticides are made by extracting the toxic agent from derris root and pyrethrum flowers. The use of these materials has become very extensive within the past few years. The term "derris root" is used rather indiscriminately for different species of roots containing rotenone as the toxic agent. Amongst these roots are derris root, cube root, timbo root, and barbasco. The mill generally used for grinding these roots is the roller mill with air separation. A 50-in. mill will produce about 700 lb. per hr. of cube root, about 600 lb. per hr. of derris root, and about the same amount of the other roots. The fineness to which this material is reduced is generally about 95 per cent through 200 mesh. The power consumption is from 125 to 150 kw.-hr. per ton of finished product. The same type of equipment is used for pulverizing pyrethrum flowers. The capacity on pyrethrum flowers when grinding to a fineness of 95 per cent through 200 mesh is about 25 per cent less than that obtained on derris root. The horse power is about 160 to 175 kw.-hr. per ton.

Drugs, Pharmaceuticals, and Spices. Nearly all drugs and pharmaceuticals are ground in small quantities, sometimes so small that mortar and pestle or hand-operated jar mills are used. Materials of fibrous character are first passed through a rotary cutter or a high-speed hammer mill; when a very fine product is required, stone mills or attrition mills are used. Many materials, such as tonka, vanilla beans, and rose leaves, are ground in batches in pebble mills or in small ball mills consisting of a narrow cylinder with one large ball, the diameter of which equals the width of the cylinder. Oily seeds, such as olibanum, mustard, and cochineal, when pulverized very fine, are usually ground in a pounder mill, similar to a stamp mill. Mustard, in pulp-paste or semi-paste form, is generally ground in horizontal stone mills.

A mill of different type, fitted with grinding plates, is used extensively for grinding drugs and spices. One mill of this type is the Quaker City grinding mill. Many spices are first reduced on breakers and crackers, fitted with corrugated steel cones, plates, or buhrstones. The fine product from these breakers is usually a good feed for a finishing mill.

When drugs, chemicals, and spices are to be granulated, they are usually ground by roller mills in closed circuit with bolting reel and sifter. The mill most frequently used is a four-roller mill. The upper roll pair may be set rather openly, the product passes to the lower pair and then to a sifter, where fines are removed. Tailings, after passing through a coarse screen to remove shreds, threads of fiber, and similar objectionable material are returned to the upper rolls. A 9- by 18-in., four-roller mill, grinding pepper to 30 mesh, showed an hourly capacity of 600 lb., with pulley speed 450 r.p.m. and 15 h.p., including sifter and elevators.

Resins, Gums, Waxes, and Molding Powders. The grinding characteristics of the various resins, gums, waxes, and molding powders depend greatly upon their softening temperatures. When a finely divided product is required, it is often necessary to use a water-jacketed mill or a pulverizer with air separator in which cooled air is introduced into the system. Not all waxes can be ground, inasmuch as some of them are soft at the temperatures obtainable. However, a great many of them can be powdered if precautions are taken to prevent overheating. Hammer and cage mills are generally used for this purpose.

Table 88. Operating Results with a Roller Mill Grinding Synthetic Resin

Fineness, % through 200 mesh	85	90	95	99
Capacity, lb. per hr	2500	2200	1800	1200
Horse power required	35	35	32	30

Most gums and resins, natural or artificial, when used in the paint, varnish, or plastic industries, are not ground very fine, and any hammer or cage mill will produce a suitable product. Roll crushers will often give a sufficiently fine product. Certain resins used in the bakelite industries must be pulverized very fine; pebble mills, cooled with water or brine, in closed circuit with an air separator, are used. Pulverizers equipped with an air separator, both of the roller- and hammer-mill type, are also used, especially if the finished product is to be over 98 or 99 per cent through 200 mesh. Results obtained grinding a synthetic resin in a roller mill equipped with air separator are given in Table 88.

Specifications for molding powders vary widely, from an 8-mesh to a 60-mesh product; generally the coarser products 12-, 14-, or 20-mesh material.

Specifications usually prescribe a minimum of fines (below 100 and 200 mesh). For most purposes the ideal molding powder would consist of particles testing −20 mesh and +100 mesh. Molding powders are generally produced with hammer mills, either of the screen type or equipped with air separator. The latter is usually equipped with a sifter; sometimes the material from the cyclone passes through an air separator to remove the impalpable powder.

Curves *A* and *B* in Fig. 30 give the screen analysis of a molding powder produced with screen pulverizers fitted with an 8-mesh screen. Curve *C* gives the data obtained with an air-separation pulverizer unit operated to give a minimum of 100-mesh material, which amounted to only 12 per cent. This material was passed over an 8-mesh screen to remove oversize, and the resulting product passed through an air separator to remove the 100-mesh particles. Curve *D* gives the screen analysis of the final granular product.

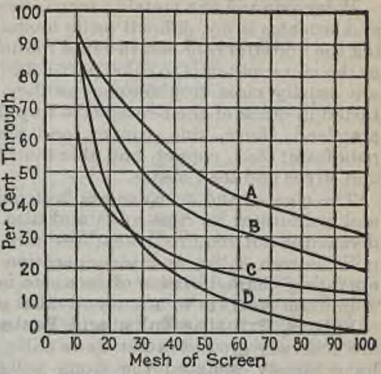


FIG. 30.—Screen analysis of molding powders produced by various installations.

The following material may be ground at ordinary temperatures if only the regular commercial fineness is required: amber, arabac, tragacanth, rosin, olibanum, gum benzoin, myrrh, guaiacum, and montan wax. If a finer product is required, hammer mills or attrition mills in closed circuit, with screens or air separators, are used.

Table 89. Operating Results Obtained with Hammer Mills Producing Soap Powder

Type of pulverizer	Fineness of mesh, in.	Production, lb. per hr.	Horse power
No. 2 Jay Bee	3/16	5000	30
Williams Infant	5/32	1000	12
No. 1 Jr. Williams	5/32	4500	35
No. 1 Raymond screen	1/8	5000	30

Grinding of Soaps. Soaps in a finely divided form may be classified as soap powder, powdered soap, and chips or flakes. The term soap powder is applied to a granular product, 12 to 16 mesh, with a certain amount of fines, which is produced in hammer mills with perforated or slotted screens. Table 89 gives the operating results obtained with several types of hammer mills producing soap powders.

The capacities given in Table 89 greatly depend upon the moisture content of the soap.

Powdered soap is a finely ground powder, with 99 per cent or more through 200 mesh. Grinding to this fineness, a No. 00 Raymond automatic pulverizer will handle 300 to 350 lb. per hr., with 15 to 17 h.p. Cooling, generally by introduction of cold air in the air-separating system, is sometimes required in grinding soap very fine. Grinding in closed circuit, with screens or air separation is often advantageous, giving a granular product and preventing overheating.

Pulverizing of the metallic soaps, stearates, palmitates, resinates, laurates, and erucates is not difficult using modern equipment with provision for keeping the material cool and in rapid motion. Batch grinding is not practicable as the material tends to cake, particularly if a fine product is needed. Oleates are usually most troublesome, as they tend to become plastic and creamy. Listed in order of their resistance to pulverization some of the metallic soaps are: lead, silver, zinc, copper, and nickel stearate; zinc, lead, and copper palmitate; lead, copper, and zinc laurate; silver, mercury, and lead erucate; and silver and lead oleate.

The oleates and erucates are best pulverized by multi-cage mills; laurates and palmitates in cage mills and also in hammer mills if particularly fine division is not required; stearates may generally be pulverized in multi-cage mills, screen mills, and air-separation hammer mills. Table 90 gives the operating characteristics of hammer mills when grinding zinc stearate and aluminum stearate to a finely divided powder.

Paints, Printing Inks, and Pastes. Paints, printing inks, and pastes are generally ground in flat stone mills, roll mills, and pebble mills; the latter have largely replaced the stone mills for grinding pigments in oil. For grinding oils, Japan, paste, paints, and creams, the buhrstone mills are so constructed that they may be water-cooled and easily taken apart for cleaning when changing from one product to another. An 18-in. 2-h.p., Day-jacketed ointment mill with 8-gal. hopper has a capacity of 400 to 600 gal.

Table 90. Operating Data for Hammer Mills Grinding Zinc and Aluminum Stearates

	Zinc stearate	Aluminum stearate
Type of pulverizer	Screen-type hammer mill	
Capacity, lb. per hr.	500	300
Fineness, % through 325 mesh	60	70
In closed circuit with air separator:		
Capacity, lb. per hr.	100	75
Fineness, % through 325 mesh	99.5	99.7
Horse power required	25	20

The roller mill is used extensively for grinding large quantities of paints, blocking, flat-drying wall paints, enamels, colors in varnishes, and various types of inks. This mill (Fig. 1*i*) consists of two or more rolls operated at different speeds, which not only tends to force the material between the rolls but also subjects it to a rubbing action. All modern roller mills have rolls bored for water cooling. When the material has been ground to proper consistency it is discharged with a scraper. Capacities vary widely with the type of material ground. Power consumption of the Day roller mills is given in Table 91.

The typical roller mill is built in standard sizes 5 by 12, 12 by 30, 16 by 40, and 20 by 60 in. Herringbone gear drive on both ends of the rolls assures even pressure and speed over the entire surface. Pressure between rolls is hydraulic and is controlled by gages so that any desired pressure may be maintained. Thus the operator is able to duplicate the fineness of any particular grind at any time. Adjustable side plates held lightly, but positively, against the end of the rolls eliminate the use of fountain stop blocks, which, when riding on the rolls, tend to wear grooves through which the material may pass unevenly ground. Operating characteristics of a Vasel mill are given in Table 92.

Table 91. Power Required by Various Day Roller Mills

Type of mill	Size, in.	Horse power
Midget, three-roll	5 × 12	2
Standard three-roll	9 × 24	5
Standard three-roll	9 × 34	7.5
Standard three-roll	12 × 22	15
Standard three-roll	16 × 40	20
Five roller	12 × 32	20–25
Five roller	14 × 38	30–50

In many cases the roller mill has advantages over the flat stone mill. Pigments can be ground to a much heavier paste; a greater rubbing effect is thus obtained. Redressing of stones is eliminated, the roller mills lasting from 8 to 10 years without regrinding. The grinding and rubber processes are visible at all times, and change from one color to another requires less than 10 min. A 12- by 32-in. roller mill has an average capacity of two or three times that of a 30-in. stone mill.

Table 92. Operating Characteristics of 16- by 40-in. Roller Mill

Product	Passes	Horse power	Weight, lb.	Time, min.	Receiving roll		Delivery roll	
					Pressure, lb.	Speed, r.p.m.	Pressure, lb.	Speed, r.p.m.
Soap	1	12	10	14⅚	450	22	520	88
Soap, 15% moisture	1	5	1¾	5	150	20	200	80
Soap, 30% moisture	1	5	1¾	4½	150	20	200	80
White, for window shades	1	6	115	20	400	24	440	96
Yellow, for window shades	1	10	70	13	260	28	420	112
Prussian blue, first pass	4	20	80	8	500	24	700	96
second pass		20		13	520	22	700	88
third pass		22		14	600	18	700	72
fourth pass		22		14	600	18	700	72
Chocolate	1	..	1265	100	480	20	240	80
Milk-chocolate paste No. 884	1	..	1350	120	420	20	250	80
Chocolate	1	..	2900	295	430	20	350	80
Red toner litho ink	3	17	80	93	700	30	450	120
Black printing ink	2	15	115	33	300	30	500	120

A comparison of the grinding area of buhrstone and pebble mills of the same diameter show that the buhr has only a fraction of the combined areas of the pebbles and the lining of pebble mills. Other advantages of pebble mills are: For the same size, the power required for the pebble mill is only three-sevenths of the buhr; grinding time is about three-eighths, and the amount of semi-paste obtained is about 1⅕ times as great. For the same power and grinding time, the pebble mill has about seven times the capacity. After allowing liberally for charging and discharging time, the pebble mill will be about 500 per cent more efficient than the buhrstone mill. Occasionally the quality of the product may be of greater consequence than the operating advantages. Buhrstone mills often produce a more desirable particle shape and may be selected for that reason.

Pebble mills are operated both wet and dry. They are used in the ceramic industries for the grinding of raw and calcined clays, flint, feldspathic materials, frit, glaze, vitreous enamel, paper coatings, and various powders and colors. In the paint industry, they are used for the grinding of heavy pastes and semi-pastes, most types of paints, undercoats, enamels, flat varnish, lacquers, stains, and lithographic colors (for tin cans). Pebble mills are also used in the ink industry.

Among the advantages offered by pebble mills are: an increase of 400 per cent in capacity with the same capital investment and the same power consumption; no preliminary mixer is required; less floor space is required; no attention is needed during grinding; there is no loss by evaporation when pigments are ground in a volatile vehicle and no loss of material on account of drying and filming; products are ground to uniform color, fineness and consistency; no dressing is required and cleaning is easy. A comparison of the relative performances of pebble and buhrstone mills is given in Table 93.

The "paint capacity" of a pebble mill is conservatively figured as 50 per cent of the volume. Total volume of the mill in Table 93 is 128 gal.; volume of pebbles, including voids, is 45 per cent of volume. Volume of voids is 40 per cent of pebble volume, which is 27 per cent of the mill volume. "Unoccupied" volume of mill is 23 per cent of the total volume.

Table 93. Comparison of Pebble Mill and Buhrstone Mill

Mill	Type	Size, ft.	Speed, r.p.m.	Horse power	Capacity, gal.	Grinding time, hr.
Pebble	B	2½ × 3½	40	3	64	1.5–2
Buhrstone water-cooled with mixer	..	2½	35	7	53	4–5

The pebble mill is particularly well adapted for the grinding of colors in oil. Paints are ground to a semi-paste consistency; the consistency may be further reduced by adding thinner to the mill and allowing the mill to rotate a short time, or by emptying the semi-paste into a mixer and adding the thinner to the mixer.

The Patterson Type D. J. pebble mill is jacketed and equipped with temperature control which permits the grinding of heavy pastes or other materials which solidify when cold. If desired, the material may be cooled during grinding by circulating brine or cold water through the jacket. Steam, hot or cold water, or brine is admitted through the shaft at one end of the mill and leaves at the other end, circulating back and forth over the entire surface of the shell. Hot or cold grinding, mixing under pressure or vacuum, drying, chemical reactions or distillation with recovery of solvents may be carried out with this type of mill. Operating characteristics of several sizes are given in Table 94.

The Abbe pebble mills for grinding paints are made in sizes from 1-ft. 6-in. diameter and 1 ft. 6 in. long to 7-ft. 6-in. diameter and 10 ft. long. These mills are of the batch or intermittent type, the cylinder being about half filled with pebbles or porcelain or metal balls. A special wet-discharge valve is used for emptying the mill. Operating data for a No. 2 mill, 5-ft. diameter and 4 ft. long, grinding batches of different kinds of pastes, are given in Table 95.

Table 96 gives the correct grinding proportions and bulking values for semi-paste colors.

Table 94. Operating Data of the D. J. Type of Pebble Mill

Size of mill, in.	Total volume without pebble charge, gal.			Capacity 50% of pebble charge. Dry sand basis, pound			Horse power	
	Silex	Porcelain	Unlined	Silex	Porcelain	Unlined	Start	Run
12 × 18	....	10	18	......	35	60	0.75	0.5
18 × 24	....	27	43	......	100	145	1	0.75
30 × 42	80	86	125	270	285	415	3	2
48 × 60	350	370	455	1,150	1,240	1,525	7.5	5
72 × 60	830	885	1000	2,800	2,950	3,450	15	9
80 × 120	3200	3400	3600	10,800	11,200	12,000	40	30

Size of mill, in.	Operating capacity (liquid basis 50% pebble charge), gallon			Operating capacity one-third pebble charge. unlined	Production after thinning, gal.	Charge, lb.	
	Silex	Porcelain	Unlined			Pebbles	Balls
12 × 18	....	6	10	14	14%	65	200
24 × 36	21	22	38	53	More than operating capacity	300	800
36 × 48	77	85	115	155		1,025	2,400
60 × 48	245	260	315	450		3,150	6,600
96 × 120	1780	1800	2000	2800		22,300	41,000

Table 95. Operating Data for No. 2 Abbe Pebble Mill

	Flat white	White enamel	Black enamel
Charge, gal.	250	250	250
Time of grinding, hr.	2–4	8–15	30–60
H.p. required:			
For starting	8	8	8
For operation	4	4	4

Table 96. Correct Grinding Proportions for Semi-paste Colors in Linseed Oil

Colors	Pigment, %	Oil, %	Bulking value, semi-paste, gal. per 100 lb.
Basic lead carbonate	80	20	4.0
Lithopone	70	30	5.82
Carbon black	20	80	11.65
Ultramarine blue	45	55	9.40
Chrome yellow	70	30	5.29
Graphite	35	65	10.18

OTHER METHODS OF COMMINUTION

INTRODUCTION

By Allan P. Colburn

The conversion of naturally occurring or manufactured materials into finely divided forms has been commercially important for a great many years. Crushing, grinding, and pulverizing of stones, many kinds of ores, and various other materials have been known and described by many generations since the time of Agricola and before. The first part of the present section (Sec. 16) describes in detail the modern crushing, grinding, and pulverizing operations, techniques, and equipment. The purpose of this part of the section is to discuss briefly other commercially important methods and equipment for producing finely divided materials that frequently must be obtained in special shapes not feasible by the ordinary methods of crushing, grinding, and pulverizing.

The demand for equipment and processes to give finely divided products has been met for the most part by the previously described processes and equipment relating to crushing, grinding, pulverizing, shredding, cutting, etc. For several centuries, however, there has been a demand for equipment and processes that would produce finely divided materials in special shapes (spheroidal, etc.). The spheroidal and other shapes of finely divided particles cannot be obtained by the ordinary comminution methods; hence special inventions were developed to meet these needs. Lead shot was perhaps the first commercially important special shape of finely-divided material that could not be produced on the then existing equipment. Shotting towers were developed for this need, and there still exist many of the old shot towers erected to produce the lead shot that was so important to pioneers in the United States both as weapons used in obtaining food and in warfare.

The growth of the chemical industries has accelerated greatly the demand for new methods of comminuting natural and manufactured materials in order to produce special shapes and sizes that are not producible by ordinary comminuting apparatus. This demand for new shapes and sizes has arisen from two principal sources: (1) from consumer or customer demand, which has required materials like soap, trisodium phosphate, sodium bisulfate in the shapes of spheres, flakes, granules, etc., which the ultimate consumer demanded on account of improved appearance, convenience, or economy of use or processing. The granulation of fertilizers is a growing development in this field. (2) From technological or manufacturing demands: the use of raw materials and intermediates in special forms, such as spheres, flakes, granules, etc., has proved to be essential and economically or technically highly desirable. Thus in many chemical processes a raw material or intermediate in a special form is required or desirable in order to effect economies in processing—either physical, as in conveying, or chemical in controlling reactions. Of increasing importance are catalytic operations, of which a prime factor is the preparation of catalysts in granular form to provide proper surface without excessive resistance to flow.

A newly growing field is that of powder metallurgy in which metallic powders are produced for many purposes such as the manufacture of porous bearings, refractory metals, sprayed coatings, etc. The saving on labor and scrap in this process is significant for its rapid development.

The remainder of this section will treat the theory of dispersion of liquid droplets, spray nozzles, spray painting, granulation, flaking, and metal powders. Information on spray drying is included in Sec. 13, Drying. It is expected that the future will show rapid development of these and other methods of comminution.

THEORY OF DISPERSION OF LIQUID DROPLETS

By R. V. Kleinschmidt

References: A very complete résumé of this subject as applied particularly to fuel atomization in internal combustion engines is given by Castleman, *Bur. Standards J. Research*, **6**, 369–376 (1931). Other references: Plateau, "Statique expérimentale et Théoretique, etc.," Paris, 1873; Scheubel, *Wiss. Ges. Luftfahrt Jahrsbuch*, 1927, p. 140.

Mechanism. The basic mechanism of droplet formation consists in drawing out the liquid into a slender stream or filament. Lord Rayleigh [*Proc. London Math. Soc.*, **10**, 4 (1879) and "Theory of Sound," Chap. XX] has shown that a liquid cylinder is unstable and that any slight displacement will cause it to neck down in places and bulge out in others, eventually collapsing into droplets. Just at the moment of collapse a second set of very much smaller droplets forms from the last filaments of liquid connecting the primary droplets. The phenomenon may be readily observed in a thin stream of water from a faucet and is sufficiently regular to be observed by stroboscopic light. The existence of the secondary droplets may also be demonstrated by passing such a stream through a gentle breeze which will deflect the secondary droplets so that they may be caught and studied.

The direct formation of streams of liquid thin enough to produce fine sprays is not usually practical; hence secondary actions are resorted to. Two of these methods are as follows:

1. Filaments of liquid are dragged out by the impingement of high-velocity turbulent air or steam jets on the liquid surface. In this case a primary droplet is dragged away from the surface, carrying behind it a filament which may be stretched out by the velocity and turbulence of the gas to a very fine thread before it collapses to droplets. Such action produces a wide range of particle sizes, many of them exceedingly fine.

2. The other common method of forming filaments is to spread the liquid out into a thin sheet. This sheet then draws up into a filament on its free edge, and this filament, in turn, breaks down into droplets. Such sheets or films of liquid, when projected into air at a high velocity, often exhibit another and extremely interesting phenomenon. The rapid relative motion of the film of liquid and the surrounding air sets up turbulence in the air which, reacting on the liquid film, causes it to wave or flap, just like a flag flying in the breeze. At its free edge, this flapping often becomes so violent that the film actually rolls up and joins itself into a tube which breaks away from the sheet and, being unstable, as a solid filament is, necks down and breaks into droplets. In this case, however, the droplets are hollow, having enclosed a considerable amount of air. Often these particles also enclose other smaller particles previously formed. These hollow particles are usually very thin-walled and present a large surface, which is frequently desirable.

Energy Relations. The energy required to form a liquid into droplets is composed mainly of three parts: (1) energy required to form surface against surface tension, which is simply the surface tension times the additional surface formed. In the case of water at room temperature, the net energy per unit volume required to form droplets 1μ in diameter, corresponds to a pressure on the liquid of only 0.005 lb. per sq. in. (2) Since the time during which droplet formation usually takes place is very short, often a few microseconds, the rate of deformation of the liquid is very high, and viscous forces become enormous. The energy required to produce this deformation is, therefore, appreciable although not readily computed. (3) There is energy lost due to inefficient application of energy to the fluid. When the energy is supplied directly to the liquid by a pump and released in a well-designed nozzle, this last efficiency is probably high, but, in the case of air or steam jets impinging on the liquid, the transfer of energy is relatively low. Offsetting this is the fact that very large amounts of energy can be applied in a compressible fluid such as air or steam, as compared with that stored in a liquid under pressure. The latter is simply 144 PV ft.-lb. per lb. of liquid (where P = pressure, lb. per sq. in., and V = specific volume of liquid, cu. ft. per lb.) or $144P/62.3$, or $2.3P$ for water at room temperature. The available energy of 1 lb. of saturated steam at 60 lb. per sq. in. gage expanding to atmospheric pressure is 36,300 ft.-lb. In order to concentrate this energy in a pound of liquid water, there would be required a pressure of over 15,000 lb. per sq. in. Although, in general, increasing amounts of energy applied to the atomization of a liquid tend to produce finer and finer particles, there is probably a theoretical and certainly a practical limit to which this can be carried. As the filaments of liquid become finer and finer, their rate of collapse increases rapidly owing to the high surface energy per unit volume of liquid. The rate and amount of deformation required also increase. Sauter (*Forsch. Gebiete Ingenieurw.*, 1928, No. 312) found that at high air speeds atomization in a certain type of nozzle approached asymptotically a droplet size of about 6μ.

Dispersion. Equally important with the formation of liquid droplets is their adequate dispersion, since, if allowed to come in contact, they will immediately recombine. In the case of atomization with a stream of high-velocity gas very little recombination probably occurs, since all particles are carried away in the gas stream with approximately the same velocity. In the case of pressure atomization, however, dispersion becomes important. In this case the droplets are projected into more or less quiescent air. If it is assumed that the particles are small enough to encounter resistance in accordance with Stokes's law and that they all leave the nozzle with the same velocity, they will travel out into the gas for distances which are approximately proportional to their diameters before being stopped by the friction of the air. The smaller particles will tend to collect in considerable numbers near the nozzle and to collide with the larger particles which are still moving with appreciable velocities in this region. If the droplet-size distribution from the nozzle were accurately known, it would be possible to compute the probable amount of recombination due to collisions by the methods outlined by Kleinschmidt [*Chem. & Met. Eng.*, **46**, 487–490 (1939)]. This is seldom possible, and the only practical value of such considerations is to indicate the importance of maintaining an adequate flow of gas past the nozzle to remove the droplets as formed.

SPRAY NOZZLES

By H. G. Houghton

A spray nozzle is a device for breaking up a liquid into drops. The applications of spray nozzles are numerous and varied and consequently a large number of different forms are in use. All spray nozzles may be classified under one of the following types:

1. *Pressure nozzles* in which the fluid is under pressure and is broken up by its inherent instability and its impact on the atmosphere or by its impact on another jet or a fixed plate.

2. *Rotating nozzles* in which the fluid is fed at low pressure to the center of a rapidly rotating disk or cup. Centrifugal force causes the fluid to be broken up into drops.

3. *Gas-atomizing nozzles* in which the fluid is subjected to the disrupting effect of a high-velocity jet of gas. There are several forms of each of these types in common use, and these will now be described in turn.

Pressure Nozzles

Hollow-cone Nozzles. The pressure nozzles find the widest field of application and are available in a variety of forms and sizes. The most common of these is the so-called hollow-cone nozzle. In this nozzle the fluid is fed into a whirl chamber through tangential passages or through a fixed spiral so that it acquires a rapid rotation. The orifice is placed on the axis of the whirl chamber, and the fluid exits in the form of a hollow, conical sheet which then breaks up into drops. Such nozzles are illustrated in Figs. 1, 2, and 3. Hollow-cone nozzles are made with orifices from 0.02 to 2 in. in diameter with corresponding discharge rates of from less than 1 to more than 200 gal. per min. The larger sizes are used for cooling ponds, for washing gravel and sand, aerating water, etc., and are usually operated at relatively low pressures. Smaller nozzles may be used for spray drying,

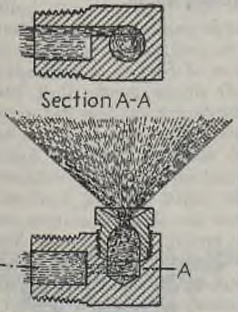


FIG. 1.—A small hollow-cone nozzle of the tangential type. (*Courtesy of The Spray Engineering Co.*)

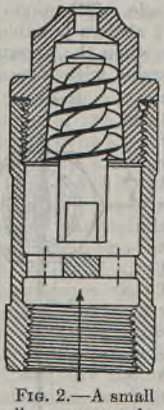


FIG. 2.—A small hollow-cone nozzle of the fixed spiral type. (*Courtesy of Schutte and Koerting Co.*)

FIG. 3.—A large hollow-cone nozzle. (*Courtesy of Schutte and Koerting Co.*)

air washers and humidifiers, oil burners, gas absorption, etc., and are usually operated at somewhat higher pressures. In common with all pressure nozzles the capacity of a given nozzle is nearly proportional to the square root of the pressure except at extremely high pressures where friction limits the discharge. Operating pressures do not usually exceed 300 lb. per sq. in. except in special cases such as milk-powdering sprays where pressures of from 1000 to 2500 lb. per sq. in. are used. For a given design of nozzle the discharge at constant pressure is approximately proportional to the area of the orifice, although the orifice does not run full. The discharge does not vary much with the viscosity of the fluid, until the viscosity is more than ten times that of water, although the drop size is considerably altered, as will be pointed out below. The included angle of the spray cone usually increases slowly with pressure to a maximum and then decreases, but it is largely determined by the proportions of the nozzle. A spiral with a short pitch produces a wide-angle spray, and conversely a large pitch spiral gives a small included angle. The angle may be from 15 to 135 deg., but it is not always possible to obtain stock nozzles of a desired angle when the pressure and discharge rate are also fixed. Nozzles with a small included angle tend to produce a solid-cone rather than a hollow-cone spray.

FIG. 4.—Solid-cone nozzle. (*Courtesy of The Spray Engineering Co.*)

Solid-cone Nozzles. The solid-cone nozzle is a modification of the hollow-cone nozzle which is used when complete coverage of a fixed area is desired. Such nozzles are used for certain washing applications, for cooling and aerating water, and for other purposes where the more uniform spatial distribution of the drops is advantageous. The construction and operation of a typical solid-cone nozzle are illustrated in Fig. 4. The nozzle is essentially a hollow-cone nozzle with the addition of an axial jet which strikes the rotating fluid just within the orifice. The break-up is largely due to this impact and the resulting turbulence. The fluid appears to leave the orifice in drop form, whereas in a hollow-cone nozzle a short conical fluid sheet which breaks

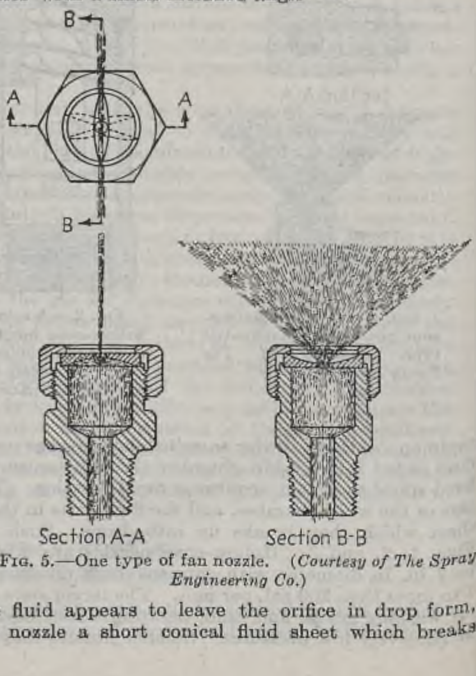


FIG. 5.—One type of fan nozzle. (*Courtesy of The Spray Engineering Co.*)

up outside the orifice is usually observed. To obtain a uniform spatial distribution, it is necessary to design the nozzle so that the proper relation exists between the amount of liquid fed to the center jet, the amount which is rotated, and the orifice size. Normally, more of the fluid is given a rotary motion than is passed through the axial jet. A separate feed line may be connected to the axial jet so that two liquids or a liquid and a gas can be intimately mixed. This is often useful for chemical applications.

The included angle of the solid-cone spray is a function of the design of the nozzle and is nearly independent of pressure. Various commercial solid-cone nozzles produce cones with included angles of from 30 to 100 deg. As indicated above, hollow-cone nozzles with small included angles (less than about 30 deg.) give a solid-cone spray without the addition of a center jet. Solid-cone nozzles are not usually available in such small sizes as are hollow-cone nozzles, but stock sizes have discharge rates from less than 1 gal. per min. to several hundred gallons per minute.

Fan Nozzles. A third form of pressure nozzle is the so-called fan nozzle. By means of milled cuts or channels on the rear face of the orifice plate, and sometimes an elongated orifice, or by means of two inclined jets, the fluid is caused to exit in the form of a flat fan-shaped fluid sheet which then breaks up into drops. Typical fan nozzles are shown in Figs. 5 and 6. Owing to surface tension, the edges of the sheet are usually bounded by solid streams or "horns," particularly in the smaller sizes, which may comprise from one-fourth to one-half of the total amount of liquid sprayed. These streams break up into larger drops than the central sheet. The horns are usually not so pronounced in the larger sizes, and for included angles of spray that are less than about 50 deg. Fan nozzles are useful when it is desired to distribute the spray along a line such as in washing, cleaning, coating, or cooling material in a continuous process. The included angle of the fan is from 10 to 130 deg. in standard nozzles, and capacities range from 0.1 to 20 gal. per min.

Impact Nozzles. Another type of nozzle which is used for certain special purposes is the impact nozzle. A solid stream of fluid under pressure is caused to strike a fixed surface or another similar stream. By a proper orientation and shape of the plate or by varying the size and direction of the two fluid streams, it is possible to obtain a hollow cone, fan- or disk-shaped fluid sheet. Such nozzles are not widely used, the two common applications being in some types of air-moistening equipment and for automatic sprinkler heads. A humidifying nozzle of this type is shown in Fig. 7.

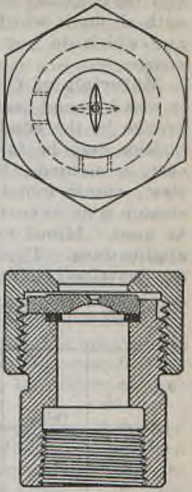


FIG. 6.—Another design of fan nozzle. (*Courtesy of Schutte and Koerting Co.*)

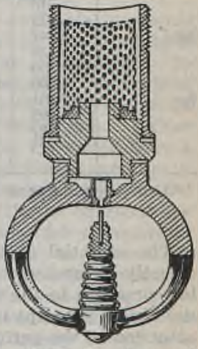


FIG. 7.—A small impact nozzle as used in a direct humidifier. (*Courtesy of Parks-Cramer Co.*)

Pressure nozzles as a class are relatively simple, small, and inexpensive, and they usually consume less power than other types. They may be used with all fluids which have a viscosity less than about 300 to 500 sec. Saybolt and which do not contain solid particles larger than the passages in the nozzle.

Materials of Construction. Pressure nozzles are commonly furnished in cast iron and cast brass or bronze in the larger sizes and in steel, brass, and bronze in the smaller sizes. When corrosion or erosion is important, the nozzles may be formed from any material which can be either machined, cast, or molded. Some of the more common special materials are stainless steel, monel metal, hard lead, ceramics, hard rubber, and glass. When erosion is an important consideration, tips of stellite or other hard alloys may be used. Monel metal nozzles are particularly useful for high-temperature applications. Typical pressure-capacity data for standard pressure nozzles are contained in Table 1.

Table 1. Discharge Rates and Included Angle of Spray of Typical Pressure Nozzles*

Nozzle type	Orifice diameter, in.	Discharge, gal. per min., and included angle of spray							
		10 lb. per sq. in.		25 lb. per sq. in.		50 lb. per sq. in.		100 lb. per sq. in.	
		Discharge	Angle, deg.	Discharge	Angle, deg.	Discharge	Angle, deg.	Discharge	Angle, deg.
Hollow cone	0.046	......	..	0.10	65	0.135	68	0.183	75
	.140	0.535	82	0.81	88	1.10	90	1.50	93
	.218	1.25	83	1.88	86	2.55	89	3.45	92
	.375	7.2	62	11.8	70	16.5	70		
Solid cone	.047	......	..	0.167	65	0.235	70	0.34	70
	.188	1.60	55	2.46	58	3.42	60	4.78	60
	.250	3.35	65	5.40	70	7.50	70	10.4	75
	.500	17.5	86	27.5	84	38.7	73		
Fan	.031	0.085	40	0.132	90	0.182	110	0.252	110
	.093	0.70	70	1.12	76	1.57	80	2.25	80
	.187	2.25	50	3.70	59	5.35	65	7.70	65
	.375	9.50	66	15.40	74	22.10	75	30.75	75

* Data furnished through the courtesy of the Spray Engineering Co.

Rotating Nozzles

The essential part of a rotating nozzle is a disk or cup which is usually directly connected to an electric motor. The fluid to be sprayed is fed under low pressure to the center of the rotating disk. Various forms of disks are used in an attempt to improve the spraying characteristics. Vanes are often attached to the periphery of the disk or mounted separately a short distance from the periphery to assist the break-up or to remove some of the larger drops. The rotary nozzle is particularly useful for spraying viscous liquids, slurries, and liquids containing solid particles which would clog other nozzles. They are also used in some air washers, in small air-moistening units, and in domestic oil burners. The spray is distributed in all directions in the plane of the disk, and this is often a disadvantage. Disk speeds depend on the application and size of the nozzle and vary from a few hundred to several thousand r.p.m. The quantity of fluid sprayed may be readily controlled over wide limits. Small units may spray only a fraction of a gallon per hour for air moistening, while some large units operate at a discharge rate of 100 gal. per min. The size of the drops produced may be varied by changing the speed of rotation and the discharge rate, high speeds and low discharge rates giving smaller drops. Rotating nozzles commonly require somewhat more power to operate than a pressure nozzle for a given application. This is probably due to friction losses between the fluid and the disk and between the fluid and the air. In addition, a pump must often be used to deliver the fluid to the disk. The rotating nozzle is relatively large and expensive and is not commonly used for purposes to which pressure nozzles are equally applicable.

Gas-atomizing Nozzles

In a gas-atomizing nozzle the liquid is broken up by impingement with a high-velocity stream of gas, usually air or steam. The fluid may be fed under pressure, low gravity head, or sucked up by the injector action of the gas stream. The contact between the fluid and the gas may take place entirely outside the nozzle or within a chamber from which the spray exits through an orifice. The shape of the cloud of spray may be controlled by the shape of the orifice in the internal mixing types and by additional gas jets in the external mixing type.

Applications. Gas-atomizing nozzles are used when very small drops are desired. They are also capable of spraying more viscous fluids than pressure nozzles. They are commonly used for spray painting, for air and material moistening, for the application of insecticides, and in oil burners. Except for the oil-burner application the discharge rate of gas-atomizing nozzles is small, seldom exceeding 10 gal. per hr. Considerably more power is required to spray at a given rate with a gas-atomizing nozzle than with a pressure nozzle because the fluid is much more finely divided.

Paint Spraying. Atomizers for paint spraying are available in a wide variety of types. External mixing is more common but internal mixing is also used. It is often desirable to have a fan-shaped spray for painting. In external mixing nozzles this is accomplished by two external air jets which impinge on the spray from opposite sides and flatten it out. The degree of flattening may be varied by varying the pressure of the forming jets. Valves are often provided for this purpose, so that the spray may be varied from a fan shape to a conical shape. The amount of paint supplied is commonly adjusted by varying the travel of the control valve. Spray guns are operated at air pressures from 20 to 80 lb. per sq. in. but 40 to 60 lb. per sq. in. is the usual range. The more viscous the paint the more pressure is required. Production-type spray guns usually require from 4 to 8 cu. ft. of free air per minute at a pressure of 40 lb. per sq. in. The volume of fluid delivered is a function of the air pressure and of the fluid pressure. In some spray guns the paint is carried in a small tank attached to the gun and is fed either by pressure or suction. For production work the paint is usually stored in a large separate tank under either gravity or pressure head, and is supplied to the guns through rubber hoses. Further information on paint spraying is given on p. 1994.

Humidification. Gas-atomizing nozzles are often used for direct humidification in plants where controlled humidity is required, such as textile and paper mills. A typical unit is shown in Fig. 8. A number of atomizers are mounted on the supply pipes which are attached to the ceiling. The nozzles are arranged to spray horizontally and are adjusted so that the spray will be completely evaporated before reaching the floor or machinery beneath them. Humidifying nozzles spray from 1 to 10 gal. per hr. and require from 40 to 80 cu. ft. of free air per gallon of water. The air pressure is usually about 30 lb. per sq. in., and the water may be delivered by suction from a level slightly below the atomizer or under a pressure of from 10 to 30 lb. per sq. in.

Oil Burners. Although most small oil burners use pressure nozzles, gas-atomizing nozzles are commonly used in the larger installations, particularly when the heavier grades of oil are burned. Preheaters are often used to reduce the viscosity of the oil before atomization. Although air may be employed as the atomizing agent, steam is more often used. The air or steam pressure may be from 20 to 100 lb. per sq. in., with 60 lb. per sq. in. being a typical value. The oil feed may be by gravity or suction, but in the larger installations the oil is usually delivered under a pressure of from 10 to 60 lb. per sq. in. A typical oil-burning nozzle requires from 0.10 to 0.15 lb. of steam per pound of fuel. A steam-atomizing oil-burner nozzle is shown in Fig. 13, p. 2354.

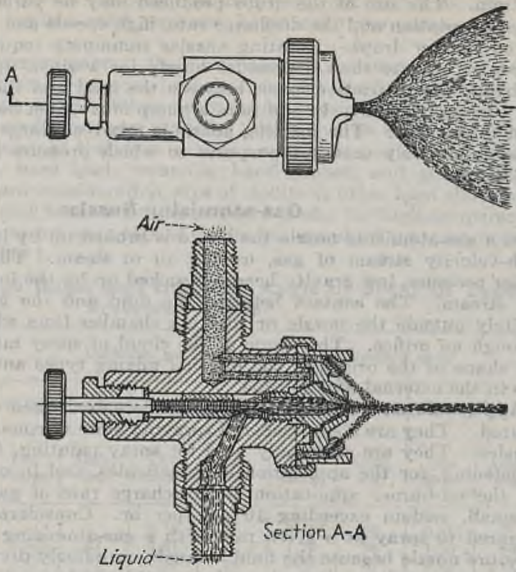


FIG. 8.—A small gas-atomizing nozzle as used for direct humidification. (*Courtesy of The Spray Engineering Co.*)

Spray Drying. Gas-atomizing nozzles are also often used for spray drying of soap and other products. Steam is often used as the atomizing agent to assist in the drying. Requirements in this field are so varied that no typical data can be given.

Drop Size

For many applications of spray nozzles the size of the drops produced is of considerable importance. When a liquid is disrupted, it first breaks into small filaments which are detached from the mass. The filaments contract due to surface tension, and an oscillation is set up which results in the formation of one large drop and one or more smaller drops from each filament. Since the filaments are not all the same size, a considerable range of drop sizes invariably results. The greater the disrupting force, the smaller the filaments and hence the smaller the drops. The filaments are most likely to be more nearly of the same size if the sheet is thin and of uniform velocity and thickness in the region of break-up. In a practical sense this means that small nozzles with smooth passages and orifices will produce the narrowest range of drop sizes. Also the higher the pressure the smaller will be the drops, other things being equal. It is found that for pressure nozzles the drop size is approximately inversely proportional to the square root of the pressure.

The viscosity, surface tension, and density of the liquid also affect the drop size. In the case of pressure nozzles it has been found that the drop size increases nearly as the square root of the viscosity. It is also known that the drop size is inversely proportional to the surface tension and to the density, but the exact laws of variation have not been established.

Pressure nozzles of like capacity give a similar distribution of drop sizes when operated at a given pressure. The hollow-cone nozzles usually yield a somewhat smaller range of drop sizes than the solid-cone nozzles. The central sheet of small fan nozzles is also particularly good in this respect, but the "horns" at the edge of the sheet break up into much larger drops so that the advantage is largely lost. For applications which require the smallest possible range of drop sizes a large number of small nozzles should be used in preference to a few nozzles of large capacity. If a maximum number of small drops is required, nozzles of the smallest size practicable should be used and operated at the highest possible pressure. (Pressures in excess of about 300 lb. per sq. in. will have no useful effect except in the case of special high-pressure nozzles.)

Table 2. The Drop-size Distributions Produced by Three Hollow-cone Nozzles of the Same Design

Nominal drop diameter, μ	The number of drops in each size group					
	0.063-in. orifice diameter			0.086-in. orifice diameter		0.128-in. orifice diameter 200 lb. per sq. in.
	50 lb. per sq. in.	100 lb. per sq. in.	200 lb. per sq. in.	100 lb. per sq. in.	200 lb. per sq. in.	
10	375	800	1700	100	300	100
25	200	280	580	60	150	50
50	160	180	260	41	100	45
100	50	60	70	26	34	27
150	27	31	35	14	18	15
200	19	23	27	9	12	11
300	8	9	11	5	8	6
400	2	4	4	4	7	3
500	1	1	...	2	1	2
600	1	...	...	1	...	...

NOTE: 1 μ (micron) = 10^{-4} cm. = 0.0000394 in. The nominal diameter is the mid-diameter of a drop group which includes a finite range of sizes. The "25" group includes drops from 17.5 to 37.5 μ, the "50" group contains drops from 37.5 to 75 μ, etc. The number of drops has been adjusted in each case so that the total amount of fluid sprayed is the same for each size distribution.

Because of the wide range of drop sizes formed by a spray nozzle, it is difficult to define an average drop size which will be significant for all purposes. It is usually better to have detailed information on the frequency distribution of the drop sizes. It is not practicable to present complete data of this sort here because of the large number of variables involved. However, a few typical drop-size distributions which illustrate the effect of pressure variations and of the nozzle size are given in Table 2. These figures may be taken as a fair example of the performance of pressure nozzles. If liquids of different physical properties are sprayed, the same size distributions will be obtained but at different pressures. For example, if a fluid has the same density and surface tension as water but twice the viscosity, the pressure would have to be about doubled in order to obtain the same size distribution.

From the standpoint of drop size, the only difference between a rotating nozzle and a pressure nozzle is that in the former the liquid is formed into a thin sheet of suitable velocity by centrifugal force instead of by direct fluid pressure. As a result, the drop-size distribution of a rotating nozzle is quite similar to that of a pressure nozzle. Because there are no small passages in a rotating nozzle, the flow is maintained at much higher viscosities than in a pressure nozzle. The effect of variations in viscosity, density, and surface tension on the drop size has not been investigated, and it is probable that somewhat different laws apply. It is undoubtedly true, however, that high rotational speed, low viscosity, and a small discharge will tend to produce small drops and a minimum range of drop size.

Table 3. The Drop-size Distribution of a Small Atomizing Nozzle

Drop diam., μ	Number of drops	Drop diam., μ	Number of drops
2	390,000	35	1,730
5	340,000	40	1,080
10	165,000	45	650
15	40,200	50	430
20	11,680	60	350
25	4,970	70	220
30	2,160		

NOTE: The fluid pressure and the gas pressure were each 15 lb. per sq. in. The total quantity of fluid represented by this size distribution is the same as that in Table 2, so that the numbers of drops are directly comparable.

As ordinarily operated, gas-atomizing nozzles produce much smaller drops than pressure or rotating nozzles. Although the smaller atomizing nozzles tend to give a somewhat narrower range of drop sizes than the larger nozzles, the size factor is not so important as in the case of the pressure nozzles. The determining factor is the relation between the quantities of gas and of fluid. When insufficient gas is used, large drops are formed which are readily visible since they are projected well beyond the cloud of small drops. The quantities of liquid and of gas should always be adjusted so that no such large drops are in evidence. Assuming that such proper operation is maintained, the drop size can be controlled by varying the gas pressure, higher pressures yielding smaller drops. The fluid pressure has little effect on the drop size and for the most part only determines the quantity of fluid delivered, which is also a function of the gas pressure. As in the case of the other types of nozzles, the physical properties of the fluid affect the drop size, but the quantitative relations are not known. It may probably be assumed that the drop size increases with viscosity and decreases with surface tension and possibly with density. For purposes of comparison with Table 2 a drop-size distribution of a small air-atomizing nozzle is given in Table 3.

Table 4. Common Applications for Spray Nozzles

First number*: type most used.
Second number: type frequently used.
Third number: type sometimes used.

Types

1. Solid-cone wide-angle spray.
2. Hollow-cone wide-angle spray.
3. Narrow-angle spray.
4. Pressure atomizing spray.
5. Tangential spray.
6. Flat spray.
7. Deflector or impact spray.
8. Air- or gas-atomizing spray.
9. Rotating-disk spray.

Pressure Nozzles

Cooling circulating water for condenser (5, 1, 6)
Spray type condensers
Aerating and purifying water supplies (5, 1, 6)
Scrubbing and washing gases (1, 3, 9)
Humidification and dehumidification (4, 8, 3)
Spray refrigeration (5, 1)
Gas absorption and adsorption (1, 3, 5)
Spray drying (4, 8)
Chemical processes where a large free surface is required (1, 4, 8)
Distributing oil over the fuel bed in gas machines (1)
Enriching gas with a liquid distillate (1, 4)
Oil burners (4, 8, 9)
Desuperheaters (4)
Washing or coating materials in process (4, 2, 8)
Washing liquids (1, 4)
Washing automobiles, railway coaches, etc. (6, 3)
Washing coal, sand, gravel, etc. (2, 6)
Beating down foam (1, 3, 6)
Cooling mill rolls (1, 4, 6)
Descaling hot billets (3, 6)
Quenching coke and pig iron (5, 1)
Settling dust (4, 1)
Applying insecticides, weed killers, etc. (1, 7, 8)
Applying asphalt to highways (1, 7, 6)
Fire protection (7)
Ornamental sprays

Rotating Nozzles

Spraying viscous liquids and slurries (7, 9, 8)
Oil burners (4, 8, 9)
Small air moisteners (8)
Spray drying (4, 8)
Air washing (4, 1, 9)

Gas-atomizing Nozzles

Spray painting (8)
Oil burners (4, 8, 9)
Spray drying (4, 8)
Air moistening (8, 4)
Moistening materials with water or other fluids (8, 4)
Spraying small quantities of insecticides, etc. (8, 4)
Metal coating (8)
Applying cements, refractories, etc. (8, 7)

* Classification kindly supplied by S. G. Ketterer.

SPRAY PAINTING

By S. L. Godshalk

One of the common uses of spray nozzles is for the application of paint, varnishes, and enamels. The advantages of this type of application over brushing are as follows: (1) greater speed of application; (2) more uniformity of film thickness; (3) better control of film thickness; and (4) faster drying finishes may be used. The disadvantages are as follows: (1) larger investment in equipment, such as air compressors, spray booths, etc., and (2) greater loss of material due to overspray, etc. However, the disadvantages are minor in comparison with the increased speed of application and the choice of finishing materials available for spray use. For example, the introduction of fast-drying nitrocellulose finishes in the automotive industry has cut the time of finishing from days to hours. The application of spray nozzles in the form of spray guns played an important part in this reduction.

Spray application can be employed in almost any industry where finishes are involved. Likewise, most types of finishes can be used by spray application with modifications in nozzles for varied operating conditions.

Forms of Spray Equipment

Spray guns are available in a variety of types. They may be external or internal mixing; suction, gravity, or pressure feed; manually or automatically controlled; low or high pressure. In addition spray guns have adjustments for size and pattern of spray as well as for amount of delivery. A typical spray gun is shown by Fig. 9.

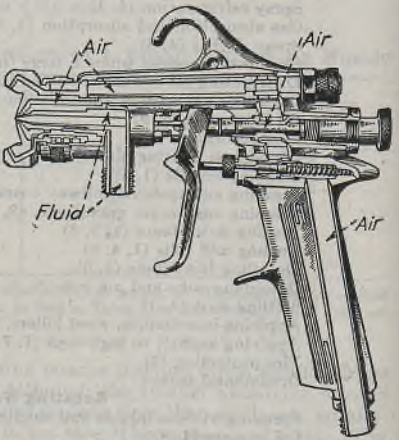


FIG. 9.—Typical spray gun. (*Courtesy of DeVilbiss Co.*)

Selection of Spray and Auxiliary Equipment

Because of the wide variety of spray equipment available and the numerous types of finishes offered by paint manufacturers, it is important that these factors be taken into consideration when new equipment is installed. Careful thought concerning present as well as future needs given to the selection of this equipment should result in a flexible paint shop. It frequently happens that spray installations have been made to take care of present needs, but with the expansion of business this equipment cannot adequately take care of increased production; hence the paint shop becomes the bottleneck of the operation.

The following generalizations may be of some help in the selection of spray equipment. Improvements are constantly being made in both finishes and spray equipment, and the advice and suggestions of the manufacturers should be of considerable value.

1. The atomizing phase of spray guns is subject to changes in design for most efficient results, depending upon the type of finish being used. In

other words, nitrocellulose lacquers, synthetic-resin finishes, or varnish products give best results when used with the recommended equipment. The spray-gun manufacturer and the supplier of finish should be consulted for recommendations.

2. The choice of suction, gravity, or pressure feed depends largely upon the amount of paint used and the number of colors involved. Where the consumption of finishes is very small or where occasional spraying is necessary; the suction type, such as a cup gun, is generally satisfactory. Where larger quantities of finish are consumed, gravity or pressure feed is desirable. Of the two, pressure feed is the more reliable since the pressure is always uniform, whereas in gravity feed the pressure will vary with the fullness of the container unless a constant-level tank is employed.

3. Where pressure feed is used, agitator pressure tanks in or adjacent to the spray booths are satisfactory, particularly where numerous colors and only moderate volumes are involved. In cases where large volumes in relatively few colors are involved, a mixing tank in the paint room with the material piped to the spray gun is most satisfactory. One advantage of both of these installations is that one man can be designated to do thinning, to control the viscosity, and to have the various colors and types of finishes available for the spray booth as the production schedules demand.

4. Where large quantities of finish are used, it is sometimes desirable to equip guns with a locking device to control size or pattern of spray and air pressure. This is important where different spray operators finish parts of an assembled object or where maximum efficiency of operation from the viewpoint of finishes is concerned.

5. Air compressors should, of course, be of sufficient capacity to take care of all spray guns, line losses, air dusting if any, and air agitators, plus a reserve to take care of additional requirements for the immediate future. As auxiliary equipment, water and oil filters should be provided to eliminate contamination.

6. The spray booth is also an important piece of equipment in spray painting, since there is spray dust which must be eliminated to improve the operator's safety, efficiency, and better workmanship and because the dust collects on adjacent buildings and grounds if allowed to escape in the air. In addition, there is a probability of economy if spray dust is reclaimed. Here again the initial cost must be weighed against local conditions and the amount of paint consumed. For much industrial work the conventional spray booth with air exhaust is satisfactory. In certain areas exhausts of spray dust in the air may come under the heading of "public nuisance" because of the spray dust settling on houses, etc. In these cases a "water washed" spray booth, *i.e.*, where the spray dust is collected in the form of a sludge by passing through a curtain of water, is recommended. This type is also used where it is considered desirable to collect sludge for reclamation.

In plants where dust and dirt in the atmosphere may be a problem or freedom from dirt in the paint film is a factor, it is desirable to maintain a positive pressure in the spray area. For this purpose filtered air is supplied.

The air velocity in spray booths should be at least 100 lin. ft. per min.

Spraying Difficulties and Suggested Corrections

The following are a few of the difficulties which may be encountered in spray application and some generalizations for their elimination:

Difficulty	Causes and Correction
1. Excessive orange peel	*a.* Spray viscosity too high
	b. Wrong spray gun for material being sprayed
	c. Wrong thinner being used
	d. Film thickness too low
	e. Air pressure too high or too low
2. Excessive flow and sags	*a.* Over thinned
	b. Wrong thinner being used
	c. Film thickness too high
	d. Wrong spray gun being used
3. Craters or bubbles in film	Drain oil and water separator and see if functioning properly
4. Poor atomization	*a.* Insufficient reduction
	b. Air pressure too low
5. Overspray dust (pebbly condition on surface)	*a.* Overspray dust settling on finish due to poor ventilation
	b. Improper handling of parts on conveyor chain
6. Low gloss or haze	*a.* Poor ventilation during drying or baking
	b. Thinner evaporates too fast, causing moisture precipitation on surfaces during humid weather particularly with nitrocellulose products. Use a more slowly evaporating thinner

GRANULATION

By Lawrence H. Bailey

The purposes of granulating may be as follows: (1) to prepare the material for further processing, such as briquetting or tableting; (2) to debulk or densify the material for more convenient storage or shipment; (3) to prevent dust losses in subsequent furnace treatment or from shipping containers; (4) to reduce dust nuisance in the case of irritating or otherwise obnoxious material; (5) to prepare the material in a condition which will not form large cakes or lumps, especially if the material is hygroscopic.

The granular form of materials is often produced by building up the granules from fine powder, instead of by breaking down larger aggregates by means of grinders, crushers, or other machines. Often tablets or pellets such as are now increasingly employed as catalysts or catalyst carriers can be formed much more readily from granules than from powders; indeed, sometimes they cannot be formed from the powdered form at all.

There are several general methods of producing granules from powder: (1) moistening with water, solvents, or binding solutions with subsequent screening, grinding, or rotary-drying operations, either in the damp condition or in the dry condition, or both; (2) briquetting of the fine powder, followed by grinding or screening; (3) fusing to produce the necessary adhesion, giving granules directly in some cases, or in other cases, producing a mass, sheet, or film which can be broken down to the granular condition; (4) preparing the powder in the form of a suspension in liquid, where it is sometimes possible and desirable to reduce it in one operation to the granule stage by spray drying.

Granulation with a Binding Agent

Binding Agents. In the first of the above methods, the required adhesion is obtained from, or brought into effect by, the liquid used for granulating. With water-soluble materials or mixtures containing such material, the addition of water alone may be sufficient, as in the case of sugars, salts, extracts, etc. If water will not develop the required adhesiveness, a solvent for the material, or one or more ingredients of the mixture, may be used. If none of the ingredients develop adhesiveness by the addition of water or solvent, a binder is introduced by dissolving it in the water or solvent used for wetting up the powder. In some cases, a dried binder in powdered form is mixed with the powder being treated, and the mixture is then moistened to activate the binder. In general, the dry binder subsequently moistened is not so effective as a solution of the same binder, but in some cases this method gives the desired results and may effect some economies in processing.

In some cases the material being granulated may develop too much adhesiveness. It may not mix smoothly with the granulation solution, or may become excessively sticky, producing unmanageable masses. In such cases the wetting power of the solvent may be reduced by mixing it with other solvents; for example, water may be mixed with alcohol when handling sticky water-soluble materials. In using alcohol-soluble materials, the alcohol may be diluted with water or other solvents to reduce its solvent power.

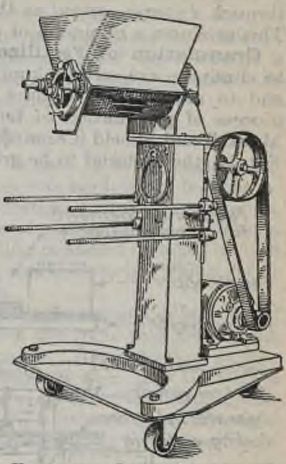


Fig. 10.—Oscillating granulator, assembly. (*Courtesy of F. J. Stokes Machine Co.*)

Some of the materials which have been used in aqueous binding solution for granulation are sugars, glue, gelatine, dextrins, gums, starch, flour, molasses, and sulfite waste liquor; and with other solvents, shellac, waxes, lacquers, etc. Among binders introduced dry and later activated are pulverized sugar, spray-dried glucose, malt extract, or almost any of the above-mentioned water-soluble binders. As stated before, however, a larger percentage of the binder is required than when it is introduced in solution.

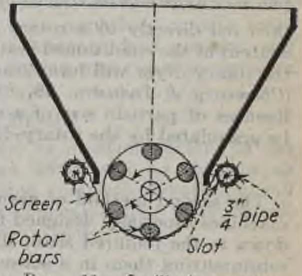


Fig. 11.—Oscillating granulator, schematic section. (*Courtesy of F. J. Stokes Machine Co.*)

Granulating Stage. After the damp mixture is made, it must be processed to produce the required granular condition. The mixture is usually dried down to a proper moisture content and then crushed or ground in equipment which will give the desired screen analysis, using for this purpose a suitable mill, grinder, or granulator. One usual requirement in the finished granulation is a minimum of fine dust. Also the machine must not develop enough heat to make the material gum up and necessitate a stoppage of operation. A granulator consisting of a set of bars arranged in cylindrical form and oscillating over a screen of suitable mesh usually produces the desired results in the case of fairly coarse granules, say 20 mesh or over. Such an oscillating granulation is shown by Figs. 10 and 11. With smaller mesh size this type of machine is rather slow, and other types of grinders are more effective, such as plate or cone mills, crushing rolls, etc.

One very effective method of handling the damp material is to granulate through a coarse screen, as above, and effect further reduction after drying. This produces a minimum of fines.

Granulation of Fertilizers. Many fertilizer products are granulated to eliminate excessive caking and lumping of the material before it is used and to provide the product in a convenient form. A typical large-scale process of granulation of fertilizers is the Davidson process described by Mackall and Shoeld [*Chem. & Met. Eng.*, **47**, 102–105 (1940)]. As shown by Fig. 12, the material to be granulated is first conditioned by moistening and then fed directly to a rotary drier. By carefully controlling the moisture content of the conditioned material, which requires constant visual attention, the rotary dryer will form granules of the desired form. Hardesty and Ross [*Chemistry & Industry*, **58**, 885–886 (1939)] found that the plasticity and fineness of particle size of a material determine the ease with which it can be granulated by the rotary-drying method.

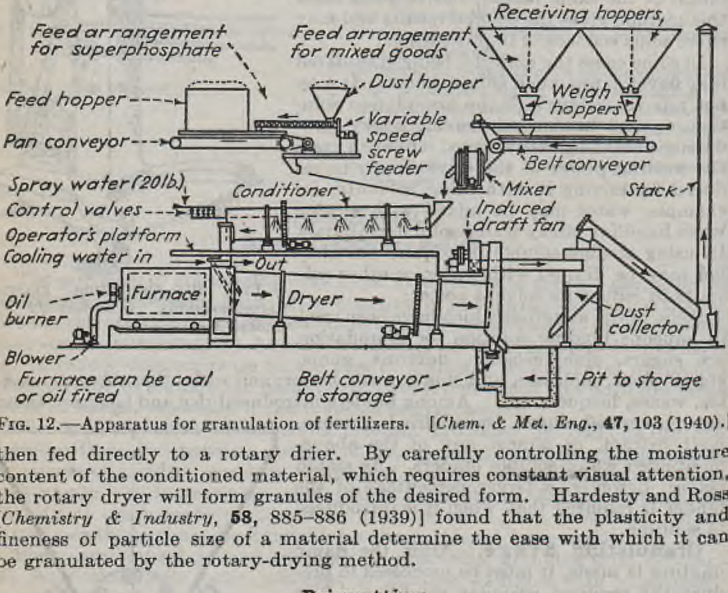


FIG. 12.—Apparatus for granulation of fertilizers. [*Chem. & Met. Eng.*, **47**, 103 (1940).]

Briquetting

The second method of granulating is to briquette the fine powder, using machines especially designed for this work, and then to break the briquettes down to the required sizes, sifting out the fines if they are undesirable and rebriquetting them in a subsequent operation. This method was originally developed in the pharmaceutical industry when working with material such as aspirin or other mixtures which react when wet and which could, therefore, not be granulated by wet methods. It is now used in producing granular plastic molding powders and various other granular materials. It is also used for reducing dust nuisance with materials such as sodium or calcium hypochlorite.

When the granular material produced is to be briquetted, it is sometimes important to have a considerable percentage of smaller granules and some fine material, as material of the properly diversified screen sizes will make a stronger briquette than one made from more uniform granules. In briquetting it may be necessary to add a lubricant so that the compressed material will not stick to the punch faces or to the die but rather will be assisted in ejection from the die. Commonly used lubricants are oils and oil mixtures, powdered waxy materials, powdered soap, talcum powder, metallic stearates, and boric acid. In general, only a very small percentage of the lubricant is required, especially if it is an extremely fine powder. In nearly all cases, the lubricant is added to the otherwise finished granulation.

Granulation by Fusion

The third, or fusion, method of granulation, is used in a few highly specialized industries. Mixtures containing pitch or wax binders can be agglomerated by means of heating and then can be reduced to granules after cooling. A familiar example is found in the manufacture of plastic molding powders. A mixture of the resin with wood flour or other filler is passed through friction rolls to develop adhesiveness and produce a sheet material which, after cooling, is reduced to the familiar granular form.

The fusion method is used for producing granular effervescent salts. When a mixture of bicarbonate and citric acid is heated, the citric acid melts in its crystal water and causes adherence of the particles. Rapid cooling arrests reaction or effervescence. The resulting material is then put through a screen and dried to remove the remainder of the water of crystallization, producing a stable material which will effervesce readily and smoothly when placed in water. Several other materials containing fusible crystal salts can be handled in a similar manner.

Waxes and fusible organic or inorganic material of various kinds can be granulated, flaked, or chipped by chilling the melted material in the form of a film on the surface of a cooled revolving drum and removing by means of a knife. The character of the flake or chip can be varied by changing the temperature of the liquid material or of the cooling drum or by the method of sharpening the knife. In some cases the material can be taken off in sheet form and then reduced to flake form. A few of the materials handled by these methods are as follows: naphthalene, beta naphthol, caustic soda, potash, cyanides, carnauba wax, sulfonated detergents, lead acetate, and trisodium phosphate. Further description of flaking operations is given on this page.

Granulation by Spray Drying

The fourth, or spray-drying method, is used to a certain extent in the ceramic industry where the raw materials are wet-ground as an aqueous suspension. It has the advantage of permitting complete dewatering and granulation in one operation but is probably applicable only to industries where grinding or mixing as an aqueous suspension is desirable.

FLAKING

By D. J. Van Marle

Flaking, as described in this section, is the continuous process of converting certain molten materials into flake form by applying the material in a thin layer to a revolving drum, on which the material cools, solidifies, and is removed in flake form by a stationary knife. This method conserves time and working space and eliminates the necessity of crushing the finished product before shipment or use. Being a continuous process, it economically replaces solidification in bulk and offers increased production at lower cost.

Drum Flakers. Usually the flaking operation is performed on the surface of a revolving drum, similar to that shown by Fig. 13, that is cooled by water, brine, or direct expansion of a refrigerant. A flaker or cooling drum is very simple in construction, consisting of a drum with hollow trunnions, mounted in bearings. The drum dips in a shallow, heated pan filled with the liquid. A knife or doctor held firmly against the drum removes the product from the surface in solid form. In most cases the layer of solid material at this point is sufficiently brittle to break into flakes or grains. Since the reduction to flake size is obtained by the chipping action of the knife, size of the flakes is not very regular, varying with the properties of the product and the method of operation.

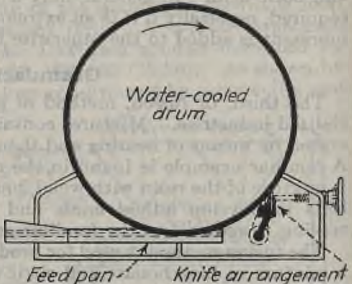


FIG. 13.—Typical drum flaker.

For products which do not adhere readily to a cold drum or which drop off too quickly, a double-drum flaker is preferred to a single-drum machine. A double-drum flaker, shown by Fig. 14, consists of two drums placed close together and revolved in opposite directions. The liquid is fed between the drums, being prevented from flowing out at the end of the drums by dam plates or endboards held against the heads. With the drums turning toward each other at the center, a layer is formed on the drum surfaces, the thickness of which is determined by the clearance between the drums. This thickness is limited by the maximum clearance beyond which the liquid begins to run through. There is danger of incrustation of the endboards or of solidification of the liquid in the valley between the drums. If the drums revolve away from each other at the center a heavier layer can be formed.

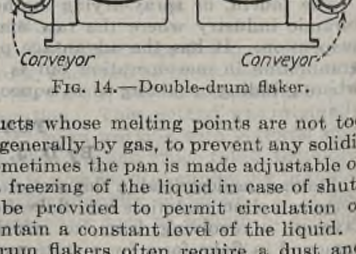


FIG. 14.—Double-drum flaker.

Drums are usually made of cast iron. Bronze, nickel, stainless steel or chromium plate are also used. The drums are provided with an internal baffle which leaves an annular space at the periphery of the drum through which the cooling medium flows, entering through one trunnion and leaving through the other. The pan can be made of any suitable metal and can be steam jacketed or provided with a heating coil for products whose melting points are not too high. Otherwise it is heated directly, generally by gas, to prevent any solidification of the product in the pan. Sometimes the pan is made adjustable or able to be dumped quickly to prevent freezing of the liquid in case of shutdowns. An overflow pipe also can be provided to permit circulation of the liquid through the pan and to maintain a constant level of the liquid.

When used on organic materials, drum flakers often require a dust and fume-tight ventilated enclosure, which limits accessibility and observation of liquid level in the pans unless a sight gage is installed.

Revolving tables and traveling metallic belts also have been used for flaking. Water is sprayed against the bottom to carry off the heat. These designs lend themselves better to positive control of the flake thickness and to the production of very thick flakes.

Applications of Drum Flakers. Operation and construction of the flaking machine are so simple that it can be applied to practically all chemical products which have a definite but not too low a melting point, both inorganic and organic, including salts which melt and dissolve completely in their water of crystallization. Such products include caustic soda with a melting point of 318°C., caustic potash, sulfur, 70 to 75 per cent calcium chloride, magnesium chloride, 60 per cent sodium sulfide, sodium acetate (U.S. Patent 1,911,479), trisodium phosphate, sulfur, beta naphthol (U.S. Patent 1,594,390), benzidine (U.S. Patent, 1,591,688), naphthylamines, phthalic anhydride, and paradichlorbenzene with a melting point of only 53°C. Flaked ice is made by the same operation on a specially designed flaking machine, described on p. 2622. Calcium chloride flakes are liable to cake to such an extent that it has been found desirable to subject them to surface dehydration (U.S. Patent 1,527,121). The flaking operation can also be applied to products of more indefinite composition such as asphalt, pitch, paraffin, various waxes, and stearic acid. Although not strictly a flaking operation, the cooling of lard on a drum surface cooled by artificial refrigeration may be included [*Chem. & Met. Eng.*, **31**, 699 (1924)].

Operating Factors. Depending on the individual properties of the material and conditions under which the flaking operation is carried out, the cooled product will be in the form of flakes, generally of irregular shape, or more or less granular. Some products adhere more strongly to the drum surface than others and require considerable knife tension for their removal. The degree of adherence also varies with the metal of which the drum is made, as well as with the polished condition of the drum surface, affecting thereby the size of the flakes to some extent as well as the capacity. For one product this was increased from 45 to 55 lb. per hr. per sq. ft. by changing from a cast-iron to a bronze drum. Wetting of the drum surface may assist in removing the solid product as has been proposed for niter cake (U.S. Patent 1,312,430).

Flakes become thicker if depth of the liquid is increased, or if drum speed, cooling water temperature or liquid temperature are reduced. Best results are obtained when the temperature of the liquid in the pan is close to the solidifying point in order to make the material solidify quickly when it comes in contact with the drum.

Drum Cooling. Generally, it is not necessary to maintain a uniform temperature of the cooling water because a large enough temperature difference exists at the point where the flakes are removed from the drum surface. The water, in most cases, flows parallel to the axis of the drum, while the temperature of the material varies along the circumference. Therefore only a small temperature increase of the water can be allowed and a liberal flow of water must be maintained. Occasionally the water temperature must be more closely controlled, better results being obtained with water which is not too cold. In that case the water can be circulated through an overflow tank and only enough cold water admitted to maintain the desired temperature. Such a procedure is recommended if the material, when cooled too much, adheres too strongly to the drum surface, causing excessive knife wear. If cold water of uniform temperature is required, or water economy becomes important, the necessary flow may be provided by recirculation of the water through a tubular or vacuum cooling system. Low temperature is required only for products of very low melting point. Ordinary seasonal fluctuation in cooling water temperature, in most cases, is not objectionable. Such fluctuation does have its effect on capacity, sometimes causing a difference of about 20 per cent between summer and winter production. If necessary, drum speed can be adjusted to compensate for the variation in water temperature. Insufficient flow of water, leaving the drum at too high a temperature, affects the size of the flakes. In case of a hydrated inorganic salt, with a melting point of 77°C., flakes were much larger when the cooling water left the drum at 20°C. than when it left at 40°C. Flakes were slightly thinner at higher water temperature.

Increased level of the liquid increases the length of travel of the drum through the liquid, thereby giving an opportunity for a larger amount of the liquid to adhere to the drum, increasing the flake thickness and capacity.

Drum Capacity and Flake Thickness. Unit capacity of flakers varies a great deal. In view of the moderate values of latent heat of fusion compared to latent heat of evaporation and of specific heat in the solid state, the amount of heat to be transferred through the drum surface is not very great. Consequently, capacity is high, varying from as low as 10 lb. per hr. per sq. ft. of drum surface to as high as 150 lb. Drum speed has a greater effect on flake thickness than on capacity for the simple reason that greater speed reduces the time of contact between a point on the drum surface and liquid. This results in a decrease in the amount of product adhering to the surface at any point, making a thinner flake. At the same time hourly capacity increases. Speed of the drum therefore may be regulated to produce a flake of desired thickness or to obtain maximum capacity. Depending on the product, drum speed varies from 1 to 20 r.p.m. With one product a capacity of 10 lb. per hr. per sq. ft. was obtained at 0.2 r.p.m. producing flakes ⅛ in. thick. At 1 r.p.m. the thickness was reduced to about $\frac{1}{64}$ in., but capacity was increased to 14 lb. per hr. per sq. ft. Another product at 1 r.p.m. formed a thick sheet at a capacity of 7.5 lb. per hr. per sq. ft., and at 4 r.p.m. flakes were produced while capacity was increased to 12.5 lb. per sq. ft.

The following table illustrates the influence of these factors in flaking an organic chemical.

Drum speed, r.p.m.	Product in, °F.	Product out, °F.	Water in, °F.	Water out, °F.	Water flow, gal. per min.	Capacity, lb. per hr.	Flake thickness, in.
10½	291	95	40	52	12	568	0.015
10½	293	113	40	70	4.1	550	.014
10½	293	126	78	95	7.8	503	.013
5½	298	104	43	52	9.6	440	.017
5½	302	113	60	69	9.3	341	.015
5½	302	140	80	90	8.2	342	Rough

Temperature of the flakes often is fairly high, 40° to 80°C. It varies with the drum speed, increasing in one case from 55°C. at 7 r.p.m. to 70°C. at 9 r.p.m.

Heat-transfer Rates. In regard to heat transfer, it is practically impossible to establish a definite heat-transfer coefficient on account of the irregular variation of temperature around the circumference of the drums and the increase of temperature of the cooling water parallel to the drum axis. Even if an average water temperature is taken, no average temperature difference can be calculated. Three definite stages occur in the conversion of the liquid into the solid state. First the liquid is cooled to the solidification point. In the next stage solidification takes place at constant temperature with absorption of the latent heat of fusion by the cooling water. Finally the solid product is cooled. Under actual conditions these stages possibly overlap, the layer next to the drum surface being a stage ahead of the outside layer. Heat transfer, for this reason, can be expressed best in B.t.u. per hour per square foot of drum surface. It is easily determined by measuring the flow and the temperature increase of the water. On this basis heat transfer was found to be about 2500 B.t.u. per hr. per sq. ft. for a wax with a melting point of 80°C., fed at 90°C., and cooled to 45°C. For a hydrated salt melting at 77°C., fed at 95°C., and cooled to 55°C., heat transfer was 5400 B.t.u. per hr. per sq. ft., and for an organic chemical with a melting point of 130°C., fed at 170°C., cooled to 65°C., heat transfer was about 6500 B.t.u. per hr. per sq. ft. Heat transfer rates of 30,000 to 40,000 B.t.u. per hr. per sq. ft. may be obtained with products of high melting point such as caustic soda.

Costs. Owing to the high capacity of drum flakers, flaking cost per pound of product is low, consisting of a small amount of heat to keep the product in the pan in a liquid condition; power to revolve the drum amounting to 0.1 to 0.15 h.p. per sq. ft. drum surface; power to pump the liquid to the pan if necessary and to pump the water to carry off the heat liberated. Operation is practically automatic as far as the flaker itself is concerned. Labor is required only to package the flakes. Cost of the equipment is very reasonable in view of the high production obtained. In cast-iron construction this cost varies from $40 to $50 per sq. ft. drum surface in accordance with the size of the machine.

Auxiliary Equipment. Flaking apparatus is practically self-contained. Auxiliary equipment is of standard design. If gravity flow of the liquid to the feed pan is not possible, a pump must be provided for the transfer of the liquid. Submerged centrifugal pumps give satisfactory service for the purpose. In case a constant level is maintained in the feed pan by means of an overflow pipe, a circulating pump is provided. Preferably the flaker is placed at a higher level than the liquid storage tank, liquid being pumped into the feed pan and the overflow returned to the storage tank by gravity.

On the discharge side of the flaker, automatic-packing and weighing machinery can be employed to advantage. A breaker attachment can be made part of the flaker if it is necessary to reduce the size of the flakes as they are removed from the drum surface. Where artificial refrigeration is needed, standard refrigerating practice can be applied.

Many flakers are fitted with screw conveyors for transporting flakes to the delivery point. These conveyors may help to break up overly large flakes, such as 3 by 4 in., if they occur, but also may cause undesirable fines.

PRODUCTION AND APPLICATION OF METAL POWDERS

By A. P. Colburn

REFERENCES: Jones, "Principles of Powder Metallurgy," Longmans, New York, 1937. Allen, *Steel*, **104**, 43–54 (1939). Comstock, *Iron Age*, **143**, 40–41 (1939); *Metal Progress*, **35**, 343, 347, 465–467, 576–581 (1939). Delmonte, *Modern Plastics*, **16**, 49–50 (1939). Hardy, *Eng. Mining J.*, **140**, 85–86 (1939); *Metal Progress*, **35**, 171–172 (1939); **36**, 57–59 (1939). Schlecht and Trageser, *Chem. Fabrik*, **12**, 243–244 (1939). Also editorial report in *Metal Progress*, **33**, 263–265 (1938).

A development of increasing importance is the production of metal powders for utilization in the manufacture of metal articles. The particles are bonded together by means of high pressure and of heat and are thus molded to shape. Often machining can be eliminated with savings on labor and scrap. Owing to the great pressures required, as high as 40 tons per sq. in., application is limited to rather small objects.

While the development of alloys in powder form offers an interesting future, the chief commercial metallic powders available are the pure metals, including copper, nickel, cobalt, chromium, aluminum, magnesium, silicon, lead, zinc, iron, tungsten, molybdenum, tantalum, silver, gold, platinum, and iridium.

Methods of Production

There are 12 principal methods by which metallic powders may be obtained as follows:

1. Machining. Filings give the finest particles. Lathe turnings (especially iron) give a product suitable for use as a chemical reagent or as a material for mixing into hard-finish concrete. Except for making expensive dental alloys, this method of manufacture is too expensive unless the powders are obtained as a by-product.

2. Crushing and Grinding—Ball, Stamp, or Attrition Mills. The metal is usually, though not always, shotted before the milling operation. Brittle metals or alloys can be ground easily; the harder metals or alloys (unless cooled) generate so much heat that they become malleable. Sometimes scrap cast iron is crushed and ground for use as a chemical reducing agent (with acid). The more malleable metals, when ground, form flakes, and by such an operation the various powders used for pigments are produced. In the grinding of the more malleable metals, the cooling, to prevent welding into lumps, is usually done with a lubricant such as stearic acid carried in a suitable solvent. The technique of manufacture is very similar to the fine grinding of an ore prior to concentration operations—a continuous flow of the liquid carries out the finest particles, which are then separated into classifiers and the coarser portions returned to the mill. The sludges containing the more finely divided material are then dried by evaporation which leaves every metallic particle covered with the lubricant. The metal particles are then polished in a drum containing internal brushes. The resulting products are aged for varying periods, weeks or months, during which time they undergo some surface change by which they acquire the property of floating on the surface of a paint vehicle and thus dry into a skin of overlapping flakes. Aluminum flake is used in curing rubber and in fireworks in addition to its use in paint. Copper flakes are used for commutator and collector rings.

3. Shotting Molten Metal in Water or Air. The operation of shotting is carried out by pouring liquid metal through a screen and permitting the drops to become spheroidal through surface-tension properties during the free fall into a quenching tank. Shotted lead, for use in firearms, is made in this way; aluminum shot for use in deoxidizing steel, is also made in this way. "Feathered" and "sherardizing" zinc is made by dropping liquid zinc into water.

4. Graining or Granulation by Stirring Molten Metal While Solidifying. If a molten metal is stirred vigorously during the process of solidification, finely divided particles of the metal with a surface coating of the oxide of the metal involved are formed. The resulting particles are rather coarse in size but are, for some metals, sufficiently finely divided; examples of such use are aluminum for use in the Thermit process and granulated brass for use in brazing.

5. Atomizing or Disintegrating with Steam, Compressed Air, or Other Fluids. In the atomization process a trickle of metal is dropped into a blast of steam or air; an inert gas could be used, but it would greatly increase the danger of spontaneous combustion because particles coated with a surface of the metallic oxide (containing about 0.2 per cent oxide by weight) are reasonably safe. The particles produced by the atomization process are not spherical but are rather of a ragged teardrop shape. The particle size of metallic powders obtained by atomization is controlled to a great extent by blowing the steam or air under the surface of the molten metal. Atomized zinc particles are used as a chemical reducing agent and for sherardizing; atomized aluminum particles are used in fireworks and in calorizing; and aluminum plus copper particles are used in certain types of heating pads for therapeutic medicinal purposes.

6. Condensation of Metal Vapor. A pertinent example of the condensation method of producing finely divided metal powders is in the operation whereby "blue powder" zinc is condensed to a frost rather than a liquid. This operation of condensation gives a more finely divided particle size but it contains more oxide, particularly if it is secured as the cheap by-product of a distillation process.

7. Dissociation of Metallic Carbonyls. The metallic powders obtained through the dissociation of metallic carbonyls are spherical and built up in layers, much as is an onion, and are said to range from about 1 to 10μ in size. Iron and nickel obtained from carbonyl powders are said to be commercially available.

The general method of producing metallic carbonyls for subsequent dissociation into metallic powders is to obtain roasted oxides (usually in the form of roasted ore), which are reduced with hydrogen and then treated in a superatmospheric pressure vessel with carbon monoxide at a slightly elevated temperature. The carbonyl of the metal, *e.g.*, iron carbonyl, $Fe(CO)_5$, is evolved as a gas. When the temperature and pressure are decreased, frequently by dilution of the gas with nitrogen, the metallic carbonyl decomposes, and the metal is precipitated in the form of a shower of finely divided particles. Such particles are very pure except for their content of carbon and oxygen.

8. Reduction of Oxide Powders. Typical products made by the reduction of oxide powders are such refractory metals as tungsten and tantalum. Chemically purified oxides are heated and reduced to a metallic sponge, usually with hydrogen as the reducing agent.

9. Chemical Precipitation. In addition to the manufacture of metallic copper particles (and zinc sulfate as a by-product) by the treatment of copper sulfate with metallic zinc, mention should be made of the production of "cement" copper by precipitating copper from mine water or dilute leaching solutions by passing it over iron scraps. The material produced by this last-named process usually requires further purification, although such a material as finely divided tin made by precipitation on zinc from chloride solution is frequently useful as it is.

10. Electrolytic Deposition. The process of electrodeposition is utilized to manufacture many metallic powders. The conditions of operating the electrolytic cell are varied to produce a coherent plate which is composed of a spongy deposit. Increasing the current density or the temperature or by operating with a highly acid solution is said to be effective in producing the types of electrochemically deposited metallic powders. The products obtained are light and fluffy crystals. They are usually washed carefully and then dried in a more or less inert gas.

An example of this process is the production of tantalum (and columbium) powder by the electrolysis of fluorides. This powder is coarser than that obtained from the reduced oxide and contains up to 100 volumes of hydrogen which must be eliminated.

11. The Hydride Process. Metals of the fourth group of the Periodic Table react with, or absorb, hydrogen. Metallic calcium and calcium powders are produced by this process, and other hydrides, such as titanium and zirconium, are produced by reaction of calcium hydride with the oxides of other metals.

12. Distillation. The removal of a volatile constituent from an amalgam or alloy by distillation or dissolution leaves the more refractory metal behind as a very finely divided sponge that is highly reactive. This is useful for pyrophoric materials or for catalysts.

Applications of Metal Powders

A variety of products is possible from metallic powders, dependent upon the processing to which the powder is subjected and on the particular combination of powders used. Thus there are two general types of particle bonding: (1) particle to particle adhesion may be developed (without melting) as in tungsten filaments, and (2) the bonding may be effected by the cementing action of a constituent which is molten at some stage of the consolidating operations, such as in the cemented hard carbides. There are three general types of combinations: (1) of similar metallic particles, such as the usual forms of tungsten, molybdenum, and tantalum; (2) of mixtures of different metallic substances, such as the welding electrodes of tungsten and copper; and (3) of combinations of metallic and non-metallic particles, such as silver-graphite contact material.

The following gives a classification of applications of metallic powders at present:

1. Preparation of Refractory Metals. Examples of such refractory metals that are produced from powders because they are unprocurable otherwise are tungsten, tantalum, columbium, and molybdenum.

2. Manufacture of a Metallurgical "Concrete." A metallurgical "concrete" is defined as an aggregate which retains the characteristics of its components. This class of products includes the following major examples:

a. Cutting or Abrasive Materials. Hard carbides of metals, borides of metals, diamonds cemented with some ductile metal such as cobalt.

b. Electrical contactors of silver (for conductivity) and graphite (for lubrication).

3. Manufacture for Unique Structural Effects.

a. Porous bearings and oil-pump gears.

b. Continuous sponge of high conductivity copper strengthened by hard, unyielding tungsten for spot-welder electrodes.

c. Contact points with flaky copper arranged in parallel layers to conduct current.

d. Disks having magnetic and non-magnetic components.

e. Compound mixtures, which by controlled heating limit the amount of alloying at contact points and thus make articles of two or more metals which transcend the microstructural limitations of melting and casting. This is particularly important in magnetic alloys such as iron-cobalt-nickel.

f. Addition of a somewhat porous surface to a piece of rolled metal. Examples of such use are as follows:

(1) Clutch disks of steel with a thin layer of copper on the surface.
(2) Sprayed coatings from a "flame gun" using metal powders instead of fusible wires.

g. Extruded metal products. Bimetallic materials, pressed to give a laminated metal with a metallic bond, are not limited to low temperature, as are soldered products.

4. Use in Powder Form.

a. Shotted aluminum for deoxidizing steel.

b. Aluminum powders and ammonium nitrate have detonating properties. Also aluminum and magnesium powders with oxidizing agents, such as perchlorates, in pyrotechnics.

c. "Gold bronze powders," which are flaked alpha-brass, used in graphic and decorative arts.

d. Many uses such as catalysts, dental metals, ceramic decoratives.

5. Manufacturing Savings. Because of the elimination of waste and the reduction in labor requirements, growth is expected in this field. Of particular promise are products involving highly pure metals. Also, the manufacture of coins from metal powders rather than from metal strips may be cheaper, especially if the costs of handling metal strip scraps are included in the comparative economics of the two sources of metal.

b. Addition of a somewhat porous surface to a piece of rolled metal. Examples of such use are as follows:

(1) Mild steel strip with a thin layer of copper on the surface.

(2) Sintered products from a "Bone-clip," where many powders [illegible] of ductile wires.

c. Extended metal products: Bimetallic materials, powder alloys, a laminated metal with a metallic bond, etc. not limited to low temperature [illegible] products.

4. **Use in Powder Form**

a. Sintered aluminum for deoxidizing steel.

b. Aluminum powders and magnesium powders have desirable properties. The aluminum and magnesium powders with oxidizing agents, such as perchlorates, in pyrotechnics.

c. "Gold bronze powders," which are flakes, alpha-brass, used in graphic and decorative arts.

d. Many uses as surface catalysts, chemicals and ceramic decoratives.

5. **Manufacturing Savings.** Because of the elimination of scrap and the reduction in labor requirements, growth is expected in this field. Of particular economic and product importance is the [illegible]. Also, the manufacture of strips from metal powders rather than from metal strips may be [illegible], especially if the costs of handling metal strip scrap are included in the comparative economics of the two sources of metal.

SECTION 17

MEASUREMENT AND CONTROL OF PROCESS VARIABLES

BY

Henry L. Young, B. S., Sales Department, Aridye Corp., Formerly Market Research Director, *Chemical and Metallurgical Engineering*

Theodore R. Olive, A. B., Associate Editor, *Chemical and Metallurgical Engineering;* Junior Member, American Society Mechanical Engineers

REVISED BY

Henry S. Winnicki, B. S., Assistant Professor of Chemical Engineering, Clarkson College of Technology; Junior Member, American Institute of Chemical Engineers.*

CONTENTS

---

* Appreciation is expressed to E. D. Haigler, Manager of Application Engineering, The Foxboro Co., for his helpful comments in reviewing this revision.

MEASUREMENT AND CONTROL OF PROCESS VARIABLES

REFERENCES: *Thermometry and Pyrometry:* Wood and Cork, "Pyrometry," McGraw-Hill, New York, 1927. *U. S. Bur. Standards Publ.:* Pyrometer Practice, *Sci. Paper* 170, 1921; Radiation Pyrometry, *Sci. Paper* 250, 1915; Platinum Resistance Thermometry, *Sci. Paper* 124, 1909. Pyrometer Symposium, *Trans. Am. Inst. Mining Eng.*, vol. 62, 1920. Burgess and Le Chatelier, "Measurement of High Temperatures," Wiley, New York, 1912. Griffiths, "Methods of Measuring High Temperatures," Griffin, London, 1925. Griffiths, "Pyrometry," Pitman, London, 1926.

Thermometry and Pyrometry, Pressure, Flow, and Gas Analysis: Diederichs and Andrae, "Experimental Mechanical Engineering," vol. 1, Wiley, New York, 1930.

Flow: Martin, "Flow of Gases, Etc.," Henley, London, 1928. Ower, "Measurement of Air Flow," London (U. S. agents, Instruments Publishing Co.). Eason, "Flow and Measurement of Air and Gases," Griffin, London, 1919. "Flow Meter Engineering," Foxboro Co., 1930. "Flow Meter Engineering Handbook," Brown Instrument Co., 1936. "Report of A.S.M.E. Special Research Committee on Fluid Meters," Am. Soc. Mech. Eng., New York, 1937.

Physical Properties: Reilly and Rae, "Physico-chemical Methods," 2d ed., Van Nostrand, New York, 1933. Grant, "Measurement of Hydrogen-ion Concentration," Longmans, New York, 1930. Kolthoff and Furman, "Potentiometric Titrations," Wiley, New York, 1931. Dennis and Nichols, "Gas Analysis," Macmillan, New York, 1929. Lunge, "Technical Gas Analysis," revised and rewritten by H. R. Ambler, Van Nostrand, New York, 1934. Palmer and Weaver, Thermal Conductivity Gas Analysis, *U. S. Bur. Standards, Tech. Paper* 249, 1924.

Electrical Measurements: "Standard Handbook for Electrical Engineers," 7th ed., McGraw-Hill, New York, 1941. Laws, "Electrical Measurements," 2d ed., McGraw-Hill, New York, 1938. Gray, "Absolute Measurements in Electricity and Magnetism," Macmillan, New York, 1923. Smith, "Electrical Measurements in Theory and Application," 3d ed., McGraw-Hill, New York, 1934. "Handbook for Electric Metermen," National Electric Light Association, New York, 1924.

Miscellaneous: Zworykin and Wilson, "Photocells and Their Application," Wiley, New York, 1930. Hughes and DuBridge, "Photoelectric Phenomena," McGraw-Hill, New York, 1932. Henney, "Electron Tubes in Industry," 2d ed., McGraw-Hill, New York, 1937. Grebe, Boundy, and Cermak, Control of Chemical Processes, *Am. Inst. Chem. Engrs.*, June, 1933 (Interpretation: *Chem. & Met. Eng.*, October, 1933, pp. 520–525). Grebe, Elements of Automatic Control, *Ind. & Eng. Chem.*, Nov., 1937.

GENERAL CONSIDERATIONS

All chemical engineering operations involve the measurement and control of one or more process variables, the number depending on the type and complexity of the operation. Sometimes the judgment of the operator, or a simple mechanical device not ordinarily considered a controller, will suffice for proper regulation of the variable, as, for example, in determining the rate at which material can be permitted to enter a grinding mill. With the majority of process variables, however, efficient operation cannot be predicated on unaided human judgment, either because of its inaccuracy or because such control is too costly, and it is necessary to invoke the assistance of instruments or apparatus capable of measuring certain physical or chemical properties or effects of the variable.

In its simplest form such an instrument is a mechanism which measures the property chosen and indicates the measurement in terms of the instantaneous values of the variable. In this form the instrument serves as a minute-to-minute guide for manual control. In many cases, as a check on operation, it is also desirable to include automatic recording so as to keep a record of the successive values of the variable; or, if flow of a fluid, a solid, or electricity is involved, it may be valuable to integrate, *i.e.*, totalize, this flow, either with or without a record of instantaneous values. Finally the instrument may be designed not only to measure the variable but to use this measurement automatically in effecting adjustments which will maintain the variable substantially constant, or change it in accordance with some prearranged schedule. Thus the instruments and apparatus used in controlling process variables are found to fall into three broad classes: (1) indicators, (2) recorders, and (3) controllers. Various combinations of these functions are also possible, as, for example, an instrument which indicates and records, indicates and controls, or indicates, records, and controls.

Process Variables. A process variable is any condition or state of the process material, or its environment, which is subject to change. To produce a desired final result, it will ordinarily be necessary to control most of the variables associated with any process, within more or less closely held limits. Not every variable will require separate control, however, since the adjustment of some other condition may hold it within the desired limits. In a boiler producing dry, saturated steam, for example, control of the pressure will automatically yield steam of a definite temperature, but to maintain the pressure the rate of heat addition must be equal to its rate of withdrawal. This will involve, in addition to pressure regulation, control of the supplies of fuel, air, and water.

In other cases it is possible to choose which of several related manifestations of a process variable shall be used as a basis of measurement or control. In regulating fluid flow, for example, the control property may be volume or weight per unit of time, velocity, pressure drop through an orifice, head above a weir, or temperature rise for a given heat addition. In diluting an electrolyte, control may be based on density, electrical conductivity, hydrogen-ion concentration, or the weight, volume, or rate of flow of the separate components. Thus it is evident that every process must be analyzed to determine what the variables are and whether or not each variable will require separate control. When this has been accomplished, the engineer will often find that he has considerable latitude in his choice of the control property.

Choice of Control Equipment. It should first be understood that the term "control equipment" refers not only to automatic controllers but also to indicators and recorders. Once the engineer understands clearly which variables in his process must be controlled, and within what limits, he is in a position to determine the method and the instruments that must be used. Will manual control suffice, or should the regulation be automatic? Is a record necessary, and is a total of the variable desirable? Categorical answers to these questions are impossible, but the engineer must answer them before he can make a choice. If only occasional regulation by hand will be necessary, manual control is often sufficient. If the variable is an important one, its recording will ordinarily be worth while, whether the control be manual or automatic. If the variable may fluctuate widely or rapidly, and particularly if its fluctuation has an important bearing on the success of the process, an automatic controller is usually called for, since only an automatic controller can be continuously attentive. Automatic control will generally be more accurate than manual, with resultant improvement in the product. Although it cannot eliminate human supervision, it may reduce labor and hence operating cost to a marked extent. Its worth, then, in comparison with manual control, must be determined on a purely economic basis.

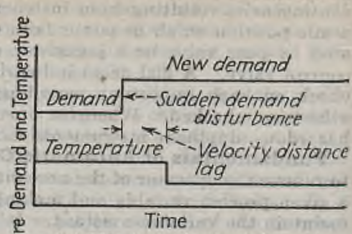
Once the general type of control has been settled, the specific method must be chosen. Here the determining factors are likely to be its suitability *vs.* its cost. For most variables the available instruments cover a fairly broad range of cost, accuracy, and reliability. Here, again, economics must govern, taking into consideration, of course, the effect of possible contingencies and hazards that may arise following the choice of a less reliable instrument. But, in addition to the choice of the best instrument that is economically suitable, independent means should be provided as far as possible for handling contingencies resulting from instrument failure. Valves which will assume a safe position on air or power failure should be selected. A manually operated by-pass valve or a jackscrew should be provided for each important control valve. A dial or an industrial thermometer should be provided as a check on each important recording thermometer not readily removed or otherwise checked. Wherever instrument failure would be particularly hazardous, duplicate instruments should always be installed.

Fundamentals of Automatic Control. Automatic control is intended to relieve the operator of the necessity of determining frequently the value of a given process variable and making the changes in the system required to maintain the variable constant. It is not, however, intended or expected to relieve him of responsibility for correct operation. In its simplest form, an automatic controller consists of a sensitive or detecting device which measures the state of the variable continuously or at frequent intervals and which cooperates with some other device, such as a valve, to bring about changes when the variable departs from a predetermined value called the **control point.** In an ideal controller the detection and correction of changes would be so rapid that the variable would remain substantially constant, regardless of conditions. Such control, however, is rarely attained nor is it often necessary. In practical operation, maintenance of the variable within a certain range of values is generally all that is required. The narrower the range, of course, the more difficult it is to produce a satisfactory controller and the more complicated and expensive the controller becomes.

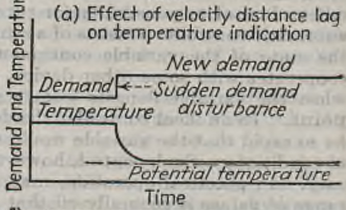
In any process there is a definite demand of matter or energy which is constantly met by a definite supply. When the demand and supply are such that equilibrium exists, the process is at the control point. A disturbance may occur such that either the demand or supply condition changes to a new value, so that the process tends to vary from the control point. Any such process disturbance is termed either a **supply disturbance** or a **demand disturbance** depending upon its location.

Process Lags. After a supply or demand disturbance occurs in a process, the detecting element must be able to determine the extent and direction of the variation from the control point before corrective action can be made by the rest of the control system. Certain characteristics of the process and control system will cause both a delay and retardation of a true indication of the extent and direction of the disturbance. These delays and retardations of actions and occurrences of both the control system and controlled process are termed **process lags,** sometimes called **time lags,** although the element of time is not involved in all process lags. Process lags are divided into three distinct classifications, *i.e.*, **velocity-distance lag, capacity lag,** and **transfer lag.**

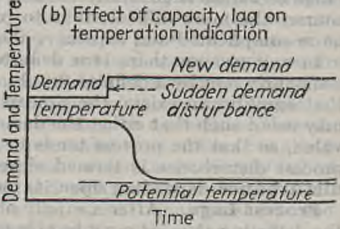
Velocity-distance lag or finite-time lag is a definite time delay between the time a supply or demand disturbance occurs and an effect is experienced by the control system. For example, two streams of water at different temperatures meet at a common point and are allowed to flow a certain distance before the temperature is measured. A definite time will be required for the water to flow from the point at which it is introduced to the point at which the temperature is measured; this then would be the velocity-distance lag. If an automatic control was operated by the temperature-measuring device, some time might elapse between the time when the temperature-sensitive element "felt" the change and corrective action was started by the controller, which would also be part of the velocity-distance lag. However, in general, commercial instruments are so constructed that the time lag can be considered negligible. Curve *a*, Fig. 1, shows the effect of velocity-distance lag upon the condition of the process after a sudden process disturbance occurs. Velocity-distance lag is the only process lag measurable in units of time.



(a) Effect of velocity distance lag on temperature indication



(b) Effect of capacity lag on temperation indication



(c) Effect of transfer lag on temperature indication

FIG. 1.—Process lags occurring with sudden process disturbances.

Capacity lag is a retardation of the magnitude of a given process variable when a related variable affecting the process is changed due to the capacity or inventory of matter or energy contained within the process itself. In practically all processes a definite inventory exists which slows down the effect of either a sudden supply or demand disturbance. Thus, if two streams of water were mixed together in a tank of appreciable capacity and the temperature of one stream was changed, the temperature of the water overflowing from the tank would slowly approach the new temperature that would exist, after the sudden process disturbance, if no tank were present. This is illustrated by curve *b*, Fig. 1. The rate of temperature change depends upon the capacity of the tank, varying inversely with it.

Mason (*Trans. Am. Soc. Mech. Eng.*, May, 1938) has developed equations for determining the temperature change experienced, due to a process disturbance, by the use of liquid-level analogies—representing thermal energy with the cross-sectional area of a cylindrical tank; the temperature with liquid head; and heat flow with liquid flow. The temperature-time relation is given by his expression

$$T_a = T_p + (T_a - T_p)_0 e^{-\left(\frac{1}{A_a R_a}\right)t}$$

where A_a = heat capacity of the process, defined as the number of B.t.u. necessary to raise the temperature 1°F., or the mass times the specific heat.

T_a = temperature existing in capacity A_a, °F.

T_p = potential temperature, °F., or the limiting temperature approached after a sudden unchanging process disturbance.

R_a = Resistance factor equal to the ratio of any value of T_a to the corresponding value of Q_a which is the heat flow leaving A_a, B.t.u. per minute.

e = base of natural logarithms.

t = time, min., after a sudden process disturbance.

The subscript (0) in the expression $(T_a - T_p)_0$, denotes that T_a is the initial value when $t = 0$. This expression is a straight line if time is plotted on a uniform scale and $T_a - T_p$ is represented on a logarithmic scale.

The actual time for any temperature change to occur is given by the equation

$$t = A_aR_a\left[2.3026 \log_{10} \frac{(T_a - T_p)_a}{T_a - T_p}\right]$$

in which the symbols are the same as before. A_aR_a is considered a time constant which determines the time required for a change in temperature to reach any given per cent of its final value and is equal to the time required for a change to reach approximately 63 per cent of its final value.

Transfer lag is a retardation of the magnitude of the change of a given process variable when a related variable affecting the process is changed due to the resistance retarding the flow of energy or matter between two or more separated capacities of the process. Transfer lag is a characteristic of all processes involving a transfer of matter or energy where a driving force or potential head is required to overcome a resistance or series of resistances in such processes as heat transfer, absorption, etc. For example, in a heat exchanger two separate capacities exist, the fluid being heated and the heating medium fluid, which means that, owing to the resistance to heat flow from one fluid to the other, there will be an additional temperature-lag effect. This is illustrated by curve *c*, Fig. 1.

The equation for evaluating the temperature at any time is given by Mason (*Tran. Am. Soc. Mech. Eng.*, May, 1938) as

$$T_a - T_p = C_be^{-K_bt} + C_ae^{-K_at}$$

in which T_a and T_p are as defined previously. K_a and K_b are found by the expression

$$K_b = -\frac{A_1}{2A_2} - \sqrt{\left(\frac{A_1^2}{2A_2}\right) - \frac{1}{A_2}}$$

$$K_a = -\frac{A_1}{2A_2} + \sqrt{\left(\frac{A_1^2}{2A_2}\right) + \frac{1}{A_2}}$$

where $A_2 = (A_aR_aA_bR_b)$.

$A_1 = (A_aR_a + A_bR_a + A_aR_b)$.

A_a and R_a are as defined previously.

A_b = heat capacity of second capacity.

R_b = ratio of the temperature between A_b and A_a to the coincident flow of heat between A_b and A_a, °F. per B.t.u. per min. Thus $R_b = (T_b - T_a)/Q_b$ in which T_b is the temperature of capacity A_b and Q_b is the flow of heat from A_b into A_a, B.t.u. per min.

C_b and C_a are found by the expressions

$$C_a = \frac{T_{a0}' - K_b(T_a - T_p)_0}{K_a - K_b}$$

$$C_b = (T_a - T_p)_0 - C_a$$

T_{a0}' is the first time derivative of T_a when $t = 0$ and is found by

$$T_a' = \frac{Q_b + Q_0 - Q_a}{A_a}$$

Q_0 being the basic heat supply entering A_a in B.t.u. per minute.

Process lags cause controllers to hunt and to assume a new control point, as in the discussion of controller types (p. 2019) and must be evaluated in the event that large lags exist if a proper controller is to be chosen.

There are two general types of controller. The simpler form includes those controllers, variously called **two-position, on-and-off,** and **open-and-shut,** in which there are only two degrees of regulation. When the variable is above the control point, the controller operates to its limit in the direction tending to decrease the variable; and below the control point, the controller operates to its limit in the reverse direction, tending to increase the variable. A graph of a variable so controlled is a sinuous line, the average of which equals the control point. Curve *a*, Fig. 2, is such a graph. For many purposes this type of control is very satisfactory, since under suitable conditions the cycle amplitude can be made exceedingly small.

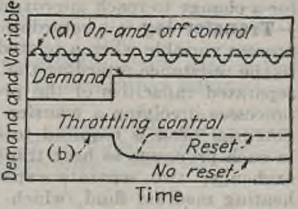


Fig. 2.—Type of regulation with on-and-off and throttling controls.

The second class of controller is known as the **proportional throttling** type. While it will meet the demand without fluctuation, so long as the demand does not change, major changes in demand result in drift or wandering of the control point. This drift can be subsequently corrected by an "automatic reset" mechanism. A graph of a variable regulated by a proportional (or a proportional-reset) controller will be substantially as in *b*, Fig. 2.

Accurate control in a batch operation is usually fairly simple; in a continuous operation, it may be much more complicated. A case of the first sort (batch) may call for heating a cold liquid in a kettle to some predetermined temperature, after which the liquid is held at this temperature. Although the liquid in contact with the heating surface may be slightly warmer than that surrounding the temperature-sensitive element, this will not generally have an undesirable effect. The gradient may be decreased by agitation, and a properly designed controller of either the **open-and-shut** or **throttling** type will have no difficulty in limiting the supply of heat to that required to compensate for radiation, or for the addition of relatively small quantities of cold material.

Suppose, however, that it is desired to dilute an electrolyte continuously with water so as to produce a uniform concentration. Throttling control is desirable, and of the several methods available it may be most satisfactory to use specific gravity (density) as the control property. Although the location of the sensitive element in the case cited above was of little importance, it becomes imperative in the continuous diluter that the gravity controller be placed at the discharge of the diluting tank, since it is the concentration (gravity) of the effluent which is to be controlled. This introduces another element into the control problem, that of process lag, which means that the controller will not "know" whether its regulation has been correct until some time after the fact. Suppose that either the quantity or the concentration of the entering electrolyte suddenly increases. Then for a period equal to the lag, the quantity of diluting water will not be increased to meet the new conditions. When the controller suddenly "discovers" the change, it will increase the flow of diluting water, but, because of the process lag, it will open the dilution valve too wide. Hence, with the "knowledge" of the controller always a step behind the actual circumstances, the resulting control will "hunt," *i.e.*, swing above and below the control point, until the fluctuations are finally damped out. The average result will not be far from correct, but the instantaneous result may vary widely. Figure 3 shows how the dilution might vary in such a case. Note that the simple **throttling** controller settles down at a new control point which is slightly different from the original. Unless this tendency is automatically corrected, as it is in certain instruments, only one condition of demand will give an average result that lies directly on the control point **as set**. Hence, when the demand changes considerably, it may be necessary to reset the control point. One of the **advantages of the on-and-off type of control**, where it can be used, is that the average of its sinuous control curve will remain substantially constant regardless of changes in demand.

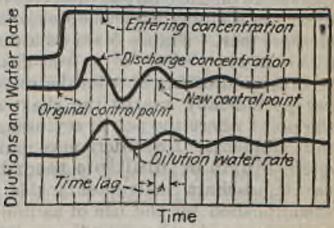


FIG. 3.—Effect of lag on performance of dilution controller.

INSTRUMENTS AND PARTS OF GENERAL APPLICATION

The following division of this section describes a number of mechanisms, parts, and accessories that are common to many types of instrument. In later divisions, where the instruments proper will be treated, these parts will not be further described.

Indicators

No matter what the variable or the type of indicator that is used, the value of the variable is translated into "position" in order that it may be read upon a scale or chart. An indicating instrument or indicator, then, consists of a detecting or sensitive device, a scale (ordinarily graduated in values of the variable), and a method of indicating the measurement on the scale. The mechanism which indicates on the scale may be purely mechanical, consisting of gearing or a linkage, or it may be electrical. In the latter case, a movement such as one of those described on pp. 2082 to 2083 will be used. A permanent-magnet, moving-coil galvanometer is most frequently encountered since, in most instruments used by the chemical engineer, direct current is to be measured, as in pyrometers and resistance thermometers.

Recorders

The pointer of an indicating instrument may be supplied with a pen or stylus arranged to travel over a paper chart which is in continuous motion. There are three principal types of recorder, those using (1) a circular chart, (2) a strip or roll chart which is continuously unwound from a roll, and (3) a strip chart mounted on a drum. The first two are the more common. The speed of the chart may be governed by a clock, whereupon the variable is plotted against time; or the speed may be controlled by the instantaneous rate of some other variable such as flow, whereupon the variable being measured is plotted against the second variable controlling the chart speed. Recorders, then, consist of a sensitive or detecting mechanism, an indicating pen or stylus, a chart, a means for advancing the chart, and a case for enclosing the entire mechanism.

Recorder Mechanism. Recorder cases may be made gas- and dust-tight. Instruments using circular charts are ordinarily fairly tight without special precaution. Some roll-chart instruments require an auxiliary case for tightness. In most round-chart instruments the position of the pen is controlled by a mechanical linkage, as in Fig. 4. Since the indicating mechanism in some roll-chart instruments has little power, special means must be taken to avoid friction of the pen or stylus on the chart. Hence a frequent procedure is to impress the indication on the chart at intervals by means of a clock-driven depressor or chopper bar. Figure 5 shows a diagrammatic view of a roll-chart recorder, indicating how the depressor bar periodically marks the chart at the point where the pen or stylus is at that moment. Recorders may make either **single or multiple records.** In round-chart instruments this is accomplished by the use of as many indicators and pens recording on the same chart as there are records. A roll-chart instrument, using an electrical indicating mechanism, often has but one indicator which is switched consecutively to a series of detecting or sensitive devices. The switch is operated at proper intervals by the clock. In order that the records may be distinguished, points are marked in different colors, by various combinations of dots and dashes, or by a number corresponding to each of the several detecting devices. Rotating print wheels, carbon paper, or movable ribbons carrying several colors are used in this distinguishing process.

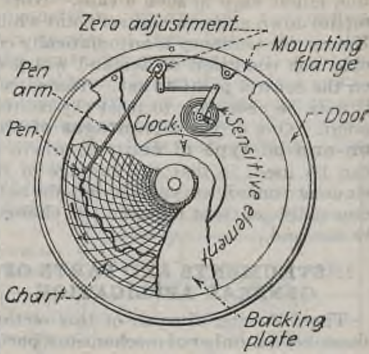


FIG. 4.—Round-chart recorder mechanism.

Pens and Styluses. Single-record recorders with mechanical indicating devices use capillary pens consisting of a reservoir for ink, terminating in a point which is held in contact with the chart. Pens are made of glass, monel metal, or some other corrosion-resisting material. The ink generally consists of water, a dye, and glycerin. To lower the ink viscosity for rapid variations, alcohol may be substituted for glycerin. For recorder movements with little power, or for very rapid fluctuations, a stylus is more generally used. For continuous contact the charts may be coated with wax or with finely divided carbon. In intermittent-contact (depressor-bar) instruments the stylus prints through a piece of carbon paper or a typewriter ribbon.

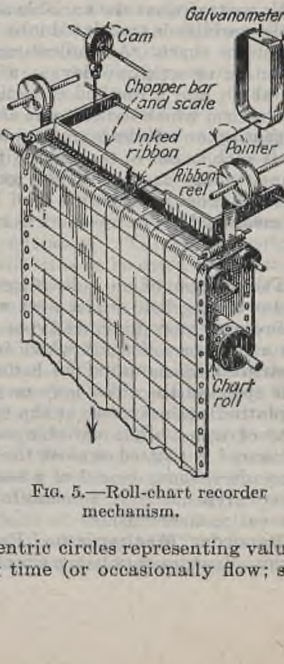


FIG. 5.—Roll-chart recorder mechanism.

Charts. Round charts, made in diameters from 4 to 12 in., are marked with concentric circles representing values of the variable, and radial arcs representing time (or occasionally flow; see above). Round-chart records are ordinarily for 24 hr. or 7 days, but rotation periods from 1 min. to 28 days are available. Roll charts are made in lengths up to 120 ft. and widths to 12 in. Lengthwise lines are provided for values of the variable and crosslines for time. Chart speeds in standard roll-chart instruments vary from ½ to 6 in. per hr., with extremes of from 1 in. per day to 3 in. per sec.

Clocks. These may be either spring driven or, where alternating current is available, of the synchronous electric type. In round-chart instruments, the clock furnishes power to rotate the chart, but in strip-chart recorders, it may be called upon only to operate electric contacts which in turn cause a solenoid to move the chart and the depressor bar. Multiple-speed clocks are available, and often are very useful in test work. Several strip-chart types and one 24 hr./24 min. round-chart type permit use of standard charts at high speeds.

Controllers

If the fluctuations of the variable can be measured, they can usually be controlled automatically. Most frequently this involves the control of flow: flow of either a fluid, a solid, or electricity. Hence it is necessary for the measuring instrument to actuate valves, dampers, switches, rheostats, or other mechanisms, either directly or through the agency of some device which will increase the power available from the detecting mechanism. In the so-called "self-operated" controllers and regulators, movement of the sensitive device operates the valve, etc., directly, but, since these movements are rarely very powerful, this method is likewise the crudest and least accurate. For greatest accuracy it is necessary to employ intermediate agencies which can readily be controlled by the detecting element and which in turn are powerful enough to adjust the controlling valve or other device. The intermediate agencies which are used thus to increase the power of the controller include fluid-relay devices known as pilots or pilot valves, mechanical-relay devices, and electric circuits.

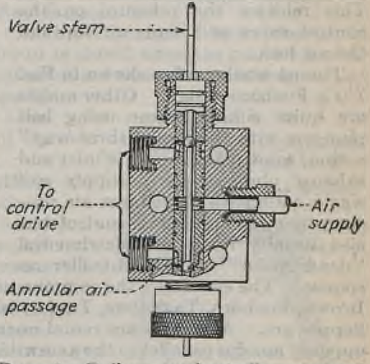


Fig. 6.—Bailey mechanically operated air-pilot valve.

Pilot devices employ a fluid under pressure which is permitted to escape at a regulated rate, or to build up pressure in the control device, under the constant control of a small valve which is actuated by the sensitive element of the controller. Air, at about 15 lb. per sq. in. pressure, is the fluid most generally used, although water, steam, oil, and sometimes process gases under pressure may also be employed. It is the pressure of this fluid, adjusted by the pilot valve and the sensitive element, that controls the process variable, acting as it does on a control valve, damper, switch, or other control device.

Air-pilot valves are of two general types: (1) valves in which the valve plug is positioned by a direct linkage from the detecting element; and (2) valves in which a very minute movement of the detecting element accomplishes a much larger movement of the valve plug through the assistance of an intermediate air pilot. The first type, Fig. 6, is used where throttling control is desired, and where ample measuring element torque is available. Through ratio adjustments in the mechanical linkage the throttling range may be varied. Through change of air connections, Fig. 6, or mechanical linkage, or sometimes through change of valve body construction (similar to Fig. 7), the control action for a given change in measured variable can be reversed.

The second type, Fig. 7, is used where either on-and-off or throttling control is desired and a minimum load is to be imposed on the measuring system. Air from the air supply flows continuously from a small nozzle until a change in the variable causes the measuring element to move a small flapper so as to close the nozzle. While air is venting through the nozzle, the valve plug is held against the left-hand seat by the bellows spring, thus closing the air leak. In this position practically the entire air-supply pressure is exerted on the control valve or other device. A very small movement of the flapper, however, serves to close the nozzle, with the result that the bellows expands and moves the valve plug against the right-hand seat. This releases the pressure on the control valve and vents air through the air leak.

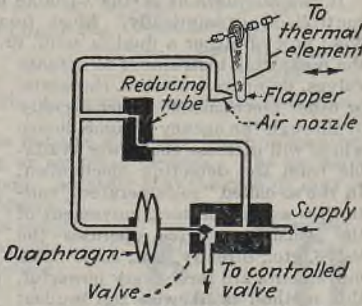


FIG. 7.—Relay-operated pilot valve.

The particular relay shown in Fig. 7 is a Foxboro design. Other makes are quite similar, some using ball plungers with the same "three-way" action, some using separate inlet and exhaust plungers for a supply and waste action which reduces air consumption for throttling control but also usually introduces detrimental "dead space" in the controller response. The cut shows the common nozzle and flapper arrangement (Bristol, Brown, Foxboro, Tagliabue, Taylor) wherein the nozzle jet is tangential to the flapper arc. Also used are radial nozzle with moving sector (American) and opposed nozzles parallel to the axis with vane moving between them (Bristol).

With a relay of the type shown in Fig. 7 designed to throw on small pressure change, an open-and-shut control action can be secured with changes of less than $\frac{1}{4}$ per cent of scale range, and extremely close control without appreciable cycling and without loss of control point is secured on the many processes for which fast open-and-shut control is suited. With the same sensitive relay, throttling-control action can be secured if output air pressure from the relay is connected to a pressure spring, the motor of which is so connected to the nozzle or flapper that any measuring-system motion tending to close the nozzle is immediately counteracted in part by a motion produced by the change in output pressure. By this means the output pressure is continuously compared with and kept proportional to the measuring-system position, and precise throttling action results. This pneumatic throttling action may also be thought of as a "pneumatic return motion" in every way equivalent to the mechanical return motion commonly used in governors and pressure regulators. With a relay designed to throw on a considerable pressure change, throttling-control action can be secured without the pneumatic return motion. The throttling range is adjusted by varying the ratio of mechanical linkage between measuring element and flapper. Slightly simpler than the pneumatic throttling construction, it is also more limited in range and sensitivity.

Electrical systems (see p. 2024) are often used instead of pilot devices. Movements of the measuring element may be made to operate electric contacts to open, close, or otherwise change the setting of the connected control device. The moving mechanism of the measuring element is provided with one or more electric contact points, usually of silver or tungsten, which make contact at certain positions with one or more stationary control-contact points. Figure 8 shows a typical contact-making controller of the indicating type. In the Foxboro "Rotax" type shown in Fig. 9 the contacts are made and broken without affecting the indication or record. The position of the control contacts can usually be adjusted so as to set the control point at any desired value of the variable. In **single-contact instruments** the contact is made only when the pointer is moving in one direction, either up or down. In **double-contact instruments** one circuit is closed when the pointer is moving up (high contact), and another circuit is closed when the pointer is moving down (low contact). Contacts are generally so built that the circuit will remain completed through a range of movement of the pointer.

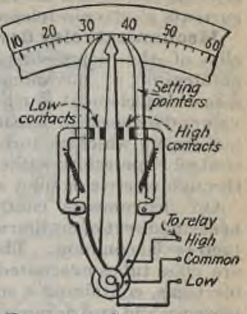


FIG. 8.—Contact-making indicating controller.

In the electronic type of **contactless controller** no physical contact is made to complete an electrical circuit. In the Alnor pyrometer controller a small vane attached to the pointer passes between two vanes which are the capacitive element of a tuned circuit. When the pointer vane interleaves with the capacitive vanes, a circuit frequency change is produced which in turn operates a relay. In the similar Wheelco instrument pointer detection is inductive rather than capacitive.

Relays (see p. 2024) are required when the current to operate the control valve or other device is greater than can be carried by the instrument contacts. Figure 10 shows two common types of relay. That in *a* is a single-contact type used with single-contact controllers. The control circuit is completed only so long as the instrument circuit is completed. The type of relay shown in *b* is used with double-contact instruments. This differs from *a* in including a *holding contact* so that the control circuit remains completed until the instrument contact is made in the opposite direction. This type of relay is used to prevent chattering even when the current demand of the control valve or other device is below the permissible

FIG. 9.—Rotax controller.

limit for the instrument contacts. Instead of operating open contacts some types of relay tilt mercury contact switches by solenoid action. In others the solenoid displaces a mercury pool which makes contact. For very low contact currents electronic relays are available.

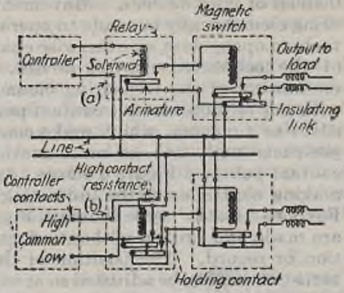


Fig. 10.—Common types of relay. (*a*) On-and-off; (*b*) holding-contact type.

Mechanical relay devices are generally of the depressor-bar type. The pointer of a galvanometer or other measuring element is clamped at intervals and its position detected by mechanical feelers which in turn carry out the control operation either directly or through electric or pilot systems.

On instruments controlling electric heating, direct operation of mercury contactors is common. These contactors are glass tubes evacuated, or filled with inert gas, containing a small amount of mercury and two or more contact points. Tilting makes or breaks the contact. Figure 11 shows two types of mercury-contactor switch: the first, *a*, a type for carrying 2 to 10 amp., and the second, *b*, for carrying from 10 to 50 or 60 amp. In the second and similar heavy-duty contactors, contact is made between pools of mercury across the edge of a refractory glass tube as in the illustration, or sometimes across a refractory dam. Metal enclosed types are made in sizes to 200 amp. rating.

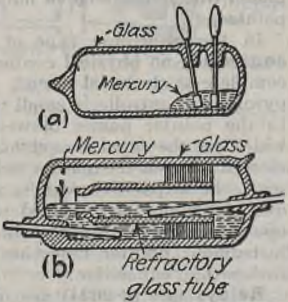


Fig. 11.—Mercury contact switches. (*a*) For 2 to 10 amp.; (*b*) for 10 to 60 amp.

Control Devices. Fluid-operated control devices are those making use of a pilot system (see above) using air, steam, water, oil, or process gases as the pilot fluid. These devices consist of a diaphragm, metal bellows, or piston operating a valve, damper, switch, etc. They may either open or close with increasing pressure (air-opened or air-closed), the type to use depending chiefly on whether it is desired to have the valve open or closed in case of failure of the control-fluid pressure. Diaphragm valves are generally of three types: **butterfly valves** for low-pressure gas or air; **globe** or **V-port valves** for medium-pressure service on all fluids; and **balanced valves** for higher pressures. Depending on size, these valves are obtainable with screwed, flanged, or union ends. Above 150 lb. per sq in. pressure, balanced valves should be used unless tight closing is required, when globe valves are necessary.

Common diaphragm valves for open-and-shut service (Fig. 12) are the single-seated disk (or globe) valves, used on all fluids at moderate pressures and for tight closure at higher pressures, and the balanced double-seated disk valves, used in larger sizes and at higher pressures. Also used are needle valves for very small capacities and butterfly valves for large low-pressure valves, as well as many special types for various chemical services. Common diaphragm valves for throttling service are the single- and double-seated V-port types, especially the latter with two or four ports shaped to give equal percentage flow increments with equal lift increments over flow ranges of 50 to 1 or more. While globe valves are sometimes used for throttling service, the flow range in which the flow-lift characteristic is really suitable for throttling control is narrow, and process data and stability are rarely such that this favorable portion can be utilized with any certainty. Also limiting the use of any single-seated valve on throttling service is the upsetting effect of varying line pressures causing a varying plunger position unless the diaphragm or motor is extraordinarily large.

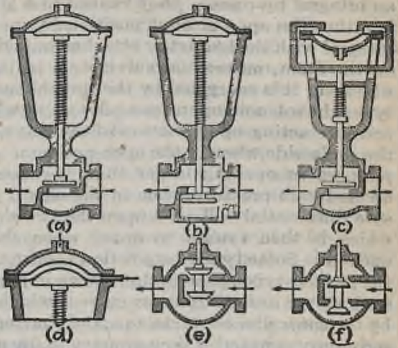


FIG. 12.—Diaphragm valves. (*a*) Air-closed globe valve; (*b*) air-opened globe valve; (*c*) air-opened globe valve, reversed top (*Foxboro*); (*d*) double diaphragm for two-function control; (*e*) and (*f*) balanced valves.

Figure 13 shows a typical wide-range valve characteristic plotted on semilog paper. These flow curves are based on the favorable condition of the valve being the main resistance to flow. Throttling valves are sometimes used for open-and-shut service instead of cheaper open-and-shut valves where it is desired to vary effective valve size frequently, this being done by varying air-supply pressure to the instrument. Less common valve constructions include the needle valve in both open-and-shut and throttling types for very small capacities; the butterfly valve in open-and-shut and limited range throttling types, usually in large sizes and at low pressures; and the plug cock; as well as a wide variety of special material valves for chemical service.

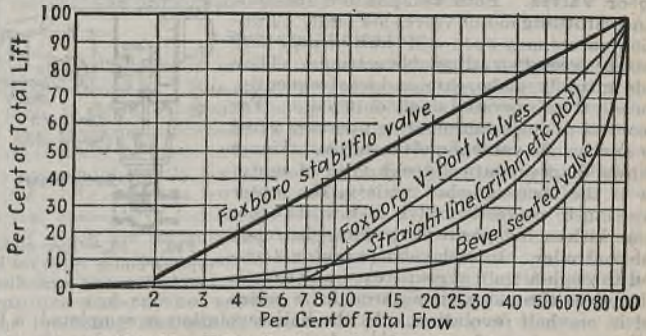


FIG. 13.—Valve characteristics of control valves.

Diaphragm Switches and Levers. For control devices other than valves, pilot-operated controllers make use of switches, dampers, etc. For operating dampers a diaphragm motor connected to a lever may be obtained. Instead of a lever the diaphragm may be connected directly to a switch for controlling a motor or an electric heating circuit.

Electric control devices are similar to the fluid operated ones, but are actuated by electric motors or solenoids or occasionally by electric heaters.

Solenoid Valves. Solenoid valves in nearly every case are of the completely open-and-shut type (Fig. 14), although they are occasionally built with an integral by-pass. They consist of a globe, needle, or balanced valve, held in either the open or shut position by means of a spring. A solenoid, either integral with the bonnet or attached outside and operating on the stem, moves the valve stem to its other extreme whenever it is energized by the control current. In some types the solenoid operates a pilot valve which permits the pressure acting upon the top of the valve to be exerted to the underside when in the open position. By making the pilot valve opening larger than the opening admitting downstream pressure when in the closed position, a pressure differential will exist upon the two sides of the valve, which is then caused to open, when the pilot valve is opened. Some types use a bellows construction instead of the packed-type construction shown in Fig. 14. Although such valves ordinarily draw current which can be handled by the controller contacts, to avoid chatter due to frequent secondary contact, it is customary to use a double-contact controller and a relay of the type shown in Fig. 10*b*. Most makers supply these valves in sizes not over 3 in. for pressures up to 150 lb. per sq. in., although specially constructed valves which make use of the pressure of the fluid handled to operate the valves have been built in sizes to 12 in. and 400 lb. per sq. in. pressure. Solenoid valves are made for both alternating and direct current.

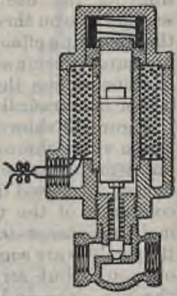


FIG. 14.—Solenoid valve.

Solenoid valves are also used as pilots for fluid-control devices. Where valves are very large or exceptionally fast action is required from remote valves, this is a convenient method.

Motor Valves. Both two-position (on-and-off) and throttling motor valves are built. Two-position valves may open and shut, or they may operate between two adjustable settings. They include butterfly, globe, gate, and, less frequently, balanced valves, operated singly or in pairs. For fuel control a single motor often operates a fuel valve and an air valve simultaneously. Figure 15 shows diagrammatically one type of motor valve of the open-and-shut variety, the motor functioning to move the valve disk to its other extreme with each opposite control impulse made by the controller. In such valves a motor is connected through a train of gears to a crank or cam which moves the valve stem through its entire travel in one-half revolution. As the half revolution is completed, a limit switch is opened, at the same time closing contacts connected to the second pair of controller contacts, so that a full revolution of the valve cam shaft is completed on the next opposite control impulse. Open-and-shut motor valves are customarily by-passed to take care of minimum flow requirements while the valve itself is closed. Such valves are often used for remote manual control. Motor units are available for attaching to standard valves.

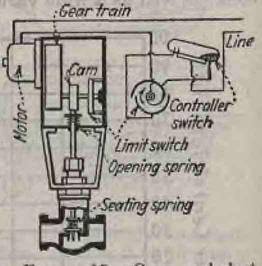


FIG. 15.—Open-and-shut type of motor valve for high-and-low contact controllers.

In some electric valves the actual valve stem operation is hydraulic, the motor driving an oil pump which in turn furnishes oil to the valve-operating cylinder. In the larger sizes the electrohydraulic operation may be cheaper and far more durable.

Throttling motor valves are of a number of types, some very ingenious. In the simplest form this gives but a crude approximation of true throttling or proportional position control and can be used only where there is very little lag between a correction made by the valve and its effect on the controller. A reversing motor operates the valve, running in one direction for a "more" contact, remaining stationary as long as the controller is not making contact, and running in the opposite direction as soon as a "less" contact is made. For greater lag, a compensating, step-by-step, or delayed-action device must be inserted in the control circuit to avoid overshooting the control point. Without some such device, even a small lag will result in "hunting." In step-by-step valves, a "more" contact will increase the valve opening slightly, whereupon the mechanism disconnects the motor from the valve for a brief, adjustable interval and waits to determine whether the valve adjustment is sufficient. If it is not sufficient, adjustments continue at intervals until the control point is reached. More complicated devices are designed to make valve changes of decreasing magnitude as the control point is approached or even to make a negative correction shortly before reaching the control point in order to avoid overshooting. A method employed in the Shallcross control (p. 2034) uses carbon contacts and a shunt-wound motor to obtain variable motor speed. True throttling or proportional position control is secured in motor valves (most often those used with potentiometer instruments) by using corresponding resistors and movable contacts in both motor mechanism and instrument. These resistors are connected in a bridge circuit with a sensitive polarized magnetic or electronic relay to detect unbalance and to operate the valve motor in a direction to rebalance the bridge. Thus the valve is maintained in a position corresponding to the value of the measured variable and proportional position control results.

Remote Indication

It is often desirable to produce or reproduce indications of a measuring device at a distance. This may be used as a means of centralizing control of a plant in conjunction with the use of remote-control valves; or it may be used as a check on operation; or as a means of obtaining indications which would otherwise be inaccessible. Most types of industrial measuring instrument possess inherently the property of indicating at a moderate distance, but this rarely exceeds a few hundred feet and is usually much less. Several electrical methods of indicating position at a distance have been devised, most of which are theoretically unlimited in the distance over which they can operate. Practical considerations usually limit these methods to a few miles. The telemetering methods most used include: (*a*) the **step-by-step** or **impulse-and-ratchet** system, in which a given small angular movement of the indicating instrument results in the sending of a single electrical impulse which causes the same angular movement at the receiving instrument; (*b*) the **rheostat-and-ammeter** system, in which the sending instrument adjusts a rheostat, the position of which is indicated at the receiving end on an ammeter; (*c*) **Wheatsone bridge** system, in which the indicator changes the position of a slide-wire contact of an adjustable resistor in the sending instrument and is balanced by a self-balancing recorder at the receiving instrument; (*d*) the **Selsyn system**; (*e*) the self-balancing **induction balance** or **induction bridge**; (*f*) **time-impulse** system; and (*g*) **pneumatic** system.

Induction Balances. Figure 16 shows the Bristol form of induction balance which will be seen to consist of two circuits having a common leg. In each circuit are two inductances, one at the sending end and one at the receiving end. The sending inductances (and also the receiving inductances) are suspended in the form of a balance over the two legs of a permanent magnet. When alternating current is applied to the circuit, the position assumed by sender and receiver will be such that the inductance of each circuit is equal. Hence any movement of the sender will be duplicated at the receiver. The sender may be connected to the indicating mechanism and the receiver, at a distance, connected to a recorder or indicating pointer.

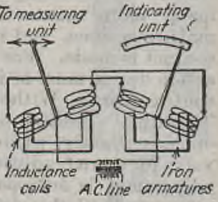


FIG. 16.—Bristol type of induction balance.

The Brown induction balance transmission system is identical in principle with the Bristol but uses cylindrical iron armatures inside center-tapped cylindrical coils. The Cochrane system is a further development of the principle using a galvanometer and mechanical relay (depressor bar) operated balancing mechanism driven by an electric motor. It is more accurate than the simple bridge because the power for record and/or control is not limited, but more costly because more complicated.

Selsyn motors are used in pairs as sender and receiver. These are small self-synchronous motors connected preferably to the same source of alternating current and interconnected as shown in Fig. 17. The Selsyn system may be thought of as a three-phase version of the induction balance. If the armature of the sender motor be rotated through any angle, the armature of the receiver motor will rotate through the same angle. This angle may be more or less than one revolution. Any method, such as the indication of an instrument, may be used to determine the position of the sender motor, and the receiver motor will accurately reproduce the indication. The De Florez remote-control valve similarly uses two Selsyns, the sender being connected by gearing to a handwheel while the receiver is connected by gearing to the valve. Since only the force exerted on the handwheel is communicated to the valve, the operator can feel any obstruction.

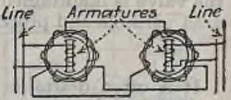


FIG. 17.—Interconnected Selsyn motors for remote indication.

Pneumatic telemetering systems are widely used in the oil industry where the distances involved are within its limitations and where the avoidance of the hazards of electrical systems or the cost of explosion proofing is important. A 100 per cent throttling range controller, usually especially designed for transmitter service, produces an output air pressure proportional to the measured variable. A pressure recorder, likewise usually especially designed for receiver service, scaled like the transmitter, located up to 1000 ft. or more distant, and connected by $\frac{3}{16}$- to $\frac{5}{16}$-in. bore copper tubing, receives the transmitted pressure. Standard control mechanisms may be included in the recorder case. Pneumatic transmission is ordinarily to be preferred when pneumatic control is to be included in the receiver.

Time Control

Time or cycle operators may be used alone or as an auxiliary to some other form of control. Fundamentally, any such operator consists of some form of clock, such as a constant-speed motor, synchronous motor, or a spring-wound clock, arranged to make or break electric contacts, to open or close air valves, or to change the setting of a pilot valve according to some predetermined time schedule. Single cam operators or **timers** consist usually of a clock which rotates a time dial. The dial carries one or more adjustable devices that serve to trip the contacts. The dial may be graduated in any desired time units. Or the dial may be stationary and the clock rotate a hand which makes contact with or trips an adjustable hand. In some forms timers must be reset manually after each operation; in others the timing repeats automatically, either immediately or after a predetermined interval. The simplest single-switch types used for sign or henyard lighting are cheap, but with increased switch capacity and especially with increased flexibility and special design the cost mounts rapidly. In **pneumatic timers** small air valves are substituted for the electric contacts. As many as eight valves can be operated by a single clock-driven timing cam. Depending on the manner in which the cam is cut, the intervals between the functioning of the several valves may be varied in any desired manner. These valves serve to operate diaphragm valves, pneumatic switches, dampers, or other control apparatus and thus may start and stop an operation and perform various intermediate steps.

Cycle controllers or **time-set controllers** contain a measuring element and a control system as well as a timing element. A time-pressure controller, for example, consists of a standard electric or pneumatic pressure controller the control point of which is set mechanically by the follower on a time cam cut to the desired time-pressure program. Time-humidity controllers used in program drying contain separate cams for the wet-bulb and dry-bulb programs, setting separate temperature controllers in the same case. The cams of recording time controllers are usually mounted in a separate second case. The cams may then rotate at the most suitable speed, usually other than the standard record-chart speeds.

Multiple cam operators or cycle timers are used where the switch or valve capacity or the complexity of the operating cycle is beyond the capabilities of single cam units. Since there is no limit to the number of operations which may be timed on a multiple cam unit, complex processes can be operated entirely automatically. Timer operation is frequently cascaded, one master timer operating other timers or time controllers.

Elapsed time controllers, widely used in cooking and dyeing operations, are controllers which contain also a mechanism to time their operation and turn them off after the set elapsed time period. A signal then indicates that the operation is completed.

PRESSURE AND VACUUM*

Devices for measuring pressure assume great importance in any study of control mechanisms, not only because of the innumerable occasions when pressure is a factor in industrial work, but also because of the many applications of these devices in other fields. Liquid level may be determined by the weight or pressure of a column, flow by the pressures before and after an

---

* See Sec. 6, pp. 780–787, and Sec. 19, p. 2186.

orifice placed in the line, and temperature by the pressure set up in a closed system due to the tendency of fluids to expand with increased temperature.

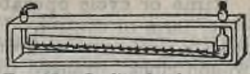


Fig. 18.—Inclined-tube pressure indicator.

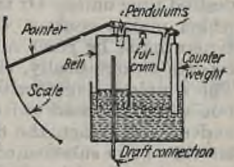


Fig. 19.—Bailey pointer gage.

Pressure may be defined as the force that is applied to, or distributed over, a surface. It is measured as the force per unit area. Practically all devices for measuring pressure utilize some form of sensitive element which is caused to move by the force. As will be seen from later descriptions, this movement of the sensitive element is made against the pressure of the surrounding atmosphere. Hence the pressure indicated by the element will be the resultant or difference between the two. Common usage has therefore taken the normal atmospheric pressure of 14.70 lb. per sq. in. as the datum line, with forces greater than this termed **gage pressure** and those below called **vacuum.** Pressure measured above zero pressure as the datum line is termed **absolute pressure.** The units most frequently used in this country are pounds per square inch pressure and inches head of mercury vacuum.

The simplest form of pressure indicator is the **liquid column,** as in mercury barometers and U-tubes (see pp. 780–787). The latter are extensively used for low gas pressures. They may be filled with kerosene, water, tetrabromethane, or mercury, depending on the range desired. **Direct-reading manometers** are constructed with large area wells in comparison with the area of the tube. The scales are specially compensated for the fall of liquid in the well so that they are direct reading. Pressures up to 2000 lb. per sq. in. are measured by direct-reading manometers. A variation is the **inclined tube** for measuring furnace draft and other slight pressures. One leg of the U-tube makes an angle of 5 to 10 deg. with the horizontal so that a 7- to 10-in. movement is obtained for pressure changes of 1 in. head of water (Fig. 18). Furnace draft is also indicated by **pointer gages** in which connection is made from the stack to the inside of a counterbalanced bell immersed in liquid. The **Bailey** gage is shown in Fig. 19. With an increase in draft the liquid within the bell will rise and the bell will be pulled down until the force is balanced against the pull of the pendulum weights which are fixed on the balance beam. The **Ellison** gage uses a linkage which is so designed that the pointer moves over a flat rather than a curved scale. Several pointer-gage elements may be mounted side by side in one case for compactness or combined with other instrument movements.

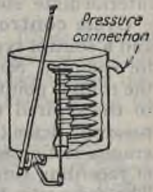


Fig. 20.—Box-type pressure element.

The most commonly used pressure-indicating devices employ some type of tube, diaphragm, or bellows, which undergoes elastic deformation with pressure variation. These are known as **pressure gages,** sometimes called **secondary gages** since they must be calibrated by comparison with a primary gage such as the manometer or dead-weight gage.

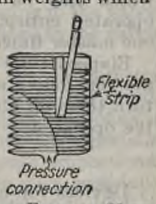


Fig. 21.—Bellows pressure element. (*Bristol.*)

Most elements designed for recording vacuum, and pressures below 15 lb. per sq. in. depend on the expansion or contraction of a bellows. Two developments from this early element are shown in Figs. 20 and 22. In the **box type** element (Fig. 20) the bellows is encased in a box to which the pressure is admitted. The compression actuates the pen arm through the multiplying linkage. This element is used for ranges of 0 to 12 in. head of water to 0 to 15 lb. per sq. in. pressure or vacuum. Overloads of 100 per cent do not damage it. An early form is shown in Fig. 21. Pressure is admitted to the inside of the bellows, and, since one side is bound by a flexible strip, the tendency to expand results in a movement of the pointer through an arc to the right (with vacuum it is to the left). In the bellows unit shown in Fig. 22, only a few diaphragms need be used. This type can be constructed to withstand pressures as high as 30 lb. per sq. in. and as low as several inches head of water.

FIG. 22.—Multiple diaphragm bellows gage element.

Slack-diaphragm gages are used for low-pressure and draft measurement. These units are well adapted for multiple mounting in a single case, up to 12 indicators being available in one case. Diaphragms are generally made of leather or oiled silk for the slack-diaphragm type.

FIG. 23.—Bourdon-tube pressure indicator.

For higher pressure indicators and for some recorders the **Bourdon tube** is used. Diaphragm gages are available but, owing to the limited motion produced, are used only where the corrosive properties of the fluid require construction of some material not suited for use in Bourdon tubes. The Bourdon tube is a curved tube whose cross section is elliptical. It is sealed on one end. When pressure is admitted, the tube tends to straighten out due to the greater total pressure on the outer side, and, conversely, a vacuum causes it to coil up. The movement is transmitted through a linkage and amplified by a rack and pinion to actuate the pointer, as shown in Fig. 23. The shape, material, and wall thickness of the tube are determined by the pressures encountered. For ranges up to about 1000 lb. per sq. in. seamless, drawn bronze tubing is generally used. Higher pressures and corrosive atmospheric conditions require the use of alloy-steel tubes which are drilled from solid stock. For severe vibration, pulsating pressures, or self-draining ability the element may be constructed of two short Bourdon springs which oppose each other. Good practice calls for the selection of a scale range about double the normal operating pressure of these gages. Periodic checking is advisable.

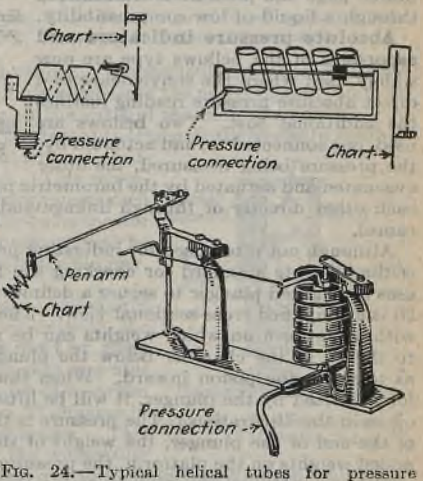


FIG. 24.—Typical helical tubes for pressure measurement.

There have been many developments of the Bourdon tube. One of these, the **helical,** is the element now largely employed in pressure recorders. It is so named because the tube is coiled in helical form. The pen arm may be soldered directly to the element, a center post may be used, or a linkage may be employed, as in Fig. 24. The effective length of the tube is from the base to the point at which soldered connection is made to the pointer or linkage. This point is varied when calibrating to get the proper deflection. Owing to the great length of the helical coil and the small arc through which the pen arm moves, these elements rarely require any multiplying with attendant angularity errors, although multiplying may easily be accomplished with the linkage type. The wall thickness, shape, and metal used in the tubes are varied to suit conditions and ranges. Great care is taken to get tubes of uniform wall thickness throughout their length, else the deflections are not uniform. Careful annealing or aging is required before calibration to prevent later changes. These elements can withstand 50 to 100 per cent overload without damage on ranges below about 0 to 1000 lb. per sq in., but greater care in overloading must be used on the higher ranges. Multiple pen recorders can be fabricated by using linkage-type elements.

The Taylor **volumetric pressure gage** is designed specifically for conditions where it is impractical to allow the fluid to flow into the pressure gage element as in homogenizers. It uses a compressible bulb, containing mercury, which is connected to a Bourdon gage by means of tubing. The tube is specially compensated so that temperature variation does not affect the pressure. A separatory diaphragm is used to separate the Bourdon tube in the Marsh gage and pressure is transmitted through a liquid of low compressibility.

Absolute pressure indicators and recorders of the bellows type are now widely used where the convenience of a direct absolute pressure reading justifies the additional cost. Two bellows are used, one connected to, and actuated by, the pressure being measured, the other evacuated and actuated by the barometric pressure. The two bellows oppose each other directly or through linkage and atmospheric pressure variations cancel.

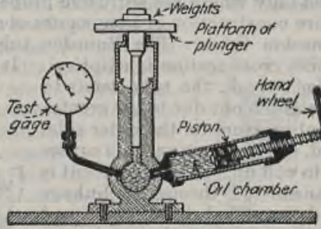


FIG. 25.—Dead-weight gage tester.

Although not often used for indicating pressures, the **dead-weight tester** is the absolute standard for checking the higher range gages. This device uses a weighted plunger to secure a definite oil pressure in a system. Figure 25 is a simplified cross-sectional view. The plunger is fitted at its upper end with a platform on which weights can be placed. Pressure can be applied to the oil in the chamber below the plunger by turning the handwheel so as to move the piston inward. When the oil pressure is greater than the force exerted by the plunger, it will be lifted off its seat and will ride on the oil as in the illustration. The pressure is then constant. Knowing the area of the end of the plunger, the weight of the plunger and platform, and the added weights on the platform, the pressure in pounds per square inch can be calculated. Hence a gage connected in the system can be checked at various points on the scale by placing appropriate weights on the platform. To prevent errors due to friction on the plunger, the tester must be level and the platform spun slowly while the reading is being taken.

For determining very high pressure, **dead-weight gages** use a suspended loading platform of great capacity, or a scale beam system. The plunger is oscillated by a motor. These gages are made for pressures as high as 50,000 lb. per sq. in. Pressure intensifiers, available for use in conjunction with dead-weight gage testers, will also develop extremely high pressure.

The dead-weight gage principle is utilized in the **Bailey power-type pressure gage** illustrated in Fig. 26. The pressure to be measured is applied to the plunger which carries on its upper end a cup containing mercury. It is constantly rotated by a small motor which also drives an oil pump, furnishing oil to a pilot valve mounted on the plunger. A slight rise or fall of the measuring plunger will allow oil to pass to the top or bottom, respectively, of the power piston in an adjacent cylinder, forcing it downward or upward. Movements of this piston cause corresponding motion of the displacer in the mercury, and thus increase or decrease the loading of the plunger, resulting in a shifting of its position so as to shut off the oil supply. Consequently, for each pressure applied to the plunger, there is a definite position of the power piston. A rack mounted on it turns a pinion gear on the shaft of the indicating device. A Selsyn motor (see p. 2026) is shown connected to transmit the pressure reading to a distant indicator or recorder but a direct-reading instrument may be substituted. This movement lends itself particularly to narrow ranges such as 385 to 415 lb. per sq. in. for a boiler-room master gage. Practically any maximum pressure may be secured by varying the relative sizes of the plunger, mercury cup, and displacer.

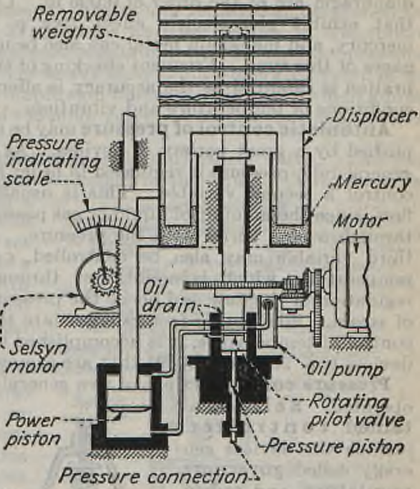


FIG. 26.—Bailey "power-type" pressure gage arranged for remote transmission.

Measurements of high pressures, particularly rapidly fluctuating ones, can be made with the **McCullom-Peters electric telemeter gage** (Baldwin-Southwark Corp.). This gage operates by reason of the changes in resistance of a carbon pile with the force applied. In Fig. 27 changes in pressure cause a flexing of the diaphragm. This movement is transmitted through the connecting rod and cantilever spring to the carbon resistors, compressing one and relieving the other. The resistors are composed of a number of annular sections of carbon, $\frac{7}{16}$-in. outside diameter and 0.020 in. thick, placed between copper disks. Connection is made from the disks to a storage battery as shown in Fig. 28,

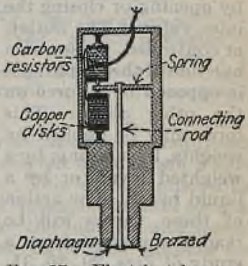


FIG. 27.—Electric telemeter gage. (*McCullom-Peters.*)

with the current regulated by means of a variable resistance and voltmeter. The fluctuations in current are measured with a milliammeter, indicating or recording; or with an oscillograph, calibrated to read directly in pressure units. Movements of the diaphragm are of the order of 0.005 in. Crystals that exhibit piezoelectric effect (see p. 2653), mercury, and manganin metal can also be used for gages of this type. Frequent checking of the calibration is essential as the accuracy is affected by conditions of temperature and vibration.

FIG. 28.—Electrical diagram for telemeter gage.

Automatic control of pressure may be accomplished by a great variety of devices. As a general rule, pressure is regulated in order to control a second variable. This is usually flow, since the amount of liquid or gas passed through a pipe varies with the pressure. A third variable may also be controlled, *e.g.* temperature, which is maintained through regulation of the flow and hence the pressure of steam. Since other factors enter into the control of temperature, it is accomplished by devices (pp. 2019 and 2056) that are somewhat different in construction.

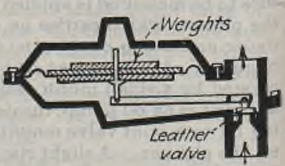


FIG. 29.—Low-pressure governor.

Pressure controllers are of two general classes, self-contained and power operated. **Self-contained controllers** include the devices generally called **governors, regulators,** and **reducing valves.** They have no indicating or recording means; they effect corrections mechanically as the result of changes in the position of a diaphragm by opening or closing the inlet valve. The outlet, or controlled, pressure on one side of the diaphragm is opposed by a force on the other side. This force may be applied by weights, by a spring, by a weighted lever, or by a liquid head. The action of these valves will be readily apparent from a study of Figs. 29, 30, 31, and 32. The first type is used for low pressures, as in city gas lines. On this service no adjustment is needed. If a change of pressure is necessary, the head may be removed and weights added or removed. Where the control point must be altered

FIG. 30.—Spring-operated pressure reducing valve.

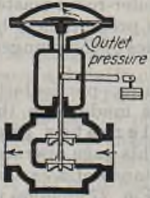


FIG. 31.—Weighted-lever pressure regulator.

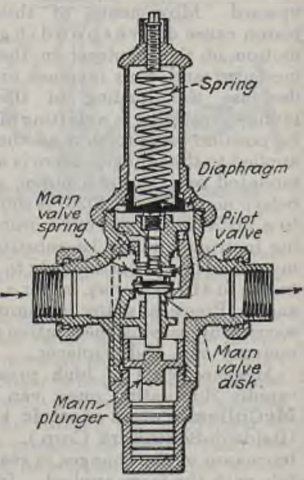


FIG. 32.—Pilot-operated pressure regulator, spring type.

frequently, use is made of a governor equipped with an extension of the loading shaft through the head, so that weights may be added or removed easily. A mercury-sealed vent may be provided as a safety blowoff. The regulator in Fig. 30 may be adjusted by turning the screw at the top so as to increase or relieve the spring tension. Addition or removal of weights on the weighted-lever regulator (Fig. 31) permits an easy change in the control point. The spring and weighted-lever regulators may be used for pressures up to about 200 lb. per sq. in. For heavy duty a pilot-operated valve of the spring type is often used. The diaphragm moves a pilot valve which admits the inlet pressure to the piston of the main valve. The principle is illustrated in Fig. 32.

The **Mercon pressure regulator** is a liquid-loaded valve. The diaphragm load is varied by changing the height of the liquid. This valve has the advantage of always having a uniform effective diaphragm area on both sides of the diaphragm regardless of load. The **Grove** regulator loads the diaphragm with air or gas pressure in a dome, which is set to equal the desired controlled pressure.

Power-operated controllers utilize external fluid or electric power to operate the valve or damper, the mechanism merely controlling the application of this power. Such controllers are more accurate and more powerful than the self-actuated type. For many applications the indicating or recording-instrument type of control described previously (pp. 2019–2021) is preferred, but elsewhere, when an indication or record is not required, the power-type regulator is commonly used. These may have open-and-shut, throttling, and throttling reset (load-compensating action) just as instrument control, with plain throttling control most common.

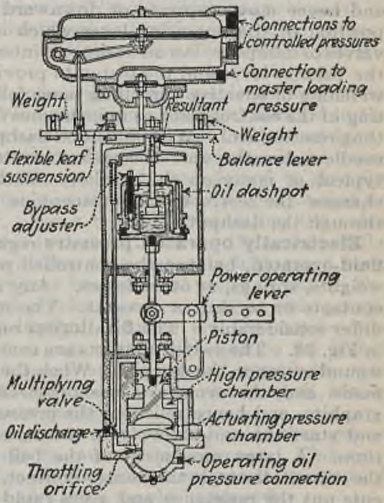


FIG. 33.—Smoot pressure controller, showing fluid-operated compensating control.

Load compensating pressure controllers were developed mainly for **combustion control** in boiler plants where it is desired to maintain a constant steam pressure despite a varying steam flow. This is accomplished by effecting changes in the rate of heat generation and consequent steam generation. The variables which must be controlled are the induced and forced draft and the fuel, or stoker, feed. These devices have been adapted for the control of gas pressures, for mixing gases in definite proportions, for operating water-gas sets, and for controlling chemical operations. Basically most of the available designs are scale beams which balance the controlled pressure against weights or other set forces. Any unbalance acts to correct itself but is compensated to prevent "hunting" or overshooting. Frequently, especially in boiler rooms, a number of units are employed, and all are held to the correct setting by a master controller. This is so arranged that the individual units may be separately adjusted, *e.g.*, the ratio of fuel to air on any one boiler or the portion of the load carried by any one boiler.

As with other pressure controls there are two kinds of transmission, fluid and electric. An example of the **fluid-operated compensated control** is the **Smoot** shown in Fig. 33. A set air pressure, or the loading pressure from the master controller, and the controlled pressure or pressures acting upon their respective diaphragms create forces which are exerted on the balance lever. The resultant moves the multiplying valve which controls the oil pressure under the piston of the regulator. Oil is supplied by a small pump at 10 lb. per sq. in. pressure. Air or steam may be used instead. The piston is acted upon by a constant pressure above and a varying pressure beneath it and hence moves upward or downward and shifts the position of the operating lever which is connected to valves or dampers. An oil dashpot is interposed between the balance lever and the piston to provide momentary widening of effective throttling range followed by resetting of the control point as in instrument types of throttling reset control. Without the oil dashpot and by-pass needle valve the unit becomes a throttling controller typical of many on the market. With sudden large changes the corrective movement is made directly through the dashpot.

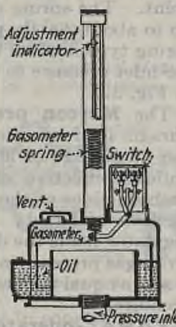


FIG. 34.—Shallcross floating-bell, low-pressure controller.

Electrically operated pressure regulators, like the fluid-operated, balance the controlled pressure against weights, springs, or other forces. Any unbalance is used to make and break contacts or to shift a rheostat. The methods of securing throttling action differ considerably. The **Shallcross** regulator for low pressures is illustrated in Fig. 34. The switch contacts are connected through resistances to a shunt-wound motor on the damper. When the pressure is normal, both contacts are made and the armature is short-circuited. The contact surfaces are of graphite, and hence relieving the pressure slightly will change the resistance and start the motor slowly to restore equilibrium. A large movement of the bell makes the circuit through the center contact, which cuts out the resistance and gives rapid movement. The **Leeds and Northrup** regulator employs an interrupter in the circuit to get step-by-step operation. It uses counterbalanced bells for low pressures, making contact to a reversible motor; and a Bourdon tube actuating the contact arm of a rheostat, for higher pressures. Reversible motor drive is used also on the **Brooke** regulator (Fig. 35). The motor, in turning the drive wheel for the valve or damper which will correct the unbalance, also moves a compensating lever so as to break the contacts and stop itself almost immediately, thereby getting an "inching" motion. The **Bailey** electric control uses a Bourdon tube with a long pointer arm on which a pin is mounted. A motor-driven plate pushes this pin against two stepped wedges. Its exact position on the wedges, due to the pressure, determines whether it will move the "more" or "less" wedge and how far it will move it. On this depend the period of contact and length of time the control motor will run to correct conditions.

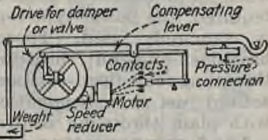


FIG. 35.—Brooke pressure controller.

General Considerations. Conditions and layout determine the exact method of connection of a pressure-sensitive mechanism to any system. It is good practice to have a valve in the line below a gage so that it may be disconnected at any time. When it is necessary to locate a gage at a different level than the connection, allowance must be made on the readings for the pressure of any column of liquid above or below the gage. On steam lines, condensate will form and cause incorrect readings. Gages may be secured with a set-back zero, but they are not generally recommended because of the difficulty of checking.

Gage elements should be protected from high temperatures and corrosive materials if maximum service is to be obtained. A coil-pipe siphon on steam lines will collect the condensate and keep steam from reaching the element. Steel elements are generally specified for ammonia and alkali lines. Other corrosive gases and liquids can be prevented from attacking gage elements by means of oil seals so designed that an inert sealing liquid is interposed between the element and the pressure system. Carbon tetrachloride and mineral oils are most frequently used.

Rapidly fluctuating pressures are ordinarily objectionable as they make readings difficult. A "deadener" of fine tubing in a long coil inserted in the line is very effective, or devices known as "snubbers" and "pulsation dampers" can be used. In the **Ray pressure snubber** the restriction is the annular passage formed by a hole containing a wire of size selected to give the desired restriction. The wire can move with pressure changes tending to prevent plugging by foreign matter. For a temporary condition the valve in the line may be partly closed. If it is desired to record the extent of the fluctuations as a clue to the causes of the disturbance, a high-speed strip chart or a rapidly turning round chart may be employed with any gage. For extreme conditions the McCullom-Peters telemeter (*q.v.*) is more satisfactory.

All gages should be checked periodically. U-tubes are the standard on the lower ranges, and an indicating test gage is generally used on the higher ranges. A three-way cock mounted below each gage will permit an easy connection of the test gage. The absolute standard for checking all higher ranges is the dead-weight tester already described.

It is well to remember in planning piping systems for liquids that valves of the quick-opening variety are likely to cause water hammer. This may easily ruin any pressure gage in the line, owing to the momentary overloads of 500 per cent or more. The action of all valves should be smooth.

FLUID FLOW

Measurement of flowing fluids can be divided into two broad classifications, quantity or positive meters and rate of flow or inferential meters. In the first type the flowing fluid is split into definite weights or volumes by the primary element, and the total numbers of such definite volumes or weights are totalized by some types of counter. The second type measures the flow of a continuous stream, the primary element indicating the instantaneous value of the flow rate. Quantity meters can be further subdivided into weighing meters and volume meters. Rate meters generally used for industrial purposes employ (1) differential pressure methods; (2) flow through a weir; (3) area methods; (4) calorimetric methods; (5) dilution methods; and (6) impulse or turbine methods.

Weighing meters are used for measuring flow of liquids, especially where grit or mud would interfere with the operation of the volumetric type. The principle of operation is illustrated by the Cochrane weighing meter,

Fig. 36. Liquid flowing into the bucket is weighed by means of the weight placed on a scale beam. When the weight of liquid and bucket reaches a certain value sufficient to counterbalance the weight, the bucket sinks and releases the latch which permits emptying. During the emptying period, inflow passes into a detaining chamber. The number of tiltings is transmitted to a counter. Recording is also possible—each tilt making a step on a chart for 10 steps, when the direction of the recording pen reverses to make 10 steps in the opposite direction.

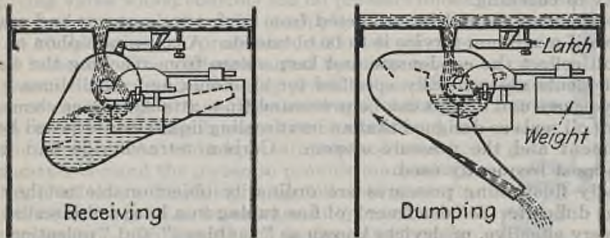


Fig. 36.—Cochrane weighing meter.

Volumetric meters for liquids are of either the gravitational or displacement type. Gravitational meters consist of two types. The **tank type** depends upon the filling of a tank or tanks which empty, when a certain volume has been attained, by tilting, float valves, or auxiliary power devices operated by float valves. Each emptying cycle operates a counter or recorder. The **rotary type** consists of a drum divided into several sections as shown in Fig. 37. Water, admitted at the center, flows into the section immediately below it. As water fills the section, the center of gravity shifts, causing the drum to rotate. The drum is connected to a counter which registers the quantity passed through.

Liquid-displacement meters are used principally for measuring water. For less than 1000 gal. water per minute, **disk meters** (see p. 866) are very common. Larger flows sometimes are measured with a combination disk- and-turbine meter, the former handling large flow.

Piston meters are used for accurate measurement of flow, particularly for small capacities. In these meters a piston is forced back and forth, tripping a valve and reversing the flow as it reaches the end.

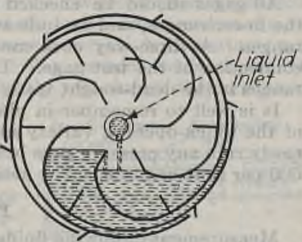


Fig. 37.—Rotary type of liquid volumetric meter.

Gas meters are of three principle types: (1) **wet gas meters** (see p. 865), (2) **dry gas meters** (see p. 865), of the diaphragm type, and (3) **cycloidal meters** (see p. 2269) which are very similar to a two-lobed cycloidal blower of the Roots type.

Certain types have, in addition to an integrator, a record chart on which temperature and pressure are recorded against flow volume, *i.e.*, rotation of the chart is proportional to volume of gas flow rather than time. A time pen may also be added as well as a differential pen showing pressure drop across the meter.

Calibrated gas holders or "provers" of the ordinary water-sealed type are sometimes used for measuring gas, particularly for checking meters.

For greatest accuracy with any of the foregoing meters, it is necessary to correct the volume for deviation from standard pressure and temperature. It should be noted that a variation of about 4-in. water pressure or 5°F. will cause a change of about 1 per cent in volume of standard gas.

Differential-pressure Methods. Bernoulli's theorem (see p. 800) is a corollary of the law of conservation of energy which states that the total energy of unit weight of a continuous mass of fluid passing through an isolated system remains constant from point to point along its line of flow, without regard to what may be done with the fluid *within* the system. An isolated system precludes the addition of energy from the outside or the removal of energy to the outside. Hence, except for changes due to friction or volume variation which change the heat content of the system, the total mechanical energy of the fluid will remain constant from point to point. Since variation in the heat content can be compensated or made practically negligible, these facts form the basis of two methods of flow measurement. In the first the velocity of the fluid is changed momentarily as it passes through a constriction, and the resulting change in static head is taken as a measure of the velocity and hence of the volume rate of flow. The devices for producing this change in static head are as follows: (1) **orifices and flow nozzles**; and (2) **venturi tubes** (see p. 843). In the second method the velocity head is measured directly, without a change in velocity, by means of the **pitot tube** (see p. 836), and this measurement is converted into velocity and volume rate of flow. **Orifices** (see p. 842) are the most commonly used differential-pressure methods of flow measurement. Of the several possible types of orifice only two, the **sharp-edged orifice** in thin plate and the **flow nozzle**, are sufficiently reproducible for general use (see p. 843). The sharp-edged orifice (Fig. 38) is easiest to reproduce accurately, is least expensive, and therefore is most used. This device is most often a circular hole in a metal diaphragm $\frac{1}{32}$ to $\frac{1}{16}$ in. (preferably the lower) thick (Fig. 38*a*), placed concentrically between flanges in the flow pipe. Occasionally it is preferable to use an offset orifice of either the eccentric (Fig. 38*b*) or segmental type (Fig. 38*c*). These orifices are useful where the fluid contains solids that might deposit against the upstream side of a concentric orifice and change its characteristics. To avoid this difficulty, one or the other may be used with the unobstructed side down so as to prevent solids from accumulating. For conditions of widely varying flow, adjustable orifices of the segmental type can be used. Figure 39 illustrates such an orifice. The orifice height can be adjusted within 0.001 in. It is used exactly as the segmental-type orifice. The Bailey Meter Co. also makes an automatic

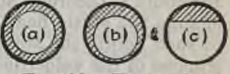


FIG. 38.—Types of orifice. (*a*) Concentric; (*b*) eccentric; (*c*) segmental.

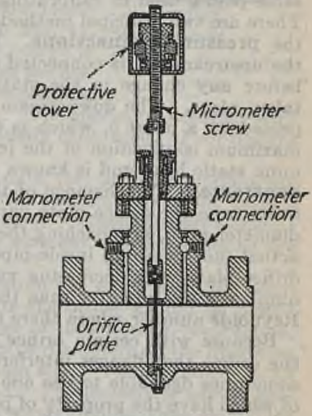


FIG. 39.—Bailey adjustable orifice.

adjustable orifice of this type which shifts automatically with major changes of flow rate.

A multiple-orifice is made by the Meriam Co. which has four different sized orifices, any one of which can be revolved into place by loosening the gaskets of the orifice plate.

The action of a concentric orifice may be observed in Fig. 40. Sudden constriction of the flow causes the jet to contract below the orifice, reducing the flow to considerably less than the theoretical. In accordance with Bernoulli's theorem any change in the velocity through the orifice must change the static head at points on either side of the orifice. For any pair of such points the static-pressure difference will vary as the difference of the squares of the velocities at and before constriction, and it is this fact which is used in determining the flow rate. But there is also a static-pressure variation from point to point on either side of the orifice. An indication of the character of this variation is given by the vertical lines in Fig. 40. It will be noted that the differential pressure h across the orifice will vary slightly depending on the points chosen for the pressure connections. The pressure connections must therefore be made for any orifice at the same points used in calibrating the orifice. There are two principal methods of locating the **pressure connections.** In the first the upstream tap is connected at a point a before any change in the static head has taken place. The downstream tap is connected at a point b, which is the point of maximum contraction of the jet and minimum static head and is known as the **vena contracta.** The distance of the *vena contracta* from the orifice plate varies with the diameter ratio, approaching the orifice plate as the ratio of orifice diameter to actual (not nominal) inside-pipe diameter increases. It also approaches the orifice plate with decreasing values of Reynolds number, but the Reynolds number is less effective than the diameter ratio, except at very low values of Reynolds number where there is a viscosity influence.

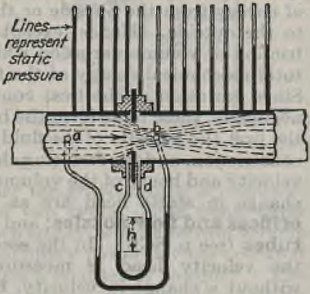


FIG. 40.—Concentric orifice, showing methods of *vena contracta* and flange connection and indicating variations in static pressure as a function of the distance from the orifice.

Because with certain orifice ratios the *vena contracta* may be so close to the orifice that flanges interfere with the location of the pressure tap, it is sometimes desirable to use one of the offset orifices (Fig. 38, b and c), both of which have the property of pushing the *vena contracta* farther downstream.

The second method of connecting the pressure taps, called **flange connection** (Fig. 40, c and d) locates the two taps in the flanges adjacent to the two sides of the orifice plate. The two methods give slightly different pressure differentials for the same orifice and flow. The differences are taken into account in the determination of the orifice coefficients. A third method occasionally used is to connect the upstream tap 2½ pipe diameters above the orifice, and the downstream tap 8 diameters below the orifice. Although this method permits considerable tolerance in tap location, the differential in this case is considerably smaller than that obtained by either of the two foregoing methods, and pipe surface has a greater effect on discharge coefficients, creating inaccuracies.

It was noted above in connection with Fig. 40 that the jet below an orifice contracts and reduces the flow to less than the theoretical. At the *vena contracta* the area of the jet is about 62 per cent of the orifice area. Also friction through the orifice reduces the flow 1 or 2 per cent below the theoretical. The net result is that a sharp-edged orifice will pass only about 61 per cent of the theoretical quantity of fluid. This value, 0.61, is called the **discharge coefficient.** Its exact value varies slightly, depending on the ratio of orifice diameter to internal pipe diameter and the Reynolds number, so it must be determined experimentally. In practice, the discharge coefficient may represent more than simply the percentage of theoretical flow which an orifice will pass. It is also used to compensate for variation in the location of the pressure taps and frequently includes a quantity known as the "velocity-of-approach factor." These additional functions account in part for the variations in coefficient curves which are published by different investigators.

Flow Nozzles. For very high flow velocity an ordinary orifice will not be sufficiently accurate. An orifice ratio, d/D, of more than 0.6 to 0.7 is usually not recommended, because inequalities in the pipe may give rise to inaccuracies. Where the flow is very high, this difficulty may be surmounted by using a well-rounded orifice or flow nozzle which, for the same differential, is capable of passing considerably more fluid than a plate orifice of the same diameter. A flow nozzle is a rounded, funnel-like aperture similar to Fig. 41. Because the stream is constricted gradually, there is little or no contraction below the nozzle. Hence the discharge coefficient will range from 0.97 to 0.99 depending on smoothness and the accuracy of the design. For maximum accuracy each different design of flow nozzle must be individually calibrated. Poor designs may have coefficients as low as 0.90.

FIG. 41.—Typical flow nozzle.

Orifice Equations. Derivation of, and data for, orifice equations are given in the A.S.M.E. Fluid Meter report, in the publications of the various meter manufacturers, and in Sec. 6 of this handbook. When primary elements are purchased alone or with the secondary element, the calculations are undertaken by the maker.

Installation of Orifices. Careless installation can easily spoil the accuracy of a good orifice meter. For best results the orifice should be placed in a long, straight run of pipe with no change of direction or obstruction, such as a valve or fitting, closer than 10 to 50 pipe diameters on the upstream side; or 1 or 2 pipe diameters on the downstream side. It may, however, be placed somewhat closer to a large-radius bend. Particularly if curves preceding the orifice are not all in one plane, a straightening vane should be placed in front of the orifice not less than 6 pipe diameters upstream. Such vanes cut the pipe for a short distance into a number of smaller, straight passages and tend to eliminate turbulence and swirling. The Natural Gas Association recommends the use of straightening vanes in gas metering where any disturbance takes place less than 100 diameters before the meter, or where a pressure regulator precedes the meter by less than 200 diameters. Pulsating flow particularly must be avoided. Orifices must be kept clean and sharp. If solids build up around the upstream periphery, the flow will be increased and the accuracy badly impaired. Orifice fittings are available with which it is not necessary to shut down the line when orifice plates are removed to be cleaned or changed. One such installation, the Byron Jackson orifice fitting, utilizes two orifice plates. To remove an orifice plate from the line, a crank shaft is turned 180 deg., which lifts the plate through the crank arrangement into an upper chamber. A shutoff valve is then lowered, permitting a cap to be removed.

Static-pressure-drop Recovery. Not all the orifice pressure drop is lost. In high-pressure metering, the permanent static-pressure loss is not usually important. The reverse may be true in low-pressure work. The recovery (*i.e.*, the differential reconverted to static pressure at a distance 4 to 5 diameters downstream) will vary between 5 to 30 per cent of the differential drop for the most frequently used d/D ratios (see Fig. 45, p. 2042). Where this loss is greater than permissible, a flow nozzle or a venturi tube may be substituted for the orifice. With large ratio nozzles, recovery up to about 65 per cent of the loss is possible. With a venturi, up to 90 per cent of the differential pressure may be recovered.

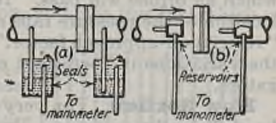


FIG. 42.—Methods of sealing orifice meters.

Seals and Purges for Orifice Meters. Where the fluid being metered will corrode the manometer or foul the manometer liquid, it is customary to provide a liquid seal in each pressure line from orifice to manometer, as shown in Fig. 42*a*.

If a condensable vapor such as steam is being metered, it is necessary to provide small reservoirs above the meters, Fig. 42*b*, so that a substantially equal head of the condensed fluid will be maintained in each connection to the manometer. These also serve as condensers quickly to replace any liquid lost through differential changes. Seals are also used when measuring liquids that will freeze, to make unnecessary the heating of the meter body and connecting lines.

Purge systems are sometimes used instead of seals to protect a meter. The piping is arranged so that a small flow of inert gas or liquid, too small to upset the flow measurement, prevents the live liquid entering the meter and piping. Air, steam, water, and oil are common purge fluids.

Orifice Meters. Orifice meters consist of an orifice and a differential-pressure measuring device which indicates or records the flow directly or indirectly. Differential measuring devices are of three types: (1) manometers, (2) liquid-sealed bells, and (3) mechanical diaphragm gages. Manometers are most generally used. For indicating meters, the manometer is usually a U-tube of glass or a single-tube manometer containing mercury, tetrabromethane, water, oil, or other liquid depending on the differential. For low ranges the tube may be inclined to spread out the graduations. Standard recorder mechanisms are also used.

Recorders use all three types of differential-measuring mechanisms mentioned above. A mercury-filled U-tube, in which an iron float rides on the mercury in one branch of the U, is most common. Motion of the float is transmitted to the recorder pen by mechanical or electrical means. Recording may be either in terms of flow or in terms of the differential pressure. Mechanical and U-tube recorders are used for measuring differentials ranging from about 0.1 in. water upward. Since flow varies as the square root of the differential, flow graduated meters either must use non-uniformly divided charts or must use some device for converting to obtain the square-root relation.

The most common type of mechanical flowmeter is the manometer and float type. Figure 43 illustrates the **Foxboro manometer** in which the float motion is brought out by a small shaft lapped to so close a fit with its bearing that a film of grease seals it for operating pressures up to 5000 lb. per sq in. The end thrust is taken on a jeweled bearing. Similar construction is offered by Brown, Emco, Metric, and Republic, and by Bailey and Builders Iron Foundry with a stuffing box in place of a lapped bearing. The **Tagliabue manometer** uses a pair of permanent magnets to bring out the float motion. One magnet turned by the float inside a non-magnetic cylinder uses a second magnet external to the cylinder to follow it and to move the connected pen. The **Cochrane manometer** shown in Fig. 44 is suspended on knife-edges and differential pressure is communicated through flexible tubes. Changes in mercury level cause the entire manometer to tilt against the pull of the weight. Oil- and mercury-sealed **Cochrane** and **Republic ring manometers** operate similarly.

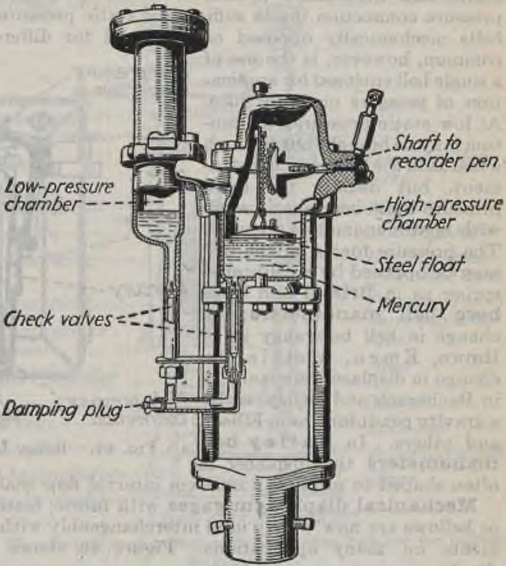


FIG. 43.—Foxboro mercury manometer mechanism.

The manometer shown in Fig. 43 has a uniform-interval differential scale or a square-root flow scale. The chain and sector mechanism, like an equivalent lever device in the Emco manometer, avoids angularity errors in the differential scale. In the Foxboro, as in the Tagliabue manometer, by substituting a paraboloid-shaped range chamber for the cylindrical one, a uniform-interval flow scale from about 10 per cent upward is secured at higher first cost and with the necessity for precise leveling and filling. Cams are used in Cochrane meters to produce manometer tilt directly proportional to flow, in **Republic ring meters** to convert manometer tilt, and in **Builders Iron Foundry meters** to translate float motion into uniform flow intervals.

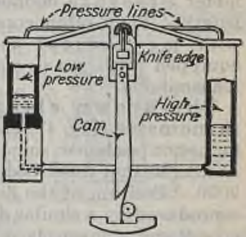


FIG. 44.—Cochrane mercury manometer.

Bell manometers are ordinarily used for low differential ranges below the practical limit for float instruments. Nevertheless, one of the best-known bell manometers is the Bailey instrument, Fig. 45. The **Ledoux floating bell** used by Bailey and Simplex is so shaped in effective area and wall section that the balancing of pressure and buoyancy forces results in bell movement proportional to flow and hence in uniform interval flow scale from zero upward.

Straight-walled bell manometers are widely used for measuring small static and differential pressures. Since a single liquid-sealed bell with pressure connection inside suffices for static pressure (Fig. 19, p. 2028), two bells mechanically opposed can be used for differential pressures. More common, however, is the use of a single bell enclosed for application of pressure on both sides. At low static pressures the motion may be brought out through a seal as in the Bacharach instrument, but usually the same pressure-tight bearings as serve with float manometers are used. The pressure forces on the bell may be opposed by a calibrated spring as in **Bristol** and **Foxboro bell manometers;** by change in bell buoyancy as in Brown, Emco, Metric; by change in displacer buoyancy as in Bacharach and Bailey; or by a gravity pendulum, as in Ellison and others. In **Bailey bell manometers** the displacer is often shaped to produce a uniform interval flow scale.

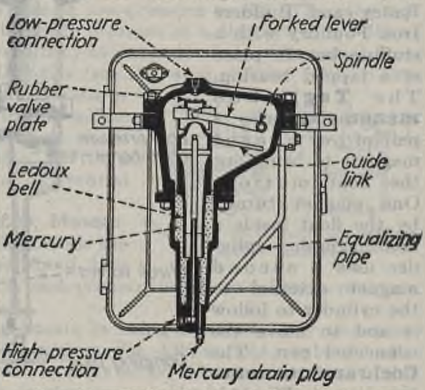


FIG. 45.—Bailey Ledoux bell manometer.

Mechanical diaphragm gages with fabric, leather, or metal diaphragms or bellows are now widely used interchangeably with liquid-sealed bell instruments on many applications. Figure 46 shows a Hays spring-opposed diaphragm unit.

Electric meters, using electrical means for determining manometer response to pressure changes, are used where remote readings are desired beyond the 100- to 300-ft. practical limit of mechanical meter pressure connection length. They often compete with mechanical meters equipped with pneumatic transmission.

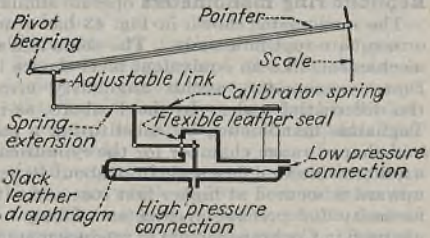


FIG. 46.—Hays differential mechanism.

The **Brown electric manometer** (Fig. 47) uses an induction balance somewhat similar to that described on p. 2026. Position of the float is reproduced by a similar divided coil and plunger placed in the receiving instrument. The **Cochrane electric manometer** is also of the induction-balance type, but does not depend upon the plunger pull at the receiving instrument to position the indicator. Instead a galvanometer is used to detect unbalance in the two inductance circuits and commands a servomotor so that balance is restored.

The **Republic manometer,** indicated schematically in Fig. 48, makes use of an ascending spiral of metal rods with which the rising mercury in the low-pressure leg of the manometer successively makes contact. The rods are connected to resistance coils so that the total resistance is proportional to the height of mercury in the manometer. By making the spiral of varying pitch, correction is made for the square-root relation between flow and differential pressure, and the chart divisions are uniform. With this instrument a modified watt-hour meter may be used as an integrator or totalizer of flow. The **Bailey electric manometer** has a primary coil of insulated wire wound on a laminated iron core, connected to a 5-volt a.c. transformer. This is placed in the low-pressure chamber, and the mercury rises around it as the differential increases. The mercury acts as the secondary winding of a transformer and admits more current to the primary as the level rises. Electrical characteristics of this manometer automatically give indications directly proportional to the flow. Indicators and recorders are operated by indicating, and integrators by totalizing, admittance meters. Electric bell manometers are offered by Brown and Republic. A tilting type manometer is used with a null-method measurement in the Centrimax made by Leeds and Northrup, Fig. 49. Manometer unbalance causes a magnetic switch to operate a motor driving a flyball governor which exerts a restoring force upon the manometer to restore balance. Since this flyball governor force is proportional to the square of the motor speed and since the manometer unbalance force is proportional to the differential or the square of the flow rate, the flow rate and motor speed are proportional to each other. Motor speed is integrated by a counter to give total flow and is measured by a tachometer to give flow rate, both values being telemetered electrically.

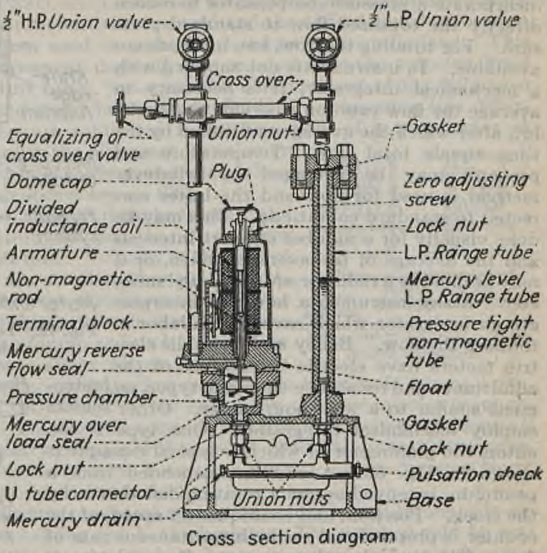


FIG. 47.—Brown manometer with induction balance for remote indication.

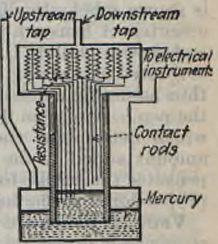


FIG. 48.—Diagrammatic representation of Republic electric manometer.

Recorder Auxiliaries. Flow recorders for gases often incorporate a static-pressure recording pen and sometimes a temperature pen. It is desirable that both should be used on industrial gas meters. This is to permit reducing the totalized flow to standard temperature and pressure. Some instruments incorporate a pressure compensator to reduce directly the totalized flow to standard pressure. For totaling the flow, two methods are available. In instruments not supplied with a mechanical integrator, it is necessary to average the flow rate for a period, usually 24 hr., after which the average multiplied by the time equals total flow. Temperature and pressure may be averaged by whatever method is used for flow and the latter corrected to standard conditions. This may be done visually for a number of short intervals and the average of the averages taken, or it may be done by a radial or square-root planimeter. Many instruments, however, incorporate an integrator which saves much labor in totaling the flow. Bailey and Republic electric meters have electric integrators of the admittance and resistance-changing types, an instrument similar to a watt-hour meter. Other meters employ mechanical integrators. One type is the automatic planimeter of which a typical example is shown in Fig. 50. A counter suspended from a pendulum is moved across a rotating disk driven by the clock. Position, and consequently speed, of the counter is proportional to the instantaneous rate of flow. The second and more generally used type is that in which at certain intervals, usually once per minute, a ratchet or clutch is caused to drive the counter a certain distance which is limited by a variable stop, set by the recorder pen. One of the simplest is shown schematically in Fig. 51. A clock-rotated eccentric *A* causes a linkage to move the arm *B* up and down. Pulley *C*, to which is attached a cam *D*, is rotated by a string *E* attached to the recorder pen, thus assuming a definite position for each position of the pen. Arm *B* in descending comes to rest on the cam *D*, which determines the length of its stroke and hence the amount added to the counter. The shape of the cam compensates the stroke for the square-root relation and makes it proportional to the instantaneous rate of flow.

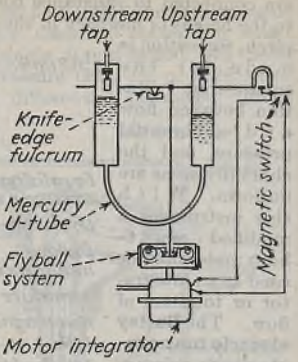


FIG. 49.—Leeds and Northrup null-method manometer.

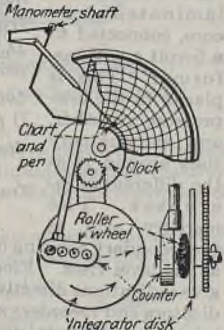


FIG. 50.—Mechanical integrator planimeter.

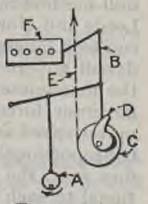


FIG. 51.—Intermittent integrator.

Venturi tubes are used in place of orifices for applications requiring high accuracy or minimum pressure loss (see p. 843).

As made by the **Builders Iron Foundry**, standard venturi tubes range in entrance diameter from 2 in., with metering capacity from 3 to 90 gal. per min., to 60 in., with metering capacity from 2780 to 81,300 gal. per min. Special meters may be built practically any size. Smaller meters are usually made of cast iron with bronze throats and larger meters of sheet metal or concrete. Various interchangeable sizes of throat are available for each standard size of venturi tube, as a means of changing the range. A venturi tube will pass substantially as much fluid as a pipe of diameter equal to the venturi inlet and outlet.

Pitot tubes (see p. 837) are much used for portable and exploratory work but, because of the low differentials produced and the difficulties of calibration and clogging, are seldom used in permanent industrial installations. This method of flow measurement depends not on changing the velocity head as in orifices and venturi tubes, but upon the measurement of the difference between the impact and static heads of the fluid. In measuring this difference, the two pressures are connected to the two sides of a manometer as in Fig. 52. The reading is the difference, or velocity head, $V^2/2g$ multiplied by a calibration factor, when the velocity can be calculated (see p. 837).

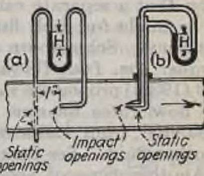


Fig. 52.—Typical pitot tubes installed.

In using a pitot tube, account must be taken of the fact that velocity within the pipe is not uniform throughout the cross section. Velocity at the wall of a pipe, when there are no disturbances (valves, fittings, etc.) within 50 pipe diameters of the pitot tube, is about one-half the center velocity. Average velocity is at about one-fourth radius from the pipe wall. Center velocity is about 20 per cent greater than the mean velocity on which the flow rate must be based. It cannot, however, be assumed that this is strictly the case when the pipe interior is rough or contains disturbing elements. In accurate work, particularly for permanent installation, it is necessary to explore the pipe at 10 to 20 points to determine the location of mean velocity or to determine the percentage decrease from center to mean velocity (velocity factor).

Industrial weir meters consist of an open or closed tank in one side of which is a partition containing the V-notch plate. The notch should be cut in thin, corrosion-resisting metal such as brass or monel, not over $\frac{1}{32}$ in. thick unless beveled on the downstream side. Edges should be sharp and square. The upstream tank should have a width not less than $7h$ (h = maximum crest height) and a depth beneath the vortex not less than $3h$. Baffles must be provided to prevent ripples. The head may be measured by means of a gage glass, hook gage, float, or by special means. According to the Bailey Meter Co., meters with a single notch for maximum head of 10 in. can handle up to 300,000 lb. water per hour and, for maximum heads of 15 in., up to 1,000,000 lb. water per hour. By using some form of float, or displacer, weir meters may be made to record. In the **Bailey meter** two cast-iron displacers, hung from the ends of a balance beam as in Fig. 53, are used to compensate for the exponential relation and to make displacement of the balance arm directly proportional to the flow. In the **Cochrane meter**, a cam is interposed between the float and the recorder pen to correct for the $\frac{5}{2}$ power relation and give an equally divided chart. Both meters are equipped with integrators.

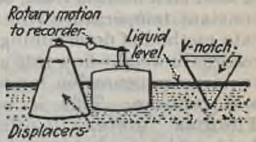


Fig. 53.—Compensating displacers of Bailey weir meter.

Area-type meters depend upon the maintenance of a fixed pressure differential across an orifice of variable area. In the **rotameter** this is accomplished by using a rotor or float inside of a tapered tube. The weight of the rotor forms a constant head or pressure differential across the annular orifice; consequently increasing flows carry the rotor upward to a position of greater annular orifice area. Since flow is proportional to the orifice area, a direct relation exists between height of rotor and flow, the calibration being an almost straight line. Rotameter tubes are made of either glass or transparent plastics. Metal rotameter tubes are available with an indicator rod attached to the rotor. A transparent sight glass is placed in front of the indicator rod. Remote indication instruments use a synchronous generator actuated by the indicator to transmit to a remote synchronous motor, or an induction-balance method.

Rotameter Calibration. One disadvantage of the rotameter has been that a separate calibration had to be made for each fluid to ensure accuracy. Schoenborn and Colburn [*Trans. Am. Inst. Chem. Engrs.*, **34**, 359 (1939)] proposed a method whereby flow rates on one fluid can be predicted from calibration data on another fluid. See Section 6.

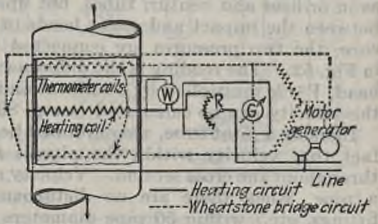


FIG. 54.—Diagrammatic arrangement of Thomas calorimetric gas meter.

Another type of area meter, also dependent upon the maintenance of a fixed pressure differential across an orifice of variable area, is the **Isometer** made by the Elgin Softener Corp. Downstream and upstream pressures are exerted on either side of a piston, which also serves as a slide pilot valve. A weight is attached to the pilot valve to maintain a constant pressure differential. The pilot valve admits fluid pressure to the piston which regulates the height of the orifice.

In the **Brown and Cochrane area meters** the rise and fall of a weighted plunger uncovering more or less port area is transmitted by the inductance-bridge devices of those makers.

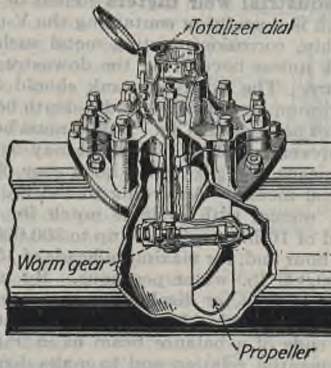


FIG. 55.—Sparling turbine meter.

Calorimetric Flow Measurement. Adding a measured quantity of heat to a flowing fluid, to produce a constant temperature rise, is an accurate method of determining flow which records in weight-flow rate and does not require compensation for temperature and pressure. This is used in the **Thomas meter** (Cutler-Hammer, Inc.) which is designed for metering natural and manufactured gas. It depends for its accuracy on the fact that the specific heat of the gas remains substantially constant regardless of composition. Figure 54 indicates the principle of the meter. Two resistance thermometers in the pipe, connected into two legs of a Wheatstone-bridge circuit, are adjusted to give zero galvanometer deflection for a 2°F. temperature difference. A balancing device adjusts the current through the heating coil so as to maintain the 2°F. rise. A recording wattmeter in the heating circuit makes a flow-rate record and a watt-hour meter integrates the flow.

Dilution or mixture metering (see p. 867) is now little used.

Impulse or turbine meters are frequently used on liquids. Many of the large water meters are of this type. Figure 55 shows a Sparling turbine meter which is made in sizes from 2 to 60 in. or even larger.

Flow Control. Instruments measuring flow rate can, with the addition of a pilot or electrically operated devices such as a valve or damper, be made to control flow.

LIQUID LEVEL

Measurement and control of level may be accomplished by the use of floats, by hydrostatic or differential pressure, by differential temperature, or by electrical methods.

Floats are applied to indicating, recording, and controlling of level. The simplest **indicator** is one in which float movements are shown on a scale by the motion of a slider connected by a table going over a pulley. This device is generally crude. An electric contact may be fitted on the slider to make a controlling device. A more complicated indicator is the **Liquidometer gage** which transmits the movement of a float to an indicating scale by a balanced hydraulic system (Fig. 56). Two bellows at the transmission end are connected to a similar pair at the indicator. The whole system is filled with a liquid, so that, when a change in position of the float compresses one of the bellows and expands the other, the set in the indicator acts similarly but in the opposite direction. Compensation for temperature changes is made by the linkage in the instrument. With changes in float position both bellows move in the same direction at *C*, and rotation is about *D*. Temperature changes, however, cause equal expansion or contraction of both systems with movement in opposite directions, therefore canceling by turning about *C*. This system is also made for recording. Another distant-type indicator actuated by a float is the **MacCreedy** (J. H. Bunnell) which was designed especially for oil-tank farms. It is essentially a dual telegraph system operating on 24 volts direct current. Each ⅛-in. movement of the float in either direction sends an impulse which moves the receiving indicator.

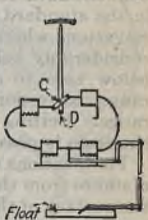


FIG. 56.—Liquidometer liquid-level gage.

Recorders of the float type are numerous. The differences are chiefly in the mechanism used for reducing the motion—rack and pinion, worm and gear, or pulley. Methods of mounting charts and clocks differ in accordance with the reduction methods used. Figure 57 shows one instrument of this type where the chart is on a cylinder which is turned by movements of the float. The pencil is drawn lengthwise on the chart by a weight, the speed being governed by the clock.

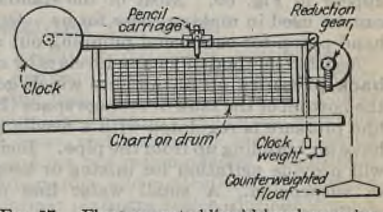


FIG. 57.—Float-operated liquid-level recorder.

Level control by means of a float may be accomplished very simply with a valve directly operated by the float arm (Fig. 58). This type is in common use. Or the valve may be placed at a distance by interposing rods and counterbalanced levers. For greater distances the float is made to operate a pilot valve to control air, gas, or water pressure on a remote diaphragm valve. Direct-acting controllers of the liquid-filled type are also used. In these a bellows is compressed by the float to cause the expansion of a similar bellows mounted on the valve stem. Electric control may be obtained by the use of a mercury-bulb contactor so arranged as to be tilted by a float.

Hydrostatic pressure methods of measuring level determine the weight or pressure of a column of liquid. For this, the various pressure indicators and recorders of the box, diaphragm, Bourdon, and helical types (*q.v.*) are used. If the liquid be in a slightly raised tank or so located that the gage may be connected and mounted at or near the datum line, no change need be made in the standard pressure gage. For greater elevations where the gage must be mounted considerably below, the pointer may be set below zero to obtain an instrument scale range corresponding with the tank level range. Setting back up to 300 per cent of the scale is possible.

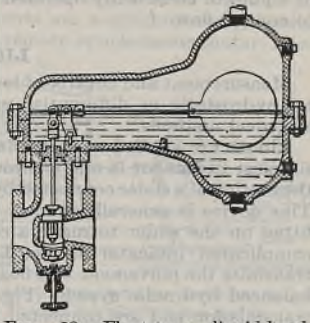


FIG. 58.—Float-type liquid-level controller.

For locations where the gage must be some distance from the point of connection, above the surface of the liquid, or otherwise so located as to make the standard pressure connection impossible, a pneumatic transmission system is used. This consists of a metal bell or "sensitive bulb," connected by a capillary tube to the instrument, the bell being open at the bottom or covered with a flexible rubber diaphragm. The air in the bell is at atmospheric pressure when no liquid is above it, but the pressure increases with the head. Various designs are available to permit connection to pipe lines, for lowering into large reservoirs, and for almost any requirement. Several of these appear in Fig. 59. Most of the standard materials for corrosion resistance can be used in making these forms. Open bells are usually provided with a hand pump for occasional pumping out and filling of the line.

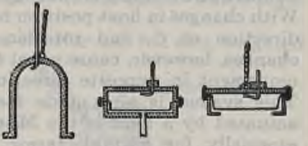


FIG. 59.—Hydrostatic bells for liquid-level measurement.

Another method of gaging levels is by measuring the **back pressure** on an air line which goes almost to the bottom of the tank or storage space (Fig. 60). The line pressure is regulated with a needle valve so as to have air bubbling up from the pipe. Incidentally, this will provide agitation for mixing or keeping material in suspension. A small water line may be used similarly if conditions permit.

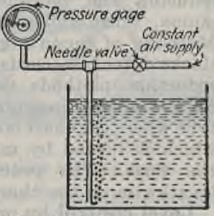


FIG. 60.—Back-pressure type liquid-level gage.

Differential Pressure. As explained on p. 2028, pressure gages actually measure the difference between the pressure within and without the element or, in other words, between the pressure at the location of the sensitive bulb or pipe connection, and the atmospheric pressure. Where, however, the pressure at the connection is due in part to pressures greater or less than atmospheric above the liquid, as in closed tanks, it is necessary to use a different type of instrument. A two-element pressure gage may be used to show the two separate pressures and the level obtained from the difference. A direct reading is possible by the use of a U-tube, as in the **King-Seeley Telegage,** or any differential pressure element such as is used in orifice-type flow meters (see pp. 2040 to 2043).

Differential Temperature. The difference in temperature between steam and water in a steam-boiler water column is utilized in several instruments for indicating, recording, and controlling boiler-water levels. The **Bristol-Derr** indicator and recorder amplifies this difference by means of a series thermocouple, wound on a brass tube which is connected parallel with the regular water column. The alternate junctions of the copper-constantan couples are placed at the top of the tube. The others are stepped up from the bottom. With no water all couples are at the steam temperature, and the instrument reads zero. Rising water cools more couples, and the millivoltage increases. The instruments are of the usual millivoltmeter type (see pp. 2062, 2064, and 2082).

The heat of the steam above the water in a boiler is used for generation of steam in a separate system in several level controllers. Figure 61 shows a generator composed of two concentric tubes, the inner connected to the top and bottom of the boiler. Water fills the annular section and is converted into steam when a drop in level increases the steam space in the tube. The consequent increase in pressure operates the feed-water inlet valve to which the generator is connected.

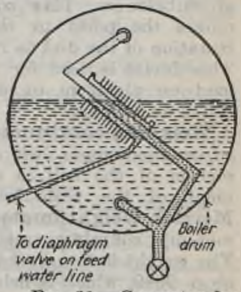


FIG. 61.—Generator for differential-temperature boiler-level controller.

Another level controller which functions by reason of this temperature difference in the water column is the **Cope.** In it temperature changes, due to boiler-level changes, expand or contract a brass tube, connected in parallel with the regular water column. Through a multiplying linkage, its changes in length operate the inlet valve.

Electrical methods can be used to indicate or control high-low levels for electrolytes. An electrode is placed with its end at the level to be indicated, and contact with the electrolyte establishes a closed circuit.

WEIGHING AND WEIGHT CONTROL

Weighing is used for determining weight of stationary, continuous, or intermittently moving loads. There are four fundamental types of scale: (1) the equal-arm balance, (2) the spring balance, (3) the multiplying-lever or steel-yard balance, and (4) the pendulum balance. Industrial weighing makes use chiefly of the compound lever and pendulum scales. Spring balances suffer from spring hysteresis and are not recommended. Industrial scales may be classified as indicators, integrators, and controllers. Indicating and integrating scales are of several forms: platform scales, dormant scales (platform scales flush with the floor), crane scales, tank and hopper scales, truck and track scales, and belt and bucket scales (these last for integrating continuous loads).

Integrating or **totalizing scales** weigh both stationary and moving loads. In the stationary type the scales may be balanced automatically or by hand. Such scales usually print a weight ticket and simultaneously register the weight on a counter. The **Merrick automatic scale** balancing mechanism has a power-applying device which drives the poise backward or forward on the beam whenever it is out of balance, until balance is restored. The poise is driven by a screw running through the center of the beam. Attached to the screw at the fulcrum end of the beam is a disk with rollers on its periphery. A small, horizontal motor-driven bolt makes contact with the rollers on both sides. When the beam is in the balanced condition, the disk is perpendicular to the belt and does not turn. When the beam leaves the balanced condition, the disk rollers contact the belt at an angle, causing the disk to rotate at a speed proportional to the degree of unbalance. This rotates the screw and moves the poise to the balanced position. Rotation of the disk is recorded on a counter. This device is used for weighing any form of load on platform or dormant scales, or in tanks or hoppers.

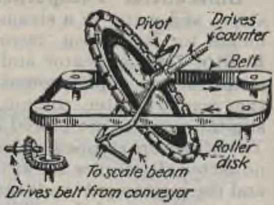


FIG. 62.—Simplified representation of Merrick weightometer integrating mechanism.

Weighing continuous loads is accomplished on a conveyor belt or bucket carrier. A section of the belt is suspended from the scale mechanism, and movement of the scale beam operates an integrator mechanism. In the **Merrick Weightometer** the load is balanced by a cylindrical displacer in a mercury container, resulting in a beam deflection proportional to load. The weight is integrated by means of a roller disk similar to that described on p. 2049, which is held in contact with the small driving belt at an angle proportional to the instantaneous load. The driving belt is driven at a speed directly proportional to the speed of the load belt, and hence the integration measures total load. The integrating mechanism is indicated in schematic form in Fig. 62. In the **Chatillon Telepoise** scale the beam is balanced for normal load by means of springs. Deviation from the normal load causes the beam to rise or fall by an amount proportional to the load. This position is converted into a series of electric impulses which register on a magnetically driven counter. The number of impulses for each rotation of the impulse-transmitting mechanism is proportional to the instantaneous load, and the number of rotations of the impulse mechanism is proportional to the speed of the conveyor belt. Hence the impulses per interval of time are a measure of the passing load for that interval.

Weighting intermittent loads so as to approximate continuous flow is accomplished with liquids in weigh tanks; or with solids in hoppers or on double-belt weighers. Weigh tanks are used singly or in pairs. The scale is balanced for a definite load and, when that load is reached, the weight of the load automatically closes the intake and opens the discharge valve. A counter registers the number of fillings of the tank. The action of weigh hoppers is similar, with the substitution of a feeder or gate-controlled feed hopper for the inlet valve. In the **Syntron batch-weighing machine** the material is fed through a feeder trough, which operates by means of vibration imparted by an electromagnet, into a rotating weigh hopper. When the weigh hopper approaches the correct weight, the feed is cut down, by decreasing the amplitude of pulsation of the electromagnet, to a small stream, until the exact weight is reached when the feed is cut off. The rotating weigh hopper consists of several sections; when the feed is cut off, the hopper rotates to discharge, and the next section is then filled. Photoelectric devices are now available for controlling different types of feeders. The weigh indicator intercepts a light beam to a photoelectric cell which operates a relay shutting off feed. For greater accuracy a second photoelectric cell is used for complete cut-off—the first only cutting off the feed to a "dribble." Instead of a weigh hopper the **Richardson Conveyoweigh**, indicated diagrammatically in Fig. 63, uses two belts, one a feed belt and the other a weighing belt. The feed belt delivers batches of equal weight to the weigh belt. When the weight on the latter tips the scale mechanism, the circuit to the feed-belt motor is broken and that belt stops. Meanwhile, the continuously operating weigh belt delivers part of its load, the scale beam rises, and the feed belt commences the next feed cycle.

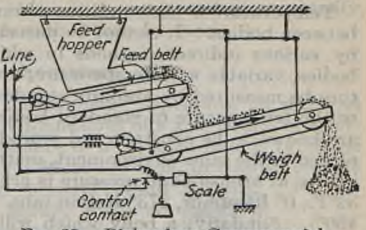


FIG. 63.—Richardson Conveyoweigh.

Weighing Controllers. Any form of automatic intermittent weigher may be used as a controller if it follows a definite time cycle. Otherwise a timing gear may be added. Continuous-weight controllers have inherent timing. This type of equipment usually consists of a belt conveyor driven at constant speed, part or all of which is balanced by a scale beam. Departure from the set control rate tends to open or close a gate at the bottom of the feed hopper so as to maintain constant weight-feed rate to the belt. Figure 64 shows how this is accomplished in the **Hardinge constant-weight feeder.** Instead of the hopper the **Syntron gravimetric feeder** uses the vibrating feeder trough described above, and the variation of weight experienced by the belt conveyor changes the amplitude of pulsation of the electromagnet of the feeder trough.

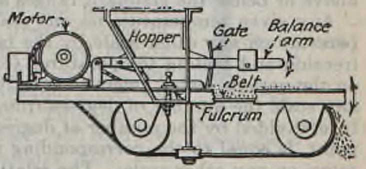


FIG. 64.—Hardinge constant-weight feeder.

Proportioning. Maintenance of the flow of solids in a definite proportion to a liquid flow is accomplished by means of either intermittent load weight or continuous-weight controllers. One system uses a displacement flow meter to measure the fluid stream, and each time a definite quantity has passed through the fluid meter an electric contact operates the solid feeder for a definite time period. In another system a flow rate meter using the differential-pressure method is used to regulate the quantity of feed by a continuous-weight controller. Differential pressure from an orifice or venturi is used to operate floats, which in turn operate an electric transmitter that sends out time signals to the weight controller.

The **Bailey coal meter** (Fig. 65) measures solids flow indirectly in terms of volume. It is installed in a pipe not more than 20 deg. from vertical and measures the volume flow of any granular material descending at a fairly uniform rate so as to fill the pipe. It consists of a screw-shaped vane which is rotated by the descending solids and which in turn drives a counter. Probable accuracy on uniform material is one-third of 1 per cent.

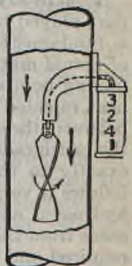


FIG. 65.—Bailey coal meter.

TEMPERATURE MEASUREMENT AND CONTROL

Temperature is the condition of matter which determines the flow of heat between bodies. It cannot be measured directly but is readily determinable by various indirect methods in which physical or electrical properties of bodies, variable with temperature, are measured instead. Nor can temperature be measured by absolute standards. It must be determined in relation to the temperature of standard bodies under conditions of temperature that are known to be constant and reproducible. Thus a body which will, when placed in the same environment, neither lose heat to nor gain heat from melting ice at atmospheric pressure is arbitrarily assigned a temperature of 0°C., 32°F., 0° Réaumur, 273° Kelvin (abs. °C.), or 492° Rankine (abs. °F. = °F. + 460°). Similarly, a body which will neither gain nor lose heat in contact with steam at atmospheric pressure is assigned a temperature of 100°C., 212°F., 80° Réaumur, 373° Kelvin, or 672° Rankine. The range of temperature between the lower and upper standard temperature in the various cases is 100°C., 180°F., 80° Réaumur, 100° Kelvin, and 180° Rankine. The degrees in each case are equal divisions of the entire range. Temperatures above or below the standard ranges are allocated by extrapolation.

Any given temperature on one scale is related to the corresponding temperature on any other scale by the fact that the range of units between the freezing and boiling temperatures (of water), although expressed differently on the various scales, refers to exactly the same temperature difference. On any scale the number of degrees from the freezing point to a given temperature, divided by the number of degrees from the freezing point to the boiling point, is equal to the corresponding ratio for the corresponding temperature range on any other scale. The relations

$$\frac{\text{C.}}{100} = \frac{\text{F.} - 32}{180} = \frac{\text{Réaumur}}{80} = \frac{\text{Kelvin} - 273}{100} = \frac{\text{Rankine} - 460}{180}$$

therefore constitute the fundamental and easiest method for converting temperatures on one scale to those of another.

Thermodynamic Temperature Scale. From a consideration of the part played by temperature in the theoretical Carnot cycle, Kelvin proposed a fundamental temperature scale which is independent of the properties of actual matter. On this scale the absolute temperature of the freezing and boiling points of pure water at a pressure of 1 atm. are proportional to the heat rejected and absorbed by a reversible thermodynamic engine operating between these temperatures, which are then in the ratio of 273 to 373 on the centigrade scale. No actual thermometer gives results which correspond exactly to this scale, although the **constant-volume gas thermometer** follows it very closely when used with the "permanent" gases such as helium, hydrogen, and nitrogen. By correcting for the known deviations of these gases from the laws of perfect gases, it is found that only small corrections are required and that these are generally insignificant for engineering purposes.

The constant-volume gas thermometer is used only as a fundamental or experimental instrument because of its inconvenience. Other more practical instruments have been devised and calibrated in terms of fixed points established by its use. A series of such fixed points for thermometer and pyrometer calibration appears in Table 1. Based on range, temperature instruments have been divided into thermometers for temperatures to about 1000°F., and pyrometers for temperatures above 1000°F., but this classification, never strictly valid, is now almost meaningless. Not only are electric resistance thermometers used at high temperatures as well as low, but also thermoelectric thermometers are used throughout the entire range of industrial temperature measurement. Therefore, it seems practical to classify thermometers by mechanism or principle rather than by range.

Physical Thermometers

There are three general classes of thermometer depending on physical changes: (1) those measuring a change in dimensions of a gas, liquid, or solid; (2) those measuring change in vapor pressure of a confined liquid; (3) those measuring a change in pressure of a gas. Most common, of course, are the liquid-in-glass (liquid expansion) and bimetal (solid expansion) thermometers of the first group, while all groups are represented in the "pressure-spring" or "filled-system" thermometers for industrial use.

Glass thermometers filled with mercury are available for the range from −38°F. to +750°F. Using borosilicate instead of lead glass, mercury thermometers filled with nitrogen under pressure are satisfactory to 950°F.; with Pyrex No. 172 glass they are useful to 1200°F. For temperatures below the freezing point of mercury, toluol, alcohol, or pentane may be used, having respectively freezing points of −143°, −171°, and −328°F. Glass thermometers are calibrated either for total immersion or for partial immersion. Total-immersion thermometers, insufficiently immersed, will read low for temperatures above the surroundings and vice versa. The stem correction for such thermometers is

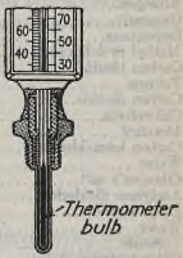


FIG. 66.—Typical industrial thermometer.

$$C = Kn(T - t)$$

where C is the correction, degrees. For Fahrenheit thermometers, K equals 0.00009; for centigrade thermometers, K equals 0.00016. The number of degree divisions of emergent mercury column equals n, T is the bulb temperature, and t is the mean temperature of the emergent column. **Industrial thermometers** (Fig. 66) are partial-immersion thermometers protected with a glass-covered metal scale and a metal tube or guard over the bulb. Thermometers of various lengths for temporary and permanent use, with scale faces set at various angles, are available. These instruments may be read at a distance of several feet. For reducing temperature lag to the bulb, the protecting casing is usually filled with a heat-transferring medium. This may be oil, mercury, copper powder, or graphite; or a low-melting metal, depending upon the temperature. For measuring temperatures of gases the bulb guard is perforated to decrease temperature lag. For installation in corrosive mediums or pressure apparatus, standard thermometer wells are available.

Bimetallic or differential expansion thermometers, formerly used only where simplicity and low cost were primary objects, are now available as high-grade instruments. Thermometer bimetal consists of sheets of metal of relatively high and low coefficients of expansion, such as brass, monel or steel, and iron, nickel-iron, or invar laminated by welding, brazing, soldering, or even riveting. The bimetal may be used in strip form or coiled into spirals or helicals. The Weston thermometer is actuated by a multiple-layer helix. By close clearances between sheath and outer layer and between subsequent

Table 1. Fixed Points for Thermometer and Pyrometer Standardization

Substance	Phase change	Temperature	
		°C.	°F.
Helium	Melts	<−271	<−456
Hydrogen	Boils	−253	−423
Oxygen	Melts	−227	−377
Nitrogen	Boils	−196	−321
Oxygen	Boils	−183	−297
Isopentane	Melts	−160	−256
Methyl cyclohexane	Melts	−126	−195
Carbon bisulfide	Melts	−112	−170
Toluene	Melts	−95.0	−139.0
Carbon dioxide	Sublimes	−78.5	−109.3
Chloroform	Melts	−63.5	−82.3
Mercury	Melts	−38.9	−38.0
Carbon tetrachloride	Melts	−22.9	−9.2
Water	Melts	00.0	+32.0
Glauber's salt	Melts	32.4	90.3
Acetylene dichloride	Boils	55.0	131.0
Ethyl alcohol	Boils	78.3	172.9
Water	Boils	100.0	212.0
Toluene	Boils	110.0	230.0
Chlorobenzene	Boils	132.0	269.6
Brombenzene	Boils	156.6	313.9
Aniline	Boils	184.5	364.1
Nitrobenzene	Boils	209.0	408.2
Tin	Melts	231.9	449.4
Diphenyl	Boils	254.6	490.3
Naphthol (α)	Boils	278.0	532.4
Diphenylamine	Boils	302.0	575.6
Lead	Melts	327.4	621.3
Mercury	Boils	357.3	675.1
Potassium dichromate	Melts	397.5	747.5
Zinc	Melts	419.4	786.9
Sulfur	Boils	444.6	832.3
Lead chloride	Melts	501.0	933.8
Calcium nitrate	Melts	561.0	1,041.8
Antimony	Melts	630.0	1,166.0
Aluminum	Melts	658.7	1,217.7
Manganous sulfate	Melts	700.0	1,292.0
Potassium chloride	Melts	770.3	1,418.5
Sodium chloride	Melts	800.4	1,472.7
Sodium carbonate	Melts	852.0	1,565.6
Sodium sulfate	Melts	884.7	1,624.5
Silver	Melts	960.5	1,760.9
Gold	Melts	1,063	1,945
Potassium sulfate	Melts	1,069	1,956
Copper	Melts	1,083	1,981
Stannic oxide	Melts	1,127	2,061
Lithium silicate	Melts	1,201	2,194
Barium fluoride	Melts	1,280	2,336
Nickel	Melts	1,452	2,646
Cobalt	Melts	1,480	2,696
Iron	Melts	1,530	2,786
Palladium	Melts	1,549	2,820
Platinum	Melts	1,755	3,191
Alumina	Melts	2,000	3,632
Tungsten	Melts	3,400	6,152

layers good responsiveness is secured. Ranges to 1000°F. and accuracies of 1 and ½ per cent are available.

Pressure-spring Thermometers. The sensitive element of these thermometers consists of a metal bulb joined by a capillary tube to some form of pressure spring. This may be a Bourdon tube, a spiral or helix of flattened tubing (see p. 2029), a capsular element, or even a mercury manometer, any of which may actuate a pointer or recorder pen.

Liquid-expansion thermometers are made in ranges between −170° and +500°F. for organic fillings and between −38° and +1000°F. for mercury fillings. The temperature scales are equally divided. The organic fillings have a high expansion coefficient and require only very small bulbs permitting good responsiveness. The mercury filling has a low expansion coefficient requiring a larger bulb and an all ferrous construction.

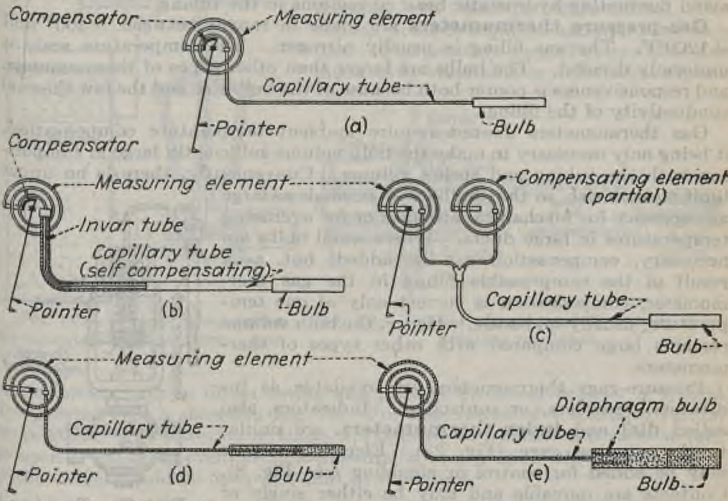


FIG. 67.—Methods of compensating pressure-spring thermometers.

Compensation for ambient temperature changes is generally necessary if the capillary tubing is more than a few feet long, particularly with the organic fillings. For case compensation a bimetallic element connected to the measuring spring offsets the spring motion occurring as a result of spring temperature changes, or spring and tubing temperature changes (assuming tubing always at spring temperature) (see Fig. 67*a*). The tubing may be compensated separately and more exactly by the invar wire method, Fig. 67*b*. By proper choice of tube and wire dimensions the differential expansion between the tube and wire may be made just to offset the expansion of the filling medium in that tubing. Complete compensation is also secured by the parallel-system method in which the case and tubing errors of the measuring system are exactly offset in a second filled system, identical with the first except for omission of the bulb (see Fig. 67*c*.)

Vapor-pressure thermometers (Fig. 67*d*) are made in ranges falling between limits of about −25° and +600°F. The bulb systems are partially filled with a wide variety of volatile liquids, of which methyl chloride, ethyl ether, alcohol, sulfur dioxide, benzol, toluol, aniline, or water are commonly used. The temperature scale is unequally divided, the interval increasing with increasing temperature. This characteristic is often utilized to increase readability in the operating range. Moderately small bulbs and high responsiveness—of great importance in control—are characteristic of vapor-pressure thermometers.

While vapor-pressure thermometers require no compensation, they have certain limitations when required to cross ambient temperature. Either the bulb volume must be sufficient to contain the filling liquid which distills from the spring and tubing when the bulb becomes the cold end (the bulb must be mounted at instrument level) or the "double-filling" arrangement (with volatile liquid only in the bulb as shown in Fig. 67*e*) must be used to avoid fluctuating hydrostatic head corrections in the tubing.

Gas-pressure thermometers are made in ranges between −300° and +1200°F. The gas filling is usually nitrogen. The temperature scale is uniformly divided. The bulbs are larger than other types of thermometers, and responsiveness is poorer both because of the bulb size and the low thermal conductivity of the filling.

Gas thermometers do not require ambient temperature compensation, it being only necessary to make the bulb volume sufficiently large in comparison with the tubing and spring volume. Conveniently, there is no upper limit to bulb size, so that bulbs may be made as large as necessary for mechanical strength or for averaging temperatures in large ducts. Where small bulbs are necessary, compensation can be added, but, as a result of the compressible filling in the gas thermometer, compensation is correct only at one temperature, usually mid-scale. Hence, the bulb volume remains large compared with other types of thermometers.

Pressure-gage thermometers are available as indicators, recorders, or controllers. Indicators, also called **dial** and **index thermometers**, are similar to the pressure gage (Fig. 23). Electric contacts may be added for control or signaling (see Fig. 8). Contacts are movable and may be either single or double. Recorders are available for all forms of chart, in any of the three classes. Up to four temperature elements, with four pens, may be had in a single instrument. Electric systems or pilot valves are added for control (see p. 2019).

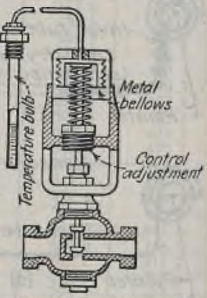


FIG. 68.—Typical self-operated temperature regulator.

Temperature regulators are usually pressure-spring thermometer systems which have no indicator or recorder attachment, but which move a valve, damper, switch, etc., to maintain a substantially constant temperature at the bulb. The simplest is the self-operated regulator (Fig. 68), usually vapor-pressure operated but also made in liquid-expansion types. The expansion element is usually a metal bellows connected by flexible tubing to the bulb. Temperature increase closes a direct-acting valve or opens a reverse-acting valve against a spring. The control point is adjusted by varying the spring pressure. Valves are of the balanced type and will throttle. For heating applications greater accuracy and closer control are obtained in a different type of regulator, whose bulb and tube system are filled with liquid which overflows, with increasing temperature, into the bellows. The bellows, located within the valve body, is exposed to the heating medium, which vaporizes the liquid and thus positions the valve. Another type of controller utilizes a pilot valve, operated by a bellows which admits the downstream reduced pressure to a chamber above a diaphragm upon whose other side is exerted the upstream pressure. The operation of this valve is similar to a pressure regulator except that a vapor-filled bellows system actuates the pilot valve instead of a spring. Such regulators are cheaper than those using a pilot system but are not so accurate. The latter, of which Fig. 69 is a typical example, use air, water, or steam pilot valves to regulate a control valve, damper, etc. (see pp. 2019–2025). The control point is adjusted and indicated on a dial. Temperature regulators are also made with bimetallic or differential expansion elements both in direct-acting and pilot or electrical types. A typical air-operated type is shown in Fig. 70. For description of diaphragm valves, see p. 2022. The selection of these will depend on whether the valve should be open or closed in event of pilot-pressure failure, or upon whether the action of the valve should be open-and-shut or throttling.

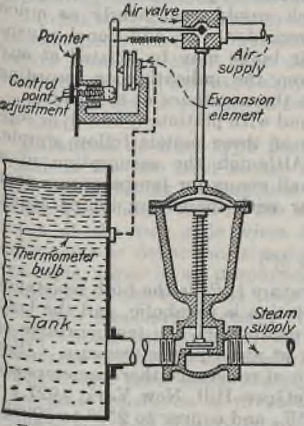


FIG. 69.—Typical direct-acting air-operated temperature controller.

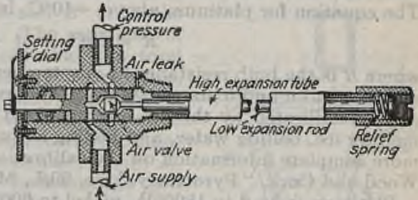


FIG. 70.—Typical air-operated bimetallic temperature regulator.

Electrical Thermometers

There are two classes of electrical thermometers: (1) resistance; (2) thermoelectric thermometers.

Resistance thermometers consist of a coil of wire of relatively high temperature coefficient of resistance, connected into one arm of a Wheatstone-bridge circuit. Resistance, and hence temperature of the coil, is measured either by balancing the bridge or by noting the deflection of the galvanometer. This is the most accurate type of thermometer. Accuracy of one-quarter of 1 per cent and better is possible in industrial instruments, which is closer than the chart can be read for long-range instruments at low temperature. Resistance thermometers are of use between about −330° and +1800°F. and may be used to indicate or record one or more temperatures, the difference of two temperatures, or the average of several temperatures. Although the cost of a single-resistance instrument is two or three times that of a comparable pressure-spring thermometer, the use of several bulbs recording or indicating on a single instrument may make multiple installations little more expensive than an equivalent number of pressure-spring thermometers. Resistance thermometers may be cheaper for installations which require remote location of the bulbs. In the case of indicating instruments, the number of bulbs that can be switched consecutively to a single indicator is limited only by the frequency with which each temperature must be read. In recording instruments, up to 16 bulbs are switched automatically and consecutively to a single recorder, provided temperature changes are not too rapid. Measurement of an equal number of temperatures would require four 4-point pressure-spring-thermometer recorders which would cost nearly as much as a single 16-point resistance-thermometer recorder. An additional advantage of resistance thermometers is that their bulbs may be located at any reasonable distance, say up to ½ mile, from the indicating or recording instrument. The distance is limited only by the resistance of the leads.

Resistance-thermometer bulbs are wound with platinum, nickel, or copper. Temperature coefficients of resistance of these metals follow simple, known laws with reasonable constancy. Although the assumption of a straight-line law for platinum gives only small errors for temperatures near zero, the error becomes progressively larger with increasing temperature. The equation for platinum above -40°C. is

$$R = R_0(1 + at + bt^2)$$

where R is the bulb resistance at any temperature t; R_0 is the bulb resistance at 0°C.; and a and b are constants. This relation is parabolic, and the bulb can be calibrated by the use of three fixed points, *i.e.*, the temperatures of melting ice, boiling water, and boiling sulfur at atmospheric pressure. (For more complete information on the calibration of resistance thermometers see Wood and Cork, "Pyrometry," pp. 69*ff*., McGraw-Hill, New York, 1927.)

Platinum is used to 1800°F., nickel to 600°F., and copper to 250° to 300°F. Except for very high temperatures, high-resistance bulbs are preferable in order to reduce the effect of line-resistance variation. Base-metal bulbs consist of metal coils supported on an insulating rod or frame and enclosed in a metal tube. Platinum bulbs are preferably sealed in a quartz tube to obviate contamination at high temperatures.

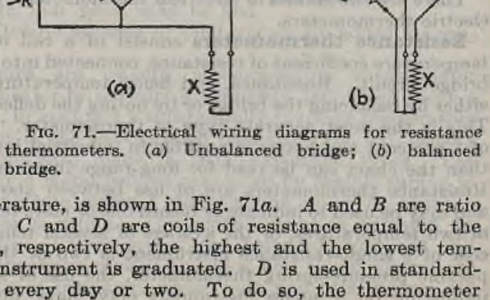


FIG. 71.—Electrical wiring diagrams for resistance thermometers. (*a*) Unbalanced bridge; (*b*) balanced bridge.

Deflection Indicators and Recorders. As noted above, the resistance of the thermometer bulb is measured by means of a balanced or an unbalanced Wheatstone bridge. The **unbalanced bridge,** using the galvanometer deflection as a measure of the temperature, is shown in Fig. 71*a*. A and B are ratio coils of equal resistance. C and D are coils of resistance equal to the resistance of the bulb at, respectively, the highest and the lowest temperatures for which the instrument is graduated. D is used in standardizing the battery voltage every day or two. To do so, the thermometer bulb X is disconnected, and D is connected into the network by switch S. Rheostat R is adjusted until the galvanometer deflects to the lowest division on the scale. Millivoltmeters are used as indicating instruments, together with a separate bridge box containing the bridge circuit and standardizing switch. Recorders are of the usual depressor-bar roll-chart type with the bridge circuit installed either within the case or in a separate box. Multipoint recorders have a clock-driven commutating switch for periodically connecting the various bulbs into the network.

In millivoltmeter instruments the small amount of power available from a galvanometer makes it necessary to add an auxiliary source of power to secure control action. The pointer may be depressed at intervals by a meter or solenoid-operated chopper bar to operate control or signal contacts, or an inductive or capacitive detection of pointer position (see p. 2021) may be used.

Balanced or null-type instruments use a circuit similar to Fig. 71*b*, which is the one used in the **Leeds and Northrup** instrument. In the case of indicating instruments the balancing is accomplished by hand, a single knob being used in the simultaneous adjustment of the two slide wires S and S' until the galvanometer deflection is zero. The resistance of the slide wires is so proportioned that the total resistance of the two ratio arms A and B, fixed coils, and their associated sections of slide wire, remain equal for all adjustments. At the same time the resistance of C varies and the bridge reaches balance when C has been made exactly equal to bulb X. The lead resistance is divided between the two arms C and X of the bridge and does not affect the balance. The three leads of each bulb must be identical copper conductors.

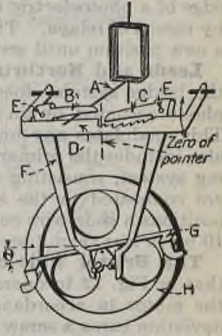


FIG. 72.—Leeds and Northrup automatic circuit-balancing device.

Recorders use ingenious automatic balancing devices to adjust the slide wires until zero deflection is obtained. The first widely used self-balancing mechanism is shown in Fig. 72. Galvanometer pointer A swings in a horizontal plane beneath two normally horizontal fingers B and C. Periodically a chopper bar D, driven by the recorder motor, ascends and, striking the pointer, moves it upward a short distance. If the galvanometer is in the zero position, the pointer is lifted between the two opposing fingers so as to touch neither. If it should be deflected to one side or the other, owing to unbalance of the bridge, then it will strike and lift one or the other of the fingers, such as B, an amount proportional to the deflection of the galvanometer. The finger pivots about E, and attached arm F rotates clutch lever G in front of the disk H. As soon as the lever has tipped, a cam pushes the clutch lever back until cork pads attached to its ends engage the disk, and another cam striking the end of the lever G turns it until it is horizontal, carrying the disk through an angle θ. This returns the arms F and the fingers B and C to their normal positions. The slide wires are mounted on a drum turning with disk H. Rotation of the disk moves the slide wires under stationary contact brushes tending to bring the bridge into balance.

Leeds and Northrup Micromax balancing device, which has superseded the mechanism described above, is a refinement which does not differ from it fundamentally. Periodically the pointer A is clamped in position by a chopper bar, whereupon two vertical fingers, which replace fingers *B* and *C*, move in toward the pointer until they touch it. If the pointer is off center, one of the fingers stops before reaching the center while the other moves past center, turning the clutch lever *G* by an amount proportional to the deflection, which then functions to rotate the disk *H* exactly as in the older method. Quicker balancing without strain on the galvanometer is the advantage of the newer method.

In both types, as soon as the movement of the disk has been completed, the clutch lever *G* is released, the galvanometer pointer *A* seeks its new position nearer the zero line, and the cycle described is repeated until the bridge is balanced. The recorder pen is operated by the movement of the disk *H*. For each position of the disk the pen has a definite position. In multipoint recorders a print wheel, turned in synchronism with the commutating switch, replaces the pen.

The Tag Celectray balancing device utilizes a mirror galvanometer in which a beam of reflected light replaces the pointer. Movement of the galvanometer causes the beam of reflected light to pass across the controlling edge of a photoelectric tube screen, which operates a reversing electric motor by means of relays. The motor in turn causes the moving contact to assume a new position until zero galvanometer deflection is obtained.

Leeds and Northrup Speedomax balancing mechanism, while utilizing the bridge system, does not use a galvanometer. Instead, a carbon microphone actuated by an a.c. armature is introduced into the bridge circuit. This modulates any unbalanced direct current in the bridge circuit, which also includes the primary of a transformer. The secondary feeds an amplifying system, consisting of two opposed grid-controlled rectifier tubes which are connected to the exciting winding of a split-field motor. The motor positions a slide-wire contact. This instrument will cross the chart width of 10 in. in less than 2 sec. without overshooting.

The Bristol wide-chart recorder uses a feeler mechanism similar to that in Fig. 72 to operate contacts feeding a reversing motor. Rotation of the motor in accordance with the amount and direction of galvanometer deviation turns a screw moving the slide-wire contact along the straight slide wire. The pen mounted on the same carriage records the motions on the adjacent chart.

The Bailey Galvatron (round chart) uses a similar contacting galvanometer but interposes an electronic relay system between contacts and reversing motor which drives the slide-wire disk and associated pen. The Bailey instrument, in the two-, three-, and four-record forms, has a separate slide wire and continuously recording pen for each record. Each unit is balanced in turn by the single galvanometer system.

The Bristol Pyromaster (round chart) uses a Weston contact galvanometer to detect unbalance and to operate a small reversing motor driving the slide-wire disk. By means of a relay system, providing for a holding current to avoid chattering and releasing after an interval for a new determination of the circuit balance or unbalance, a step-by-step balancing action is secured.

In the **Brown** balancing mechanism a stepped feeler table raised against the clamped galvanometer pointer detects its position and correspondingly selects the direction and number of teeth picked up on the ensuing balancing gear stroke. The contact yoke for the spiral slide wire and the pen carriage drive screw are geared to the balancing gear.

The Foxboro Stabilog mechanism which first combined a potentiometer measuring system with a pneumatic control system is shown in Fig. 73.

A slide wire is set manually to the desired control temperature, and deviations from the control point are indicated by galvanometer deflections. At each balancing stroke the galvanometer is clamped, its position detected by the curved cam and translated without lost motion into flapper motion which in turn results in control action tending to return the process to balance. Thus, although the measuring system alone is a deviation instrument, when combined with the throttling reset-type control mechanism and the process, it becomes a null type or self-balancing instrument.

A similar balancing mechanism is used in the wide-chart recorder. As with all recording controllers, the accuracy here cannot be closer than the turn-to-turn slide-wire interval, a limitation which is avoided in the non-recorder described above.

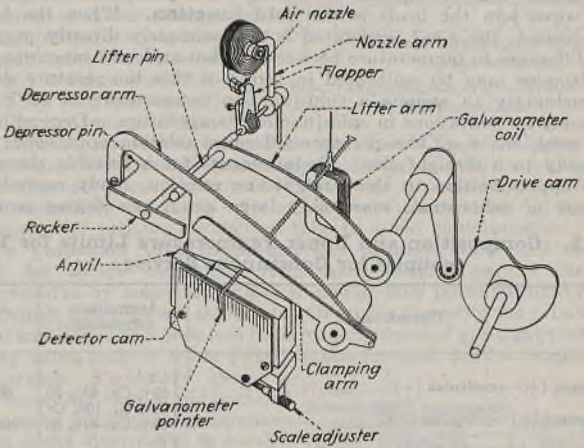


FIG. 73.—Foxboro Stabilog mechanism.

Control equipment can be built into any of these recording instruments. Electric and pneumatic systems are available for two and more position control or signals and for throttling or proportional position control with or without automatic reset. In the Leeds and Northrup instruments contact-operating disks control slide-wire drum, or pilot-operating cams mounted on the slide-wire drum shaft effect the control. A wide variety of equivalent devices suitable to the individual recorder (and control-equipment) design are available on other makes.

Multipoint controllers are widely used as alarm instruments. Under suitable but infrequent conditions they may be used for control of separate, not too rapidly responding processes. Selector switches provide for sequential connection of appropriate control circuits as well as measuring circuits.

Deviation instruments are those in which the setting of the control temperature adjusts a slide wire so that the instrument is balanced when the temperature is correct. They are frequently used as controllers. A simple chopper bar mechanism operates contacts or pneumatic pilots whenever the galvanometer deflects owing to variation of the temperature.

Accessories. For use in multiple-indicating and recording installations, various forms of switching equipment are available (see p. 2065). Standard thermometer wells are available for corrosive, high-pressure, and high-temperature service.

Thermoelectric Pyrometers (see pp. 2653 and 2703). A thermoelectric pyrometer consists of thermocouple, millivoltmeter, and lead wires to connect the two. In 1821, Seebeck discovered that a current would flow through two dissimilar wires welded together at the ends, when one junction was at a higher temperature than the other. Practical application of this discovery is made in the **thermocouple.** Two wires of dissimilar, pure metals or alloys are joined at one end by fusing together. The fused end is the **hot junction.** The unfused ends are connected through suitable leads to a millivoltmeter which completes the circuit. The point at which the thermocouple wires join the leads is the **cold junction.** When the hot junction is heated, the e.m.f. generated is approximately directly proportional to the difference in temperature between the hot and cold junctions, and the millivoltmeter may be calibrated in terms of this temperature difference. It is customary to assume a cold-junction temperature of 32°F. and to compensate for variations in cold-junction temperature. Depending on the metals used, the e.m.f.-temperature-difference relation approaches more or less closely to a straight line. Requirements for a suitable thermocouple include approximation to the straight-line relation, ready reproducibility, constancy of calibration, reasonably large e.m.f. per degree temperature difference, and reasonable cost. Table 2 shows the thermocouples most generally used, their composition, and the upper limits of temperature to which they may be used for continuous service. Thermocouples are of two general sorts, (*a*) noble metal and (*b*) base metal. The noble-metal thermocouple uses one wire of platinum and one of platinum-10 per cent rhodium or of platinum-13 per cent rhodium giving a slightly higher e.m.f. Standard wire diameters range from 0.015 to 0.024 in. Because of volatility platinum-iridium couples are no longer used. Figure 74 shows the temperature-e.m.f. relations between platinum and various other wires used for thermocouples. At any temperature the vertical distance between any two curves is the

Table 2. Composition and Upper Temperature Limits for Thermocouples for Continuous Service.

Thermocouple	Approximate composition	°F.
Copper (+)—constantan (−)	Cu 60% Cu; 40% Ni	600
Chromel (+)—constantan (−)	90% Ni; 10% Cr 60% Cu; 40% Ni	1600
Iron (+)—constantan (−)	Fe 60% Cu; 40% Ni	1600
Chromel (+)—alumel (−)	90% Ni; 10% Cr 94% Ni; 2% Al; 3% Mn; 1% Si	2200
(Platinum rhodium) (+)—platinum (−)	90% Pt; 10% Rh Pt	2800*
(Platinum rhodium) (+)—platinum (−)	87% Pt; 13% Rh Pt	2800*

* 3000°F. for intermittent service.

e.m.f. for a thermocouple using those two wires. Note that the e.m.f. is small for noble-metal thermocouples, necessitating more sensitive instruments. Furthermore, noble-metal thermocouples, being expensive, are made of small wires having high resistance, whereas base-metal thermocouples, being inexpensive, may be of low resistance.

Thermocouple Mounting. Thermocouples may lose their calibration more or less rapidly and require discarding or recalibration as a result of composition changes. Except at low temperatures or in the absence of a contaminating atmosphere, all thermocouples require some protection. Noble-metal thermocouples are particularly affected by reducing atmospheres. Iron-constantan thermocouples are affected by oxidizing; and chromel-alumel thermocouples, by alternately oxidizing and reducing atmospheres.

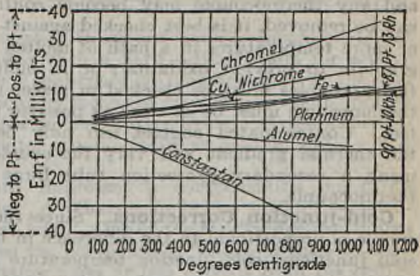


FIG. 74.—Temperature-e.m.f. relations of wires for thermocouples.

A thermocouple mounting consists of suitable protection and a terminal head for connecting the leads. Protection is of three sorts: (1) insulation for the wires, (2) primary protection tube to avoid contamination, and (3) secondary protection tube to protect the primary tube. All three are desirable for severe conditions. All thermocouples require insulation. This is provided by means of single- or double-hole porcelain or lava tubes through which the wires are threaded. Over this assembly the primary tube is slipped and joined to a terminal head by some form of gas-tight connection. Secondary tubes may be either joined to the terminal head or supported by the apparatus. Figure 75 shows a typical mounting for a noble-metal thermocouple. Complete protection, as shown in the illustration, is usually desirable for noble-metal thermocouples. Porcelain or silica tubes should be used for primary protection and are recommended for secondary protection unless greater mechanical strength is required. Metal protection tubes used for secondary protection are not so long lived at high temperatures as are ceramic tubes. For base-metal thermocouples the range of materials used for protection tubes is very wide. As the information is highly specialized, manufacturers had best be consulted in specific cases.

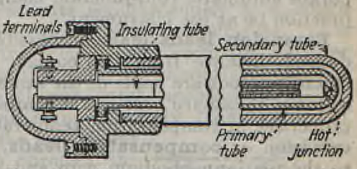


FIG. 75.—Typical mounting for noble-metal thermocouple.

Noble- vs. Base-metal Thermocouples. Up to 2200°F., choice between noble- and base-metal thermocouples depends on cost and continued accuracy. Platinum thermocouples cost several times as much but last many years if properly protected and maintained. Base-metal thermocouples require frequent checking if used at high temperature. The long-run cost of noble-metal thermocouples may be less. In a large heat-treating plant, cost per couple per year, including maintenance, was $3.50 for platinum thermocouples.

Calibrating and Checking Thermocouples. All new thermocouples are annealed above the maximum operating temperature to ensure homogeneity. Heterogeneity will later develop in base-metal thermocouples, and any thermocouple may become contaminated. If the thermocouple can be removed, it is best checked against a standard thermocouple at one or more temperatures in a bath of molten metal such as tin, covered with graphite to prevent oxidation; or in molten salt or in sulfur vapor. If thermocouples must be checked in place, the hot junction of the standard thermocouple must be as close as possible to that of the couple undergoing test. Contaminated couples can only be checked in place, as varying the thermal gradient will vary the couple output. This is simplified by using a secondary protection tube large enough to admit the standard thermocouple.

Cold-junction Corrections. Since the e.m.f. generated by a thermocouple is proportional to the difference in temperature between the hot and cold junctions, cold-junction temperature must be known before the hot-junction temperature can be determined. As instrument calibration is based on some standard cold-junction temperature, variations from actual cold-junction temperature will be reflected in the indicated temperature. In base-metal thermocouples, cold-junction temperature rise is approximately equal to the number of degrees that must be added to the indicated temperature. Approximately half the cold-junction rise must be added in case of noble-metal couples. The exact correction varies with the thermocouple and with the hot-junction temperature. Tables of exact corrections may be obtained from manufacturers. Formerly, the cold-junction temperature was made constant by jacketing the terminal head of the thermocouple, using a steam-heated or thermostatically controlled, electrically heated junction box; or by burying the cold junction 6 to 10 ft. underground where temperature will not vary more than about 5°F. throughout the year. These last methods are not much used at present since modern instruments incorporate automatic cold-junction compensators. These require that the cold junction be at the instrument.

Extension leads make it possible to extend the cold junction to any desired distance from the couple. With base-metal thermocouples such extension leads are made of the same metals used in the couple or of cheaper or lower resistance base-metal combinations having the same thermoelectric properties in compensation temperature range. With noble-metal couples, extension or **compensating leads,** consisting of a copper wire connected to the platinum-rhodium wire and a copper-nickel alloy wire connected to the platinum wire are used. Using extension or compensating leads, the point where the copper leads to the instrument are connected becomes the cold junction. This may be at the instrument or at any point between the instrument and the couple.

Millivoltmeter Indicators and Recorders. The e.m.f. produced by a thermocouple may be measured by (1) a millivoltmeter or (2) a potentiometer. Since potentiometers measure potential without drawing current, the effect of variable line resistance is eliminated. Millivoltmeter instruments, however, are calibrated for a definite line resistance. For accuracy, a reasonably constant line resistance must be maintained. Poor connections, oxidation, breakage of several wires in a stranded lead, or temperature changes in the leads may introduce inaccuracy. These errors are minimized in high-resistance instruments which must be correspondingly sensitive and high-priced. Lead resistance should be checked occasionally.

Millivoltmeter instruments are similar to those described on p. 2082. They are of high resistance, recorders being mainly of the roll-chart type (see p. 2018). Most millivoltmeters incorporate automatic cold-junction compensation as described below.

Cold-junction correction is applied in several ways. Instruments not otherwise compensated are provided with a **zero adjustment** for setting the pointer to take care of a known cold-junction temperature. If the cold junction is brought to the instrument by means of extension or compensating leads, a **bimetallic spiral** attached to the moving system of the millivoltmeter may be used to shift the pointer automatically an amount equal to the cold-junction correction.

The Wilson-Maeulen system impresses a compensating e.m.f. on that produced by the thermocouple. A Wheatstone bridge within the instrument is connected in series with one lead. One of the four bridge resistances is temperature sensitive and causes the compensator bridge balance to vary with ambient temperature, permitting more or less current from the bridge battery to flow to the millivoltmeter to correct the couple e.m.f.

Multiple Selector Switches. Within the limits imposed by the speed of the temperature fluctuations, any number of thermocouples may be connected in turn to a single instrument. Recorders use an automatic commutating switch driven by the recorder clock or motor for series connecting the several thermocouples to the millivoltmeter. Couple capacity ranges from 6 to 16 points maximum, varying with strip-chart make and type, down to 2 points for round-chart types, except Bailey 4. Indicators use a manually controlled switch which operates by the rotation of a brush contact over stationary contact buttons; by the pushing of individual buttons or toggle switches; or by inserting plugs joined by cables to the instrument into sockets connected to the thermocouples, as in a telephone switchboard. If both indicating and recording instruments are to operate in parallel on a multiple installation, the resistance of the millivoltmeters must be high in order that the connection of two instruments simultaneously to a single thermocouple shall not produce a low indication on both instruments. It is preferable to use a special cut-out switch which disconnects the recorder whenever the indicator is connected to the same thermocouple. This is conveniently included in the toggle-type selector-switch mechanisms.

Potentiometer instruments are either of the fully balanced (null) or of the deflection type. In the first the instrument is fully balanced so no current flows through the galvanometer. In the second the instrument is partially balanced, and the galvanometer shows deviation from the fully balanced condition. A simple potentiometer circuit is shown in Fig. 76. The potentiometer consists of three circuits shown respectively in solid lines, dash lines, and dotted lines. The potentiometer circuit (solid lines) is adjusted by means of the rheostat *R* against the standard cell (Weston unsaturated cadmium type) with switch *S* thrown to the right. When the galvanometer shows no current, the potential drop through the slide wire *AB* is equal for the battery and the standard cell. Then the switch *S* is thrown to the left, and the potential due to the thermocouple is determined by moving the sliding contact *X* along the slide wire until the galvanometer reads zero. Calibration of the slide wire shows what proportion of the

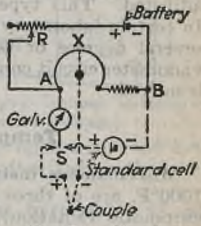


FIG. 76.—Simple potentiometer pyrometer circuit.

potential drop due to the battery is equal to the e.m.f. generated by the couple. Instead of being calibrated in terms of e.m.f., the slide wire may be calibrated in terms of temperature. In indicating instruments the slider is adjusted by hand and its position is shown on a temperature scale.

In recorders the slider is adjusted automatically by the same mechanisms that are used in resistance thermometers (see p. 2059). In certain models the battery rheostat R is also adjusted automatically at frequent intervals. In others a temperature-compensated battery provides a stable current output over long periods. In another type the battery voltage is recorded at intervals for reference.

Automatic cold-junction compensation is furnished on the potentiometer-type thermoelectric thermometers and on many indicators. Figure 77 shows a typical arrangement of the cold-junction compensation. The thermocouple is connected through the galvanometer, not to the end of slide wire as in Fig. 77, but to the junction of two coils bridging the slide wire. The left coil, wound with nickel wire, increases in resistance with rising temperature, in effect shifting the thermocouple end up the slide wire to compensate for the diminishing couple output with rising cold-junction temperature.

The temperature-sensitive compensator coil is ordinarily mounted on the terminal board of the instrument adjacent to the thermocouple cold junctions. In some instruments provision is made for location of the compensator in a remote junction box to permit ordinary copper leads from the instrument to the junction box. In other instruments the compensator remains in the instrument and an auxiliary compensating couple is carried to the junction box. This requires a pair of alloy leads instead of three copper leads.

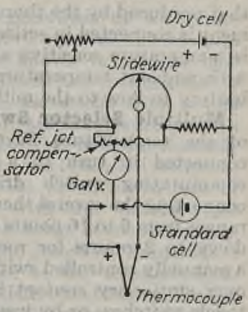


FIG. 77.—Automatic cold-junction compensation.

Deflection potentiometers differ from Fig. 76 in the fact that the slider X normally is stationary. The amount of unbalance of the potentiometer is indicated by the galvanometer. Unless the temperature change is greater than can be shown by full deflection of the galvanometer, X need not be adjusted. This type is used in indicating instruments, often controllers. In certain laboratory types the slide-wire contacts are spaced at intervals of several degrees or perhaps 1 millivolt. A variable resistance in the galvanometer circuit compensates for changes in resistance of this circuit when X is moved.

Temperature Measurement above 1000°F.

Miscellaneous instruments for the measurement of temperatures above 1000°F. are of three types: (1) fusion, (2) calorimetric, (3) optical (monochromatic radiation), and (4) total radiation. Fusion of solids has two pyrometric applications. The small pyramids made of mixtures of oxides, known usually as **Seger cones**, are useful for the approximate measurement of temperatures by steps averaging about 36° from 1100° to 3700°F. Their principal use is in ceramics where they are considered indispensable, not because of high temperature accuracy but because their *temperature-time* performance is similar to that of the materials being fired. Also it is easy to distribute them throughout a kiln or furnace charge to study temperature gradients.

The melting points of metals are often used for standardizing pyrometers. Table 1 includes melting points of certain metals and salts that can be used in the pyrometric range as well as for thermometers. In using this method, the readings of the pyrometer should be plotted against time, with the temperature of the metal bath both ascending and descending through the transition point, as in Fig. 78. This assures its correct location.

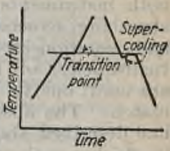


Fig. 78.—Temperature-time curves as used in calibrating thermocouples by fusion-point method.

Calorimetric or water pyrometers are seldom used at present and need no description other than to mention that the temperature rise of a known weight of water, in a container of known weight and specific heat, is used to calculate the temperature of a piece of metal, of known weight and specific heat, which is placed in the water.

Optical Pyrometers (see Radiation, pp. 1002 and 1029. According to the Stefan-Boltzmann law (see p. 1003), a black body (theoretical body which absorbs all incident radiation and reflects none, radiating only in proportion to its temperature) gives off total radiation in proportion to the fourth power of its absolute temperature. Since many heating operations (*i.e.*, objects inside furnaces) are carried out under approximately black-body conditions, two methods based on radiation of incandescent bodies are used for high-temperature measurements. The first method (optical pyrometry) measures the intensity of monochromatic radiation; and the second method (see below, Radiation Pyrometers) measures total radiation. A law formulated by Wien and reformulated more exactly by Planck shows the intensity of radiation of any given wave length to be a complex relation between the wave length and the absolute temperature. Hence, having measured the intensity of a known wave length by visual comparison with a standard, the temperature may be calculated. Instruments based on this method are suitable for temperatures from 1400° to 5000°F., with an accuracy in commercial instruments under best conditions of 10° to 20°. The presence of flames and smoke may introduce very large errors. Radiating objects in the open may read low, although oxides, such as scale on an iron billet, will often reduce the error to small proportions.

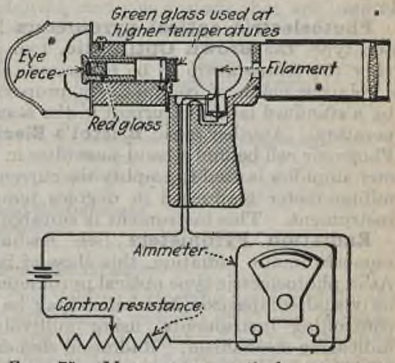


Fig. 79.—Morse type optical pyrometer. (*Leeds and Northrup.*)

The **Morse** type of disappearing-filament **optical pyrometer** (Leeds and Northrup) is shown in Fig. 79. A single wave length is selected by a red glass filter, and the intensity of radiation of this wave length from the radiating source is determined in comparison with an electric-lamp filament, which is viewed against the radiating source. The lamp current is varied until the filament disappears, and this current, as read on an ammeter, is taken as a measure of temperature. In the **"F and F" pyrometer** (Scientific

Materials Company) the lamp current remains constant, and the apparent intensity of the radiating body is varied to secure a "match" by rotating a wedge of absorbing glass. The thickness of the wedge at the point where the filament just disappears is taken as a measure of temperature. In both instruments the temperature range may be changed by the use of absorbing screens. To measure temperature where metallic oxides exist and radiation intensity varies, the **Bi-optical Pyrometer** (The Pyrometer Instrument Co.) matches the color instead of the brightness. Two wedges are used: one to obtain an intensity match and the other to obtain a color match. The absorbing wedge for obtaining an intensity match is like the one described above and functions in the same manner. The other wedge is a bichromatic wedge that permits only red and green light to pass through. By varying the position of the wedge, a different intensity ratio is obtained for the two colors, so that a yellowish-white light can be obtained. This color is matched against a standard lamp—and a "color" temperature is obtained. A third form of optical pyrometer, the **Wanner,** uses a direct-vision spectroscope and polarizing prisms to isolate monochromatic radiation. The operator, in looking in the eyepiece, sees two half-disks, one monochromatic light from a standardized electric light and the other from the radiating body. Rotation of the analyzer for the beam from the standardized lamp decreases its intensity until a "match" is obtained between the two half-disks. Rotation is taken as a measure of the intensity of radiation from the source in comparison with the standard, and hence of the temperature. The lamp is standardized periodically by sighting on a standard amyl acetate lamp.

Photoelectric optical pyrometers have been developed recently. In one type, the **Brown Optimatic,** two photoelectric cells are used: one to view the temperature to be measured; the other to maintain automatically a relative value of conductivity proportional to the first cell, illuminated by a standard lamp. Current in the standard lamp is a measure of the temperature. Another type, **Bristol's Electronic Instagraph,** uses a Weston Photronic cell behind a band-pass filter in the receiving tube, and a potentiometer amplifier is used to amplify the current. A direct-marking type-recording milliammeter graduated in degrees temperature is used as the measuring instrument. This instrument is suitable for temperatures to 2250°F.

Radiation Pyrometers (see Radiation, pp. 1002–1029). Instead of monochromatic radiation, this class of instrument measures total radiation. As in photoelectric type optical pyrometers the measurement does not depend on visual comparison and hence may be used for indicating, recording, and controlling instruments, using millivoltmeters or potentiometers as the indicating mechanism. Radiation elements are used not only in ranges up to 3300°F.—beyond the range of couples—but also in much lower ranges to measure temperatures of moving or otherwise inaccessible bodies. The **Féry pyrometer** uses a concave mirror to focus radiation on a small thermocouple, which is protected from direct radiation. The instrument is sighted on the radiating source through a telescope and the concave mirror moved backward or forward until the image of the source, as reflected by two small inclined mirrors placed near the thermocouple, becomes circular, indicating correct focus. In fixed-focus instruments, such as the **Thwing** (Fig. 80), the thermocouple is located near the apex of a conical mirror. Many radiation elements use lenses instead of mirrors to focus the radiations on the thermocouple or thermopile, sometimes mounted *in vacuo*. Lens types cannot be used below 800°F.

Radiation pyrometers are made in portable forms. One made by **Leeds and Northup** uses a lightweight radiation tube and an indicating potentiometer with neck strap. Another form, the **Pyro pyrometer**, made by the Pyrometer Instrument Company, is fully self-contained, the milli-voltmeter being built into the instrument. Still another form is the metal-spiral radiation pyrometer originated by Féry. Radiation is focused on a very small bimetallic spiral, which, in increasing in temperature, tends to uncoil and move a pointer on a scale.

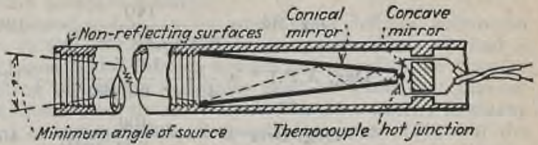


FIG. 80.—Thwing radiation pyrometer.

CONTROL BASED ON COMPOSITION

Several variables frequently met in plant operations depend primarily on the composition of the material. Where variation in the proportions of one or more components of a mixture (gas, liquid, or suspension) occurs, the variation may often be measured and controlled by measuring and controlling one or more properties of the material. Conversely, it is often possible to control these properties by controlling the composition of the mixture. If the properties can be measured, they can usually be controlled automatically. The principal properties of gases, liquids, and suspensions that are used in measuring and controlling composition are as follows:

Liquids

Density and specific gravity.
Viscosity.
Calorific value.
Electrical conductivity.
Hydrogen-ion concentration.

Gases

Density and specific gravity.
Humidity.
Calorific value.
Composition, by automatic analysis.
Thermal conductivity.
Refractive power.

Colloids and Suspensions

Consistency.

Density, Specific Gravity

Density is defined as weight per unit volume. Specific gravity of liquids and solids is the density compared with that of water at 4°C.; and of gases, density compared with air at 32°F. and 14.7 lb. per sq. in. abs. **Specific gravity** (or density) **of liquids** is determined: (1) by weighing a known volume, or weighing equal volumes of water and the liquid and comparing (pycnometer); (2) by determining the loss of weight of a plummet of known volume weighed in air and in the liquid, or by comparing the weight of a plummet of unknown volume weighed in water (at 4°C.) and in the liquid (**Westphal balance**); or (3) by means of hydrometers (see p. 776), *i.e.*, weighted glass floats which sink in the liquid to a depth dependent on the density, which is read at the liquid line on a calibrated stem extending above the liquid. Hydrometers are calibrated (1) in terms of specific gravity of liquid at 60°F., compared with water at 60°F. (called sp. gr. 60°/60°F.); (2) in percentage of a substance in a solution or mixture; or (3) in arbitrary divisions such as degrees Baumé (Bé.); degrees A.P.I. (American Petroleum

Institute); degrees Twaddell (Tw.), used in England; or degrees Brix (also called Fisher).

For liquids lighter than water:

$$\text{Deg. Bé.} = \frac{140}{\text{sp. gr. } 60°/60°\text{F.}} - 130$$

$$\text{Deg. A.P.I.*} = \frac{141.5}{\text{sp. gr. } 60°/60°\text{F.}} - 131.5$$

$$\text{Deg. Brix} = \frac{400}{\text{sp. gr. } 60°/60°\text{F.}} - 400$$

For liquids heavier than water:

$$\text{Deg. Bé.} = 145 - \frac{145}{\text{sp. gr. } 60°/60°\text{F.}}$$

$$\text{Deg. Tw.} = 200(\text{sp. gr.} - 1)$$

$$\text{Deg. Brix}\dagger = \text{arbitrary graduation}$$

To correct hydrometer readings for departures of the liquid from the calibration temperature, the coefficient of expansion of the liquid near the working temperature must be known.

Density recorders for liquids operate on a variety of principles. In the **Bailey gravity meter** (Fig. 81) several large volume displacers are hung from a balance beam within a pressure-tight tank and immersed in the liquid which is flowing through the tank. The beam is counterbalanced by a number of plates of less volume than the displacer, but of equal weight, hung from the other end of the beam. A pendulum, attached to the center of the beam at the fulcrum, causes the beam to assume a definite angular displacement for each specific gravity of the liquid. Temperature changes are compensated by using as displacers expansible containers filled with the liquid to be measured. The beam displacement is therefore independent of temperature. This displacement is communicated through a magnetic transmission to the recorder pen. The device may be used as a controller by the addition of electric contacts or pilot air valves.

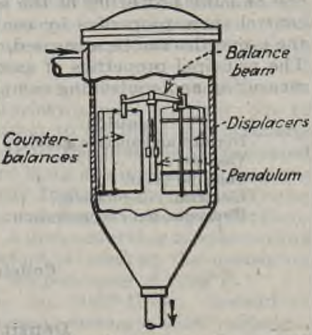


Fig. 81.—Bailey specific gravity meter.

The Anubis liquid gravitometer actually weighs a given volume of liquid. Liquid flows into and out of a counterweighted bulb through tubes acting as a spring balance. Movement of the bulb, with change in specific gravity, is transmitted through a linkage to a recording pen. Temperature compensation is provided in some models. For corrosive liquids the bulb and tubes are made of pyrex glass or some other suitable material.

Standard differential recorders may be connected to serve as gravity recorders by weighing a column of the unknown liquid. The methods of application are extremely varied to fit the variety of application conditions.

---

* Used in the United States principally for petroleum products.

† So graduated that 1° Brix = 1 per cent sugar in solution; used as a saccharimeter.

A simple and effective device for controlling the underflow of thickeners is the **Thyle density-control valve**. This device continuously weighs a certain volume of the sludge discharge and regulates a valve to control the rate of efflux to maintain constant density.

Gas density is measured ordinarily by weighing a definite volume in comparison with air. In the **Lux recorder** the gas passes slowly through a chamber containing a liquid-sealed floating bottom connected to the recorder pen and counterweighted to assume a definite position when the chamber contains air. This is shown diagrammatically in Fig. 82. In the **Ranarex instrument** (Permutit Company) (Fig. 83) the density of an unknown gas is compared with the density of a standard, such as air, by comparing the torque exerted on paddles by whirling columns of the two gases, set in motion by fans. Each gas flows continuously through a chamber. The comparison of torques is made by a mechanical linkage joining the two paddles and connected to the indicator pointer. This also operates a recorder pen. The device may be calibrated in terms of density (specific gravity) or in percentage of one component in a mixture of gases. Another type, the **American Meter gravitometer**, admits gas to a chamber at a pressure of 30 in. of mercury through a diaphragm-reducing valve weighted by air pressure at 30 in. of mercury absolute. Pressure differential across a synchronous motor-driven centrifugal fan is used as a measure of density.

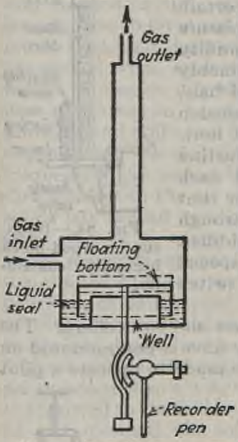


FIG. 82.—Lux gas-density recorder. (*Alpha Lux Co.*)

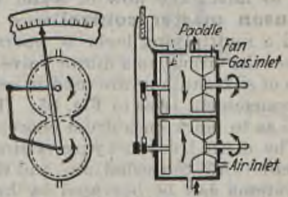


FIG. 83.—Mechanism of Ranarex gas-density recorder. (*Permutit Co.*)

Humidity

In previous sections (pp. 1079 and 1479) the methods and theory of humidity regulation and air conditioning were considered. It is proposed to treat here the mechanisms used to indicate, record, and control humidity.

The amount of water in the air may be determined as the actual quantity in a given volume (**absolute humidity**, expressed in grains of water per cubic foot of air or grams per cubic meter). Or it may be determined as the relation between the quantity of water in the particular sample and the quantity there would be in an equal volume of saturated air at the same temperature and pressure (**relative humidity**, expressed in per cent). It is well to bear in mind, however, that changes in temperature cause changes in volume, and hence this basis is a shifting one. Recent experimental work uses the weight of dry air as a basis. Absolute humidity is then expressed as grains or pounds per pound of dry air and **percentage absolute humidity** as the relation between the weight of water per pound of dry air in the sample and weight of water per pound of dry air were it saturated at the same temperature and pressure. The percentage relative humidity and the percentage absolute humidity will not agree and will differ more widely at the higher temperatures.

Relative humidity is the commonly accepted measure in industrial work. The numerous instruments developed to determine it may be divided into two groups: hygrometers and psychrometers.

Hygrometers (see pp. 1086 and 1087) depend for their operation on the expansion and contraction of certain hygroscopic substances with variations in the moisture content. This moisture content changes as the humidity of the surrounding air varies. The materials commonly used are wood, paper, silk, animal membranes, and hair. The **Bahnson humidifier** is controlled by the expansion and contraction of wood. Three blocks of pine, cut horizontally across the grain, are placed between the adjusting screw and the lever (Fig. 84). The blocks will each expand about ¼ in. for a change from 0 to 100 per cent humidity. This movement shifts the needle valve through the lever to affect the flow of water to the humidifier. The **Bahnson master control** uses a strip of special paper and a multiplying lever to operate a toggle switch which closes the circuit to a motor valve.

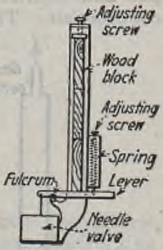


FIG. 84.—Bahnson wood-block humidity controller.

Strands of silk and hair are used in several recorders and controllers. The general arrangement is as in Fig. 85. The arm may have a pen mounted on the end so as to mark on a drum chart, or it may be made to operate a pilot valve. The membrane type is similar in arrangement. All the usual air- and electric-control systems can be operated by hair and membrane elements.

Hygrometers must be checked against some standard. Once set, these instruments will operate with fair accuracy over the ordinary working range. In general, hygroscopic materials do not expand exactly with the humidity. There is a falling off from the curve at temperatures in excess of about 150°F. For controlling, or for recording in a narrow range, this error is not of consequence. Care must be exercised to keep the elements clean and in a current of air to avoid the introduction of a lag in the readings.

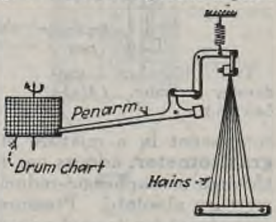


FIG. 85.—Hair hygrometer recorder.

Psychrometers function by reason of the drop in temperature due to evaporation of water. This drop varies directly with the rate of evaporation, and this, in turn, is inversely proportional to the amount of water vapor in the air. Thus, if a thermometer bulb is covered with a thin film of water and air passed rapidly over it, the temperature will fall a definite amount dependent on the initial temperature and the amount of moisture in the air, provided the air velocity is sufficiently great. The minimum is usually set at 15 ft. per sec. From the initial or dry-bulb temperature and the difference or depression of the wet bulb, the relative humidity may be calculated (p. 1085). Intermittent or check readings are usually made with a **sling psychrometer** consisting of two glass-stem thermometers mounted in some mechanism for whirling rapidly, one bulb being covered with a moistened cotton wick as in Fig. 86.

Records and control of humidity can be obtained by the use of two-pen thermometers of the types described on p. 2055, with one of the bulbs moist and the other dry. For this reason these psychrometers are frequently termed "wet- and dry-bulb thermometers." If the air circulation at the bulbs is not sufficiently great, a small motor fan may be used to suck air over the bulbs. Means for keeping the bulb moist have been the subject of much work and discussion. The most common is the cotton wick, a piece of cotton which has been boiled to remove any sizing or grease. The thickness seems not to be of so much importance as the porosity and ability to absorb water. Since this is dependent on the cleanliness, wicks must be changed frequently, the period varying with the amount of dirt and lint in the air and the amount of dissolved solids left behind by the water. The wick is made large enough to cover the bulb and to dip into a water reservoir mounted beneath it. The water is maintained at a constant level by a float valve, an overflow, or an inverted bottle. With an overflow, a needle valve should be connected in the water line to provide close adjustment.

FIG. 86.—Sling psychrometer.

A method of eliminating the changing of wicks is an alundum sleeve which fits over the bulb. Water is admitted to the inside and seeps through the pores. When properly installed and operated, the solids from the water as well as dirt from the process remains on the surface of the sleeve, whence it falls off or is easily wiped off.

The **Parks-Cramer humidity controller** employs no wick. The arrangement is as in Fig. 87, with air being sucked over the dry bulb by the spray, which saturates it before it passes over the wet bulb. The spray water must be at the approximate temperature of the wet bulb.

Leeds and Northrup have produced a direct-reading relative-humidity recorder operating from wet- and dry-bulb resistance-thermometer elements. A complex self-balancing Wheatstone-bridge circuit interprets the two separate resistance values in relative-humidity percentage.

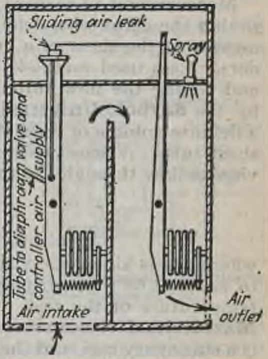


FIG. 87.—Parks-Cramer humidity controller.

Wet- and dry-bulb thermometers are naturally limited to temperatures between the freezing and boiling points of water. In addition, certain errors are encountered in both the lower and the upper portions of this range. These are not of great consequence where comparative readings only are desired in order to duplicate conditions.

The **automatic control of humidity** (see p. 1103) may be accomplished by the use of two-element temperature controllers of the types described on pp. 2019 to 2021 and 2056. The dry-bulb instrument or element then controls the temperature by a valve in the steam line to the heating coil. The wet bulb controls a valve in the water or steam line to the spray nozzles. Or, in circulating-air dryers, a damper may be shifted to govern the amount of saturated air taken from the circuit and fresh air admitted. Another method is to circulate the air through a chamber where it is cooled and saturated at the proper dewpoint and then heated to the desired temperature.

Absolute Humidity (see p. 1083). The actual moisture content of the air may be determined by absorbing the water vapor from a known volume of air or by observing the temperature at which dew forms on a polished metal surface (the dewpoint hygrometer). The **General Electric dewpoint potentiometer** was recently developed especially for determining humidity of gases at high temperature by the dew-point method. Gas is passed continuously over a polished metal mirror opposite a sight glass. Cooling of the mirror for low dewpoint temperatures is accomplished by cooling gas issuing from a jet. For high dewpoint temperatures the mirror is heated by a lamp. A fine wire thermocouple measures the temperature at which moisture appears on the mirror.

Viscosity and Consistency*

Viscosity of fluids is the measure of resistance to fluid shear at any temperature. Displacement of the fluid continues at a constant velocity, dependent on viscosity, as long as constant force is applied. In **true fluids,** there is no elastic limit that must be first overcome. In **plastic flow,** deformation will take place, but not beyond a certain limit until the force exceeds a definite yield value. This property is known as **consistency** and is exhibited by colloids and by suspensions. Very dilute suspensions, in general, show negligible variations from viscous flow, however. Concentrated suspensions, such as paper stock, exhibit a form of plastic flow, and the term "consistency" is the best that can be applied.

Measurement of viscosity is accomplished by two principal methods: (1) timing the efflux of a definite volume of liquid through a short tube; or (2) measuring the shear of a fluid in an annular space between concentric cylinders. Less used methods include timing the fall of a ball through a fluid and timing the flow through a capillary tube. Method (1) is represented by the **Saybolt Universal, Engler,** and **Redwood,** etc., viscometers where a definite volume of fluid at a definite temperature is timed in flowing from a short tube. Viscosity is expressed in seconds (Saybolt seconds, etc.). For viscous flow through the tube,

$$\frac{\mu}{\rho} = 0.22\theta - \frac{180}{\theta}$$

where μ/ρ is kinematic viscosity; θ, Saybolt seconds; μ, the viscosity relative to water at 68°F. (centipoises); and ρ, the specific gravity of the liquid at the temperature of the test. Method (2) is represented by the **Stormer** and **MacMichael viscometers** (see p. 1529). In the former the outer cylinder is a stationary cup, and the inner cylinder is timed for a number of revolutions under the application of a known force. In the latter the outer cup is rotated at known speed, and the force tending to turn the inner cylinder is measured in terms of the torsional stress in a supporting filament. These instruments are usually calibrated with water as unity.

---

* See Sec. 6, Flow of Fluids.

None of the instruments described above are well adapted to recording or to automatic control. Variations, however, of several of them are used. If a constant volume of liquid be passed continuously through a short friction tube, the magnitude of the upstream pressure, as shown by an indicating or recording pressure gage, is a measure of the viscosity at the existing temperature. By controlling the temperature or controlling the mixing of ingredients, viscosity control can be effected by such a device. The **Bradley-Osbourne viscosity controller** (Fig. 88), made by Thyle Machinery Co., is based on the principle of the MacMichael instrument. A cylinder driven by a motor through gearing is placed in the liquor conduit. Viscous resistance to the rotation of the cylinder tends to rotate the motor and gear case which are suspended from the cylinder shaft. As the motor is counterweighted to maintain a definite position when there is no load on the cylinder, the degree of rotation of the motor assembly about the shaft is proportional to the load, and hence to the viscosity of the liquid. Mercury contactor switches attached to the motor serve to open or close a heating or dilution valve and to maintain the desired viscosity (or consistency). Position of the motor may be recorded as viscosity on a position recorder.

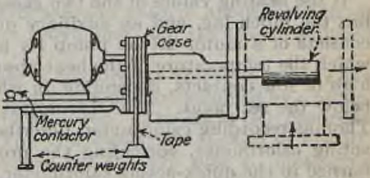


FIG. 88.—Bradley-Osbourne viscosity controller.

Consistency controllers, as used on paper stock, usually measure the torque required to rotate a paddle at constant speed in the stock. This measurement may be accomplished by means of a controlling ammeter or by any mechanical method of torque measurement. For control, the torque indications may be transmitted mechanically or electrically to a dilution valve and so control dilution rate as to produce constant consistency.

Calorific Value

This is used as a control property, principally in recording the quality of fuel gas; and in mixing gases, such as natural, coke-oven, or water gas. There are available a number of rather crude gas testers which determine heating value roughly as a function of density and calorific power, by determining the quantity of air required to burn the gas with a certain character of flame. In **calorimeters** the heat produced in burning a known value of gas is transferred to a known weight of water or a known volume of air, and the temperature rise noted. The flow of the heat-absorbing fluid is ordinarily continuous. Calorific value is calculated from the known weights and specific heats and the temperature rise. Several automatic and manually operated calorimeters are based on this principle. In the **Thomas calorimeter** (Cutler-Hammer, Inc.), air is the heat-absorbing medium as shown in Fig. 89. Separate interlocked water-sealed meters provide constant proportions of heat-absorbing air, gas, and combustion air, all at substantially atmospheric

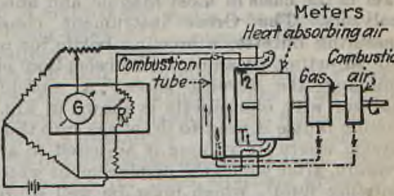


FIG. 89.—Diagrammatic arrangement of Thomas calorimeter. (*Cutler-Hammer, Inc.*)

pressure. The temperature rise of the air is measured in terms of calorific value of the gas by means of two resistance thermometers T_1 and T_2, connected to a Leeds and Northrup temperature difference recorder (see p. 2059). When two gases of different, but substantially constant, heating value are being mixed to produce a mixed gas of constant value, electric contacts on this calorimeter will control the mixing. One gas is supplied at constant rate and the other at a rate controlled by the calorimeter contacts which operate a reversing motor valve.

If the heating values of the two gases change rapidly, the control described has too much lag, and an auxiliary quick-acting calorimeter is used. This consists of a motor-driven pump for gas and air, and a combustion tube in which the temperature of the heat-absorbing air causes a thermostat to make high or low contacts, depending on whether the air temperature is above or below the set point. The thermostat controls the mixing valve, while the Thomas recording calorimeter, described above, keeps a check on the quick-acting calorimeter, correcting the proportion of heat-absorbing air to gas burned in the quick-acting calorimeter, if the latter tends to control too high or too low.

Some **combustible-gas detectors** are closely allied to calorimeters. In the **Bacharach, Mine Safety,** and **Union Carbide** types a metal filament enclosed in safety screens is connected as one arm of a Wheatstone-bridge circuit. When it is heated by the bridge current, any combustible gas in the surrounding atmosphere burns on its surface, raises its temperature, and changes its resistance. This is indicated on the bridge galvanometer. In the **Mine Safety carbon monoxide recorder,** the temperature of a catalytic material known as Hopcalite is raised by the oxidation of any carbon monoxide in gas passing through it at constant rate. This temperature is measured by a thermocouple and recorded as per cent carbon monoxide. Concentrations as low as 2 parts per million can be measured.

Gas Analyzers

A mixture of gases may be analyzed by measuring the volume decrease of the sample when several components are absorbed, one by one, in various solutions. The principle of the familiar **Orsat analyzer** is the basis of most manual and automatic gas analyzers. The Orsat instrument consists of a measuring burette, a leveling bottle for moving gas and adjusting pressure, and absorption pipettes for removing the several gases. A simplified Orsat analyzer with one pipette is shown in Fig. 90. The leveling bottle is used to draw gas into the measuring burette where its volume is adjusted to atmospheric pressure, after which the displacement water (or other confining fluid), which must be well saturated with the gases, is used to force the mixture successively into the several pipettes. During absorption of any component, when there is no further decrease in volume, this is taken as complete absorption. Total gas volume is measured after each absorption to show the percentage of that gas in the original sample. In the **Hays Orsatomat** gas is drawn into a measuring chamber by tilting the apparatus. The measuring chamber is open to the atmosphere through the outlet line. Tilting in the opposite direction causes the gas to pass into the absorption

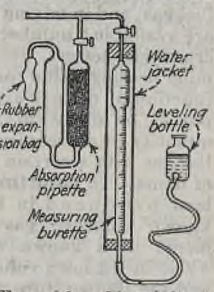


Fig. 90.—Simplified Orsat gas analyzer.

chamber, at the same time sealing the outlet line. Absorption of the absorbable gas causes a decrease in pressure and is indicated by the bellows which is used as a measure of the gas composition. For flue gas the first absorption pipette contains strong KOH solution (1 part KOH, 2 parts H_2O) for absorbing CO_2; the second pipette, alkaline pyrogallol (1 part pyrogallol and 3 parts water plus an equal volume of the above KOH solution) for absorbing O_2; and the third pipette, Cu_2Cl_2 for absorbing CO. This is made by mixing together in a large bottle containing scrap copper, 150 g. Cu_2Cl_2 and 1 l. 1.12 sp. gr. commercial HCl. The KOH solution absorbs about forty times its volume, the pyrogallate about twice its volume, and the Cu_2Cl_2 solution only about its own volume.

Automatic Orsats generally analyze a mixture for only one gas. They are operated by siphoning water, or by electricity, and are supplied by a pump, aspirator, or exhauster with the gas to be analyzed. The **Hays CO_2 recorder** uses an aspirator and water siphon. This is shown schematically in Fig. 91. With water at level 1, aspirator *A* draws gas through pipe *B*, measuring burette *C*, and tube *D* until the water level in tank *E* reaches 2, when it seals *B* and *D*. Water continues to rise toward 3, forcing the surplus gas into the rubber "atmospheric" bag *F*. At level 3 this is sealed off, and the sample, at atmospheric pressure, is of definite volume. As the level continues to rise toward 4, the gas in *B* and *C* is forced through capillary tube *G*, then to the KOH pipette where, as the KOH is displaced into the storage tank *I*, the CO_2 is absorbed by KOH retained in the steel-wool packing of *H*. The volume of KOH forced from level 5 up into the compression chamber *J* then depends on the percentage of CO_2 absorbed. The bottom of the calibrating tube *K*, which is open to the atmosphere, corresponds to the level 6 reached for the theoretical maximum CO_2 (20 per cent). Less CO_2 causes a greater rise to such a level as 7. Hence air is trapped in the upper part of *J* and a bellows *L* operates the recorder pen to register the actual per cent CO_2. Meanwhile, the water level has just reached 4, and the syphon *M* empties the tank to level 1, the cycle then being repeated. An automatic device, not shown, readjusts the KOH level to level 5, during the emptying of tank *F*, to make up for any losses.

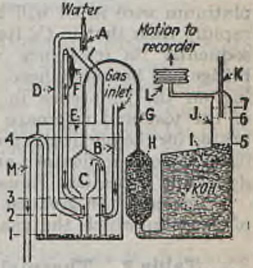


FIG. 91.—Mechanism of Hays automatic CO_2 recorder.

Other automatic analyzers are somewhat similar. The **Tag-Mono Duplex recorder** has a mercury-driven pump instead of an aspirator and measures the decrease in gas volume in a gasometer. In addition, at every alternate cycle, the instrument burns out any combustible gas in an electric furnace, and the apparent increase in CO_2 for these alternate cycles is a measure of combustible-gas content. Instead of absorbing CO_2 in KOH solution, the **Uehling Apex CO_2 analyzer** uses solid NaOH, and the decrease in gas pressure in the absorption chamber, at the entrance and exit of which there are small orifices, is taken as a measure of CO_2. A different principle is used in the **Defender portable CO_2 analyzer.** Gas is pumped into a chamber divided horizontally by a porous plate. A portion of the gas passing through the upper chamber diffuses through the porous plate and is absorbed in KOH contained in the lower chamber, creating a partial vacuum depending upon the CO_2 content. A float is used to transmit pressure variation.

Thermal Conductivity Analysis. Industrial gases differing considerably in thermal conductivity may be accurately analyzed by an instrument which determines thermal-conductivity rate in comparison with a standard. Table 3 shows the ratio of thermal conductivities of certain gases to that of air at 32°F. and at 212°F. (the latter in *italic* type). In using this method, two thermal-conductivity cells C_1 and C_2 (Fig. 92) are connected into two arms of a Wheatstone bridge. These two cells are usually built into a single block of metal in which are equal platinum resistance wires insulated from the metal. For corrosive gases glass and quartz cell units are available. As usually operated, one cell C_2 has sealed within it a reference gas, usually air. The other cell is connected in parallel with a pipe carrying the gas to be analyzed and part of this gas slowly drifts through the cell C_1. The bridge has two equal manganin ratio coils R_1 and R_2. Current from the battery B is passed through the bridge with the result that the platinum wires in the cells are heated 100° to 200°C. The bridge is first balanced with the slide wire S, using the same gas in both cells. When the gas to be analyzed is passed through C_1, if it differs from the reference gas in thermal conductivity, the platinum wire in C_1 will lose heat more or less rapidly than that in C_2, its temperature and consequently its resistance will change, and the bridge will be unbalanced by an amount depending on the difference in thermal conductivity of the test and reference gases. The indicating instrument must be calibrated for the particular gas for which it is to be used. Available instruments include both indicators and recorders, the latter in both the balanced and deflection types (see Resistance Thermometers, pp. 2057 to 2062).

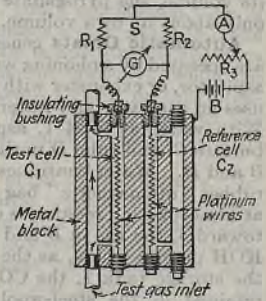


FIG. 92.—Cell and circuit for thermal-conductivity gas analysis.

In general, the greatest accuracy of analysis can be obtained in a mixture of gases in which the gas to be determined differs considerably in thermal

Table 3. Thermal Conductivities of Gases Referred to Air*

(See also pp. 959 to 961)

Figures in roman type, gases and air at 32°F.
Figures in italic type, gases and air at 212°F.

Gas	$\frac{K \text{ gas}}{K \text{ air}}$	Gas	$\frac{K \text{ gas}}{K \text{ air}}$	Gas	$\frac{K \text{ gas}}{K \text{ air}}$
Air	1.000 *1.000*	Ethane	0.876	Hydrogen sulfide	0.648 .....
Ammonia	1.15 *0.98*	Ethyl alcohol	0.708	Methane	1.127 .....
Carbon dioxide	0.585 *0.690*	Ethylene	0.698 *0.885*	Methanol	1.314 .....
Carbon disulfide	0.312	Helium	6.08 *5.53*	Nitrogen	1.004 *0.999*
Carbon monoxide	0.958	Hydrogen	7.35 *6.94*	Oxygen	1.007 *1.034*
Chlorine	0.370	Hydrogen chloride	0.635	Sulfur dioxide	0.415 .....

* *Natl. Bur. Standards, Tech. Paper* 249.

conductivity from the others. Table 3 will serve as a guide to indicate possible analyses. It should be noted, however, that oxygen can be determined in a mixture of nitrogen and oxygen, using air as the reference gas, but the differences are so slight as to require an apparatus of sensitivity too great for average industrial use.

Instead of using a sealed-in reference gas, as in the **single by-pass** type of **cell**, described above, a **double by-pass cell** substitutes a constantly changing reference gas. Single by-pass cells are used for determining one gas in a binary mixture or one gas in a mixture of several where the others differ considerably in conductivity. Double by-pass cells are used where the reference gas is necessarily changing, as in comparing a gas before and after a reaction; or it is used in analyzing a mixture of gases where more than one gas must be determined. Figure 93 shows how a mixture of hydrogen, carbon monoxide, carbon dioxide, and water vapor may be analyzed for carbon dioxide and hydrogen. Both cell units record on a single two-record recorder. Auxiliary equipment for thermal-conductivity analysis includes means for drying the gas, for maintaining a reasonably constant rate of flow (not over 20 cc. per min., *Natl. Bur. Standards, Tech. Paper* 249), and apparatus for sampling the gas flow. Automatic control is quite simply accomplished by adding high and low contacts to the recorder.

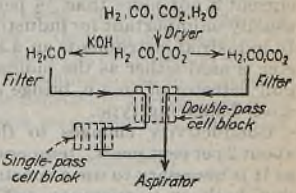


FIG. 93.—Thermal-conductivity method of analyzing a gas mixture for two components.

Analysis by Refraction. It is entirely possible to use the interferometer for analysis of gas mixtures, since the refractive power varies with the composition. However, this method is rarely used industrially because of the bulkiness of the equipment and the fact that the method depends on visual adjustment and hence (barring development of a photoelectric method) cannot record.

Electrical Conductivity

The property of electrical conductivity can often be used to determine the composition of an electrolyte. The specific resistance of an electrolyte is defined as the resistance in ohms of a column of electrolyte 1 sq. cm. in cross section and 1 cm. deep. The specific conductance is the reciprocal of specific resistance expressed in reciprocal ohms. As distinguished from the determination of hydrogen-ion concentration, the conductivity of a solution is a function of all the ions present in the solution. When the total number of ions increases, the conductivity increases, and vice versa. Hence the concentration in a solution of a single electrolyte, or group of electrolytes, may be readily determined. The method is widely used in determining the purity of boiler feed water or blowdown. It makes use of a conductivity cell (Fig. 94) containing the liquid or submerged in the liquid. The two electrodes of the cell are usually of platinum or gold, connected into one arm of an a.c. Wheatstone bridge. The resistance of the cell as shown by the instrument depends on the solution and the characteristics of the cell. For low conductivities the electrodes are placed close together and farther apart for high conductivities. To facili-

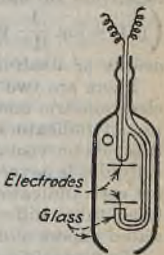


FIG. 94.—Electrical conductivity cell.

tate choosing the proper electrode area and spacing for a given bridge and solution, a cell constant is specified, given by the expression

$$R = \frac{C}{K}$$

in which R is the resistance measured in ohms, or maximum resistance measurable by the bridge; K is the expected minimum conductivity of the electrolyte to be measured; and C is the cell constant. Alternating current is used to reduce polarization. Theoretically, a high frequency is desirable, but the error introduced by using 60-cycle current is not more than ½ per cent and is usually unimportant for industrial purposes. In industrial instruments an a.c. galvanometer is used either as the indicator or as the null detector of an a.c. bridge circuit, often a self-balancing type.

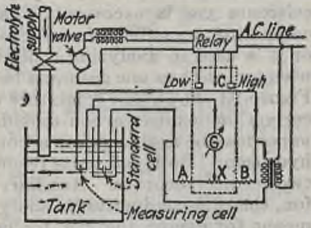


FIG. 95.—Electrical conductivity controller. (*Leeds and Northrup.*)

Conductivity changes to the extent of about 2 per cent per °C. temperature change; so it is necessary to use a constant temperature bath or a reference cell. Figure 95 shows a **conductivity controller** adapted to maintain a given concentration of electrolyte bath. Solution of the required composition is placed in a sealed conductivity cell, which is submerged in the bath to eliminate effect of temperature change. The self-balancing bridge makes low or high contacts to control the addition of electrolyte.

Hydrogen-ion Control

A great many processes are now carried out under pH control. To mention a few, pH control is of utmost importance in sugar manufacture (to prevent inversion); in paper manufacture (in controlling size precipitation); in electroplating; sewage and trade-waste disposal; food manufacture; tanning; preparing clay slips; all settling operations; and in selective flotation. pH is a measure of the hydrogen-ion concentration and is expressed as the logarithm of the reciprocal of the gram ionic hydrogen equivalents per liter $\left(\text{pH} = \log \frac{1}{\text{H}^+} \text{ per l.}\right)$, so that pH is a measure of **effective** and **not total** *acidity* or *alkalinity*.

There are two general methods of pH measurement: (1) the fundamental electrometric method, and (2) the derived colorimetric method based on the use of indicators and permanent color standards. As the second method depends on visual comparison of color, it cannot record or control automatically but is nevertheless most used for control purposes. The usual pH range for each indicator is about 1.6 pH. The color standards are usually liquid-filled tubes differing by about 0.2 pH. Although not so accurate, impregnated papers and colored glasses are sometimes used.

Electrometric pH measurement depends on the fact that two suitable electrodes in contact with a solution will develop a potential proportional to the hydrogen-ion concentration. This potential may be measured with a potentiometer. The test electrode develops a variable potential dependent on the pH of the test solution, while the other (the reference electrode) develops a constant potential independent of the pH. The hydrogen test or measuring electrode has been displaced by new types. The hydrogen electrode is a piece of platinum coated with platinum black, over which hydrogen gas is bubbled. It is not adapted for industrial use due to its susceptibility to poisons, errors caused in either oxidation or reduction systems, inaccuracies in the presence of metals below hydrogen in the electromotive series, and difficulty in adaption for continuous measurement. Quinhydrone, glass, and metallic oxide electrodes are those most commonly used for industrial purposes. The quinhydrone electrode simply consists of a platinum or gold electrode immersed in the sample solution which is saturated with quinhydrone. Its main disadvantage is that the solution under test must be saturated with quinhydrone, and it is accurate only to a pH of 7, although it can be used up to a pH of 9 for less accurate work. The most frequently used metallic oxide electrode is that of antimony. Its main advantage is its ruggedness, but it is subject to salt errors, to oxidation or reduction system errors, to change in calibration due to movement of solution, and to an over-all limit of error of ±0.2 pH. The glass electrode consists of a glass tube with a thin bulb blown at one end. This is filled with an electrolyte in which an inner platinum electrode is immersed. The glass itself acquires a surface potential when in contact with a solution, depending upon the pH. Owing to the high resistance of the glass itself, generally a vacuum-tube amplifier has to be used in the measuring circuit. Although this electrode is inaccurate in the presence of sodium salts above pH value of 9.5, it is adapted to oxidation or reduction systems, thick fluids, and continuous recording.

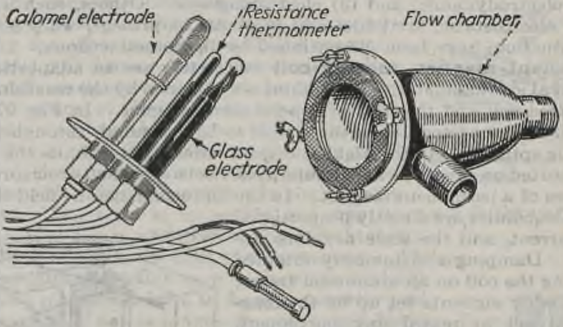


Fig. 96.—Beckman glass-electrode flow-chamber assembly.

For reference, the **calomel electrode** is nearly always used. This consists of mercury in contact with calomel, which is, in turn, in contact with a solution of KCl saturated with calomel. A gooseneck tube serves to make electrical contact with the solution surrounding the measuring electrode. The tube is often closed with an alundum thimble, or a ground joint cap, to reduce diffusion.

For industrial work the two electrodes are usually combined, and for continuous measuring a resistance thermometer or thermopile temperature compensator is added to the combination which has a considerable temperature coefficient. The assembly is usually mounted in a flow chamber. A potentiometer recorder is used where a record of the pH is to be kept. The instrument may be equipped for automatic control. In the quinhydrone-flow type of electrode, means must be provided for the continuous addition of a small amount of quinhydrone solution. According to the Leeds and Northrup Co., 6 to 8 drops of a saturated solution of quinhydrone in acetone suffices for 50-cc. test solution, or the test solution may be passed slowly over quinhydrone tablets. It is to be noted that the temperature has a considerable effect on pH measurements. Figure 96 shows the **Beckman** glass-electrode flow-chamber assembly open. Shielded leads from the electrode assembly connect with the electronic amplifier from which standard indicators, recorders, and controllers may be operated.

ELECTRICAL MEASUREMENTS*

Control and measurement of electricity are far too voluminous subjects to be covered thoroughly in this brief article. Therefore only information likely to be of assistance to the chemical engineer in plant work has been included. For laboratory instruments, power-plant equipment, and theoretical considerations, the reader is referred to the many standard works available.

Measurement of Voltage. There are three fundamental types of voltmeter now being furnished for industrial use: (1) permanent-magnet, moving-coil, (2) electrodynamic, and (3) electromagnetic. Others, such as the hot wire and electrostatic, are little used in plants at present, while some, such as the induction, have been discontinued by the manufacturers.

Permanent-magnet, moving-coil voltmeters are an adaptation of the **D'Arsonval** galvanometer. Deflections are obtained by the reaction between an energized coil and the field of a permanent magnet. In Fig. 97 the coil of fine wire is connected across the circuit to be measured, through the spiral restraining springs. (In the higher ranges a series resistor limits the current.) It is supported on pivots in the annular space between a soft-iron core and the pole pieces of a permanent magnet. In the uniform magnetic field the deflections of the pointer are directly proportional to the current, and the scale divisions are uniform. Damping is customarily obtained by winding the coil on an aluminum frame, in which eddy currents set up by the magnetic field act to retard any movement. This element is standard for d.c. potential indications and to a great extent for recording. The ranges may be as low as 0 to 1 millivolts. For ranges below 0 to 100 millivolts on recording instruments the power available is usually so small as to necessitate the use of a depressor bar or other means of eliminating pen friction. When used on these low ranges, this element is termed a "millivoltmeter." It is utilized in pyrometers, conductivity recorders, and other instruments which measure other variables by means of a direct current.

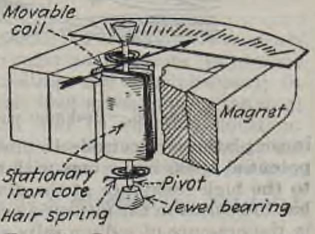


Fig. 97.—Permanent-magnet, moving-coil voltmeter.

Electrodynamic, or **moving-coil,** voltmeters depend for their operation on the reaction between the fields of fixed and moving coils connected in series. In consequence, the pointer deflections will increase according to the square of the current. All these movements may be used on either alternating or direct current, although they are generally less accurate with the latter. The **dynamometer** type has the moving coil within the fixed coil or coils, as in Fig. 98. It is used in a.c. indicators and strip-chart recorders. Another variation of the electrodynamic voltmeter is the **Kelvin balance,**

---

* See pp. 2017, 2019, and 2653.

shown diagrammatically in Fig. 99. Two coils are balanced on a knife-edge so that they are free to move between two sets of fixed coils. When connected in series, the movable coils take up a position dependent on the current strength. **Westinghouse strip-chart recorders** operate on this principle with a contact-making device to control the movement of a motor-driven pointer. The **attracting-coil** element uses but two coils in series. The movable one is wound on a horizontal shaft which passes through the fixed coil and is supported at the ends by knife-edge springs (Fig. 100). With an increased current the coils attract each other, and the pen arm is deflected. This element is used in some **Bristol round-chart recorders.** The scale is increasing, so that the best working range is in the middle third of the graduations. Where damping is desired, an oil pot is mounted beside the movement. A vane fastened to the shaft of the movable coil dips into the oil and slows the movement. Oils of various viscosities may be used to obtain the amount of damping desired.

FIG. 98.—Dynamometer movement for voltage measurement.

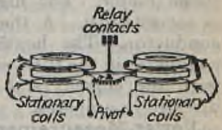


FIG. 99.—Kelvin-balance movement for voltage measurement.

Electromagnetic, or **moving-iron,** voltmeters function by the effect on soft iron of the magnetic field around a coil through which current is flowing. The chief difference between various forms is in the shape and location of the iron. In the **repulsion-iron** instruments two thin pieces of soft iron are placed within a coil. Both become similarly magnetized, and, since only one is movable, it is repelled by the other with a force proportional to the strength of the field. The scale is increasing but may be made fairly uniform by properly shaping the iron. Alternating-current instruments only are equipped with this movement. The **Weston** instrument of this type is shown in Fig. 101. The soft-iron pieces are bent to form concentric arcs, and the stationary one is tongue-shaped to get better scale characteristics. An air damper is mounted on the shaft of the movable element. It consists of a vane in a box with very small clearance so that air pressure will be set up to oppose any movement. Westinghouse uses two rectangular strips which open like a book with current increase. This element will withstand a considerable overload. The **General Electric inclined coil** uses a coil mounted at 45 deg. to the vertical shaft which carries a soft-iron vane. This gives movement of the pointer through an arc exceeding 90 deg. The **attracting-disk** element is constructed the same as the attracting coil, described above, except that the moving member is a soft-iron disk. The scale is nearly uniform in the upper part of the range, and hence this movement is employed in Bristol round-chart recorders for narrow scales starting above zero, as 90 to 140 volts, alternating current and direct current. **Solenoid** or **soft-iron plunger** elements are supplied in several makes of round-chart recorder for both alternating and direct current. As the

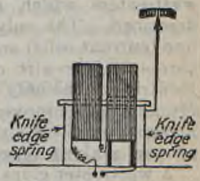


FIG. 100.—Attracting-coil element for voltmeters.

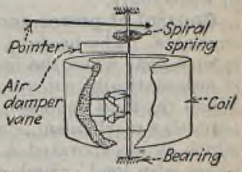


FIG. 101.—Weston repulsion-iron voltmeter.

name implies, a soft-iron plunger is pulled into a long coil or solenoid as the current increases. The scale which results increases as the square of the current. An oil damper may be used. The Westinghouse round-chart recorder has a **laminated-iron armature** which moves between the pole pieces of an electromagnet and is restrained from rotating by spiral springs. Magnetic damping is used.

Measurement of Amperage. In general, ammeters are of the same construction as voltmeters, except that relatively few turns of large-diameter wire are used instead of many turns of fine wire. Thus the resistance is low, although the number of ampere-turns is approximately the same. All types of element described under voltmeters are used for current measurements, except the attracting coil. Any d.c. ammeter may be shunted for higher capacity, but the more common practice is to design for a definite potential drop of 50, 75, 100, or 200 millivolts. This is then measured by a millivoltmeter calibrated to read directly in amperes. Alternating-current ammeters cannot be shunted, and, when the current (or potential) becomes unsafe for a self-contained instrument, a transformer is used with a 5-amp. range meter. For high-frequency circuits the **thermocouple ammeter** is satisfactory. A thermocouple is enclosed in an evacuated tube with a conductor. The heating effect of the current, as it passes through the conductor in the tube, sets up a potential in the thermocouple. This increases with the current and may be measured with a millivoltmeter calibrated directly in amperes.

Power Measurement. In d.c. circuits the power value may be obtained by measuring the voltage and amperage separately. With alternating current a power-factor variation from unity occurs with any inductive or capacity load. Hence a **wattmeter** must be used for loads other than incandescent lamps and electric heating units, the **dynamometer** and **Kelvin balance** wattmeters which are similar to the voltmeter movements previously described. The only change is to connect the fixed coils in series with the line (current coils) and the movable coils across the line (potential coils). For three- or four-wire circuits a double element is used in the dynamometer. Rewiring is the only change necessary on polyphase with the Kelvin balance. Dynamometer movements are found in both indicating and recording wattmeters, but the Kelvin balance is used only for recording instruments of the contact-making type.

A wattmeter gives the instantaneous measure of energy, or a curve showing the rate at which it is being expended. The total energy used is obtained by means of a **watt-hour meter**, sometimes called an **integrating meter**. This is essentially a small motor whose speed is proportional to the power consumption. The moving element operates a dial register to give the total in watt-hours.

Control of power consumption may be effected by several devices now available. The hook-up differs with each installation, but the arrangement is such that when the load exceeds a set figure, the least important portion is automatically cut out. A further increase cuts out the next; and so on. When the load decreases, the operations are progressively started up. The actuating mechanism is a contact-making wattmeter or demand meter, with which is incorporated a timing switch to prevent action with momentary increase in load. Such control is especially desirable where the power contracts sets the price according to the maximum demand.

Power Factor (see p. 2642). Knowledge of the power factor is necessary since most power contracts contain a clause penalizing for any marked deviations from unity. It may be determined by instruments similar to either the dynamometer or the Kelvin-balance wattmeters. In the former the difference is in having two moving coils crossed at right angles to each other. An inductance is placed in the circuit of one and a resistance in the other. The Kelvin balance actually measures the angle through which a phase shifter must be turned to get zero power factor in the measuring element. The **correction of power factor** may be effected by the substitution of synchronous motors for all operations where they can be used. Sometimes an overexcited synchronous motor, or synchronous condenser as it is then called, is allowed to float on the line for this purpose only. For smaller installations a capacitor, or static condenser, is commonly connected in the motor circuit.

Frequency. Determinations of frequency are generally measured by setting up an unbalance in the circuit. Either the cross-coil dynamometer or the Kelvin-balance movement, as used in power-factor measurement, is connected in two opposing circuits with a resistance in one and an inductance in the other. The current in the resistance is dependent on the voltage alone, while that in the inductance depends on frequency and voltage. The resultant of the pulls of the two coils will vary with the frequency. This method is used also in the Bristol which employs the same element as in the inductance balance for long-distance transmission (p. 2026). These instruments are accurate to about one-tenth cycle. Interconnection of power systems has caused a demand for instruments which will read closer, and several have been developed. The **Leeds and Northrup recorder** has a circuit composed of an impedance bridge, two arms consisting of non-inductive resistances, and two of non-inductive resistances together with condenser units. In one of these arms the condenser is in parallel with the resistance; in the other it is in series. The bridge is of the balanced or null type, balancing being accomplished by means of a contact-making galvanometer. The **Westinghouse centrifugal frequency recorder** employs a synchronous motor to drive a flyball-contacting device which starts, stops, and reverses a separate motor for moving the pen arm.

MISCELLANEOUS MECHANICAL CONTROLS

Under this heading there will be considered the means of indicating, recording, and controlling mechanical operations. The variables are **speed, position, thickness,** and **time.** In addition, **photosensitive devices** will be described.

Speed. Measurement of speed assumes great importance because of the secondary factors dependent on it, *e.g.*, voltage or frequency in electric generators, or pressure of blowers. Devices are now available for momentary-check readings, for continuous indication, for curve drawing, and for controlling, using both mechanical and electrical means.

Check readings may be made with an indicating tachometer designed to be held in the hand while testing. It is fitted with a pointed metal or rubber-tipped shaft, which is held against the center of the rotating part whose speed is to be measured. One type actually counts the number of revolutions in a fixed period of several seconds. The pointer moves around the dial, which reads in revolutions per minute, and stops automatically at the end of the interval. Another type operates like a flyball governor. Weights are so mounted on a shaft that they, revolving with it, tend to fly off as the speed increases the centrifugal force. This force is resisted by springs, and the resultant motion is transmitted to the pointer. This same principle is incorporated in some stationary indicating, recording, and controlling tachometers.

The instrument shaft is in this case driven by a belt from the main shaft instead of being pressed against it. Recorders have also been built of the pneumatic type, which depends on the vacuum created in a T-shaped tube when it is turned in a closed chamber on the axis of its vertical arm.

Electric tachometers are now most used for stationary work. They have been developed to a high degree of accuracy. A d.c. magneto of special design is driven by a belt, gear, or chain, or through direct connection from the rotating part. The potential of the magneto terminals will be a straight-line function of the speed. This may be measured with an indicating or recording voltmeter calibrated to read directly in revolutions per minute.

Of great importance in connection with speed, although not primarily a method of determining it, are the **stroboscopes** which permit the study of mechanisms while in rapid rotary or reciprocating motion. They depend on the ability of the eye to retain an image for an appreciable time. If an object is seen intermittently, the effect of a continuous view will be had when the interval between glimpses is less than the time the eye will retain the image. If the interval is too great, the image will flicker. A rapidly rotating object seen at intervals equal to the time of one revolution will therefore appear to be stationary, as it will be in the same position each time it is viewed. If the interval is slightly greater or less, it will appear to be moving forward or backward very slowly. By changing the interval, the apparent speed of the part may be varied at will. This makes possible a thorough study of the action of fast moving machinery. The methods of accomplishing this result are by interrupting the vision and by interrupting the light. The former is done by observing the part through a slotted disk or cylinder which may be turned at a set speed The light outfit consists of a contact breaker in the light circuit, driven by the machine under observation, or by a separate motor. A neon light is ordinarily used so that the object may be viewed in daylight or artificial light without interference. The flashes are then synchronized or put slightly out of step as desired.

Position. When working with inaccessible or remotely operated gates, dampers, and valves, or with pulp beaters and other equipment, it is of value to know their exact position. This is accomplished by the use of a system of levers, sometimes in conjunction with cables and pulleys, to move a pointer on a scale. The arrangement is such as to get the proper reduction of the movement for full-scale deflection. These motion recorders may be fitted with remote transmitting and receiving systems so that the reading is available at a distant point, *e.g.*, at the control switchboard. The position may be altered by manual control, push-button motor control, or automatic control.

Thickness. Variations in thickness, as of the cake on a rotary filter, may be recorded by the motion recorders described above. A roller or shoe riding on the material is connected to the levers. These instruments are limited as to accuracy on very thin materials, such as rubber sheeting or paper, but several other devices have been developed for these applications. The **Schuster magnetic gage**, designed originally for rubber calender rolls, consists of an electromagnet mounted on wheels so as to ride on the stock. The distance between the roll and the magnet varies with the thickness of the stock, and the current flowing through the magnet coils will vary with this distance. These changes in current are read directly as thickness in ten-thousandths of an inch.

Another device, designed to regulate thickness of rubber coatings is the **weight Verigraph.** This is essentially a tuned-radio circuit. The web of material passing between two condenser plates varies the capacity. A null-type recorder shows deviations from the correct thickness.

Time. Instruments are available for recording both the duration and the frequency of movements. One type, the **electric operation recorder**, functions by having an electric contactor so arranged that the motion will make and break the circuit to a small magnet coil. Mounted on its pole piece is a pen arm which draws a short line at right angles to the direction of chart travel for each change. These are used to show the starting and stopping of looms, motors, presses, and for innumerable other operations. The exact means of making the contacts will differ with each installation. One operation recorder is equipped with a timing device to ring a bell or light a lamp when the operation is finished. This contains a time switch similar to those used separately to stop an operation automatically. Another type is usually called an **auto time recorder** because of its common use on automobiles. It has a waxed or sensitized chart across which the pointer moves when its support vibrates, as with the running of the engine.

Photosensitive Cells. Certain metals emit electrons when exposed to light, as do others when heated. The latter form the basis of the electron tubes of radio, by assisting the passage of current between electrodes made of these thermionic materials. If light-sensitive metals be used, the current passing in the circuit will vary with the amount of light received by the electrodes, whence the intensity of light may be measured with great accuracy. The metals which have this photoelectric property are sodium, potassium, cesium, rubidium, lithium, strontium, and barium. Of these, cesium is now generally used because its sensitivity more nearly coincides with the range of human vision. A common design of cesium cell is shown in Fig. 102. The V-shaped cathode is coated with cesium, and the bulb may be either evacuated or gas filled.

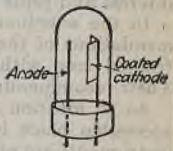


Fig. 102.—Photoelectric cell for concentrated light.

The electrical output of photoelectric cells ranges from a fraction of one microampere to several microamperes. It may be amplified by several three-electrode vacuum tubes of the ordinary radio types in series, and so measured with a microammeter or used to operate a relay for control purposes. Power-relay tubes, such as the **Grid-glow** and the **Thyratron,** combine sensitivity with considerable power output. Complete units of sensitive-cell, power-tube, and relay switch are now available.

Recently interest has been revived in the companion phenomenon of photoconductivity, first used in the selenium cell whose resistance changes with the intensity of the light. Selenium showed a considerable lag which is not found in the new selenium and copper oxide cells where a thin layer only is used.

The **Weston photronic cell** has a thin metal disk serving as the positive terminal, upon which a thin film of light-sensitive material serves as a negative terminal. Contact with the light-sensitive material is made through a metallic collector ring. A filter is used when the spectral sensitivity must conform with that of the human eye. Available in waterproof housings and having a high output—several hundred microamperes under intense illumination—which eliminates the need for amplifiers in most applications, barrier layer photocells are widely used in industry. However, for high frequencies and sensitivities the more fragile photoemission cells are used.

Applications of photoelectric cells may be grouped as sorting, usually dependent on the light reflected from the surface under examination; counting, by cutting off a beam of light; measuring radiation, as with hot bodies and daylight; and analyzing color or composition by transmitted or reflected light. Considerable experimenting may need to be done to find the optimum conditions for each application.

SUPERVISION OF INSTRUMENTS

Details of the operation and maintenance of control instruments naturally are too voluminous for treatment here. The manufacturer's direction sheets should be read carefully. They have been written in the light of past experience, and, if followed, no difficulties should be encountered. One general rule may be given here: Do not allow any person to touch an instrument except for *necessary* maintenance work. The great majority of trouble cases come from tampering by inexperienced persons. It is advisable to make some one person responsible for all instruments in the plant, whether there be one or many. For this work some handy person can usually be found. He should be given time to study over the directions and familiarize himself with maintenance details. The manufacturer's serviceman can undoubtedly help in this. Some companies have sent their maintenance men to the factories where the instruments were made. The manufacturers are willing and anxious to show them through their factories and shops and to give instruction on the work which can be done in the field. Aside from the actual knowledge gained, this has the advantage of arousing a keen interest and pride in the work.

In the selection and location of instruments it is well to get the recommendations of the manufacturers' salesmen. As a group they are factory-trained men with a background of engineering education and experience. Their recommendations merit serious consideration.

As to location of instruments it may be said in general that the most accessible place is the best. It cannot be too strongly emphasized that instruments are primarily guides for the operators, not checks on their derelictions. For this reason they should be located as near the operator's regular station as practicable. It will then not be difficult to get the operator to consult them frequently and to depend on the readings. Centralization of all instruments is usually desirable. Although the cost is ordinarily greater, it does make for a good-looking installation. Operating efficiency is enhanced if all information and controls, manual or automatic, are conveniently at hand. Needless to say, there is little gain in centralization of instruments if the process operation is not otherwise arranged for centralized operation. Where a central control board is used, it should be in the most accessible place, or the center of population, as it might be described. In older plants location of indicating and controlling instruments beside the operations with recording instruments at some central point is often good practice. In such cases remote manual controls are often advisable to permit immediate adjustments.

Automatic control has one inherent danger: so much dependence is placed on it that its failure may be disastrous. For this reason all possible safety devices should be used. On air-operated controls the valve operation should be selected so that the valves will assume the safer position on air failure. Also safety interlocks may be needed. On electrical controls, a change in the wiring will often suffice, *e.g.*, connection of the blower, the oil pump, and the control instrument on one circuit. A failure of one means the failure of all. A manually reset relay, in addition, will make certain that the attendant is present when current goes "on" again. Solenoid valves may be placed in a line to cut off with current failure, or working with a controller, to cut off for overtemperature. Switches which break an electric circuit on failure of pressure in air, gas, or oil lines are available. Numerous schemes have been worked out or can be devised easily, but this matter should receive careful consideration.

Charts from recording instruments deserve thorough study. Comparison should be made with records of previous days and years and checked against the results of the day's run. The records, if carefully marked with batch numbers and filed, will assist in assigning the cause of failure of returned material. Instruments are additional tools and fact finders and give benefit according to the amount of use.

SECTION 18

MATERIALS OF CONSTRUCTION*

BY

James A. Lee, B.S., B.A., M.A., Managing editor, *Chemical & Metallurgical Engineering;* Member, American Institute of Chemical Engineers, American Coordinating Committee on Corrosion, American Chemical Society, American Society for Testing Materials, Technical Association of the Pulp and Paper Industry, American Electrochemical Society, Royal Society of Arts.

W. S. Calcott, Ch.E., Director, Jackson Laboratory, E. I. du Pont de Nemours & Company; Member, American Institute of Chemical Engineers, American Chemical Society.

REVISED BY

A. F. Benning, B.S., Organic Chemicals Department, E. I. du Pont de Nemours & Company.

James A. Lee, B.S., B.A., M.A., Managing editor, *Chemical & Metallurgical Engineering.*

H. L. Maxwell, Ph.D., Metallurgist, Engineering Department, E. I. du Pont de Nemours & Company.

F. L. Whitney, Jr., B.S., Engineering Department, E. I. du Pont de Nemours & Company.

CONTENTS

---

*Largely from material published by *Chemical & Metallurgical Engineering*, September, 1940.

MATERIALS OF CONSTRUCTION

Introduction. Materials for chemical-plant construction fall naturally into two general classes, **metals** and **non-metals**, the former class being further divisible into pure metals and solid-solution-type alloys on the one hand and non-homogeneous alloys on the other. The latter distinction is an important one as, for corrosion-resisting purposes, the solid-solution alloys frequently behave as though they were chemical individuals of definite characteristics, while the mixed-crystal type usually has roughly the resistance of the poorest constituent, the resistance being still further diminished by the presence of numerous electrolytic cells. The difference in behavior between properly and improperly heat-treated metals of the stainless-steel class well illustrates this characteristic difference between the solid-solution and the mixed-crystal types.

In the selection of material for the construction of a chemical plant, resistance to the corroding medium is usually the determining factor as otherwise the choice would fall automatically on the cheapest material mechanically suitable, and laboratory corrosion tests are frequently the quickest and most satisfactory means of arriving at a preliminary selection of the suitable substances. Unfortunately, however, it is not possible on the basis of any existing laboratory test to predict with accuracy the behavior of the selected material in the plant. The outstanding difficulty in this connection lies not so much in carrying out the test as in interpreting the results and translating them into terms of plant performance. A laboratory test of the standardized type gives but one factor, the chemical resistance of the proposed material to the corrosive agent, and there are numerous other factors entering into the behavior of the materials in the plant.

The following method of determining this factor, which is the so-called total immersion test, represents an unaccelerated method which has been found to give reasonably concordant results which are in approximate agreement with results obtained on the large scale when the other variables are taken into account. Various other tests have been proposed and are in use, such as the salt spray, the accelerated electrolytic, the alternate immersion, the aerated total immersion, etc., but in view of the numerous other complications entering into the translation of laboratory results into plant results, the simplest test would seem the most desirable for routine preliminary work, reserving the special test methods for special cases. This preliminary test serves quite well to eliminate the materials which obviously cannot be used; further selection among those which apparently can be used can be made on the basis of a knowledge of the properties of the materials concerned and the working conditions, or by constructing larger scale equipment of the proposed material in which the operating conditions can be simulated.

THE TOTAL IMMERSION TEST

The following conditions and factors affecting the results of tests have been investigated:

Test Piece. The shape of the test piece does not affect results within a reasonable range of ratio of length of edge to surface. Determinations have been made for ratios between 2 and 10. Fine polishing is unnecessary, although tool marks and oxide coatings should be removed. Many determinations have been made on roughened and smoothed pieces.

Volume of the Corroding Solution. Investigations using volumes of solution ranging from 20 cm. to 200 cm. per sq. in. of test piece exposed show that the effect of the volume of the corroding solution becomes practically negligible if a relation of volume to square inch of test piece in excess of a certain minimum is used.

Temperature of Corroding Solution. Results of many experiments show that the temperature coefficient is high for all reactions. For theoretical work in determining the law of temperature, it is absolutely necessary to use an accurately regulated thermostat. In applying results to plant practice, similarly careful consideration must be given to temperature control in all specific tests. In comparing different metals, care must be taken that the temperature is the same for all tests.

Time of Exposure. It has been found that results of corrosion tests vary enormously with the time of exposure of the material to the corroding solution. This variation is due to initial electrochemical surface actions such as overvoltage and to the period of time required for the formation of a protective coat. Evidently initial effects of this kind must be neglected if a corrosion factor for use over a long period of time is to be obtained. The logical way is to measure the rate over an interval of time after this initial high corrosion rate has decreased and become constant. For the most careful work this must be done. The experimental data available at present indicate that corrosion should be determined at the end of 48, 96, and 240 hr. Sufficient accuracy for comparative work may usually be obtained by neglecting the corrosion over the first 48 hr. and averaging the rate over the second 48-hr. period.

Cleaning Test Pieces. Many methods have been investigated for removing coatings from the test pieces. They may consist of dissolving the coat or of removing it mechanically. In each case the error due to a particular method has been determined. The first 48-hr. immersion in the corroding solution may be considered a preparation of the test piece for the final test.

Reagents and Apparatus. Corroding Solution. Use 250 cm. of the corroding solution per test piece of given area (4.6 sq. in.) for a fairly rapid corrosion rate (0.01-in. penetration per month). The volume should be increased in proportion for pieces of greater area.

Preparation of the Sample. 1. Metal Test Piece. Size 2 by 1 by 0.1 in. (area 4.6 sq. in.). Dimensions should be accurate to 0.01 in. in order to save the time of measurement and area calculation in the laboratory. Other shapes may be used within a range of ratio of length of edge to total area of test piece between 4 and 8.

2. Preparation. The strip of indicated size may be cut from flat sheet metal or turned from pipe. Tool marks should be removed by successive use of file and emery. Exceedingly fine finishes are unnecessary, but the surface should be clean and reasonably smooth as, for example, finished with 120 grain emery.

Detailed Procedure. Standard Static Corrosion Test. Place the corroding solution in a flask or wide-mouth bottle and bring to the temperature of the test. Maintain this temperature to at least 1°C. in a carefully regulated thermostat. Suspend the test piece upon a glass hook from a stopper of such a size as to fit the flask or the bottle loosely. The stopper must be made of a material which will be unaffected by the corroding solution. Immerse the test piece in the solution when it reaches the proper temperature, closing the bottle or flask with the stopper. If the test is being made at a high temperature, a reflux condenser must be used, taking care to prevent the condensate running directly on to the test piece.

The highest accuracy is desirable. Expose the test piece for 48 hr. to the corroding solution. Remove from the solution, wash thoroughly in a stream of water, and remove any coating. This cleaning may be done by dissolving off the coating (lead sulfate in ammonium acetate solution, lead chloride in hot water, etc.) or by rubbing and scouring with or without a soft powder as a mild abrasive. Do not adopt any method of cleaning until the error due to its use has been determined. Weigh the test piece after thoroughly drying, especially wiping out the hole by which it was suspended on the glass hook. Immerse the test piece in the corroding solution for a second 48-hr. period. Care should be taken that while drying and making the first weight, no oil or grease is allowed to get on the test piece by handling, since this would give low results. After the second 48-hr. immersion period, clean and weigh again as described above. From the loss during the last 48 hr., calculate the average rate of corrosion ($t = 48$ hr.) as indicated under Calculations.

Run a check test simultaneously. Never place test pieces of different metals in the same container. It is permissible to place two pieces of the same metal in the same container, but they should not be in contact.

In case the metal develops pitting, this factor must be included in the results, since failure in any case will occur when a pit has entirely penetrated the metal. Determine the magnitude of this effect by grinding down on a metallographic grinding set until all the pits have just disappeared and solid metal is reached. The loss in weight during this grinding is determined and the pitting calculated as indicated.

Form for Card File of Corrosion Data

Material
Corrosive agent
Concentration
Temperature
Rate of corrosion, mo.
Report of data
Method No.
By
For

Form for Laboratory Data

Metal
Solution
Temperature
Test piece No.
Dimensions
  Length
  Width
  Thickness
  Area
Weight
  Before test
  After test
  Loss
Time
  Start
  Finish
  Run, hr.
Corrosion rate

Calculations. Standard Static Corrosion Test.

If W = loss in weight (in grams) of test piece during the second 48-hr. immersion; A = area of test piece in square inches; S = density of metal in grams per cubic centimeter; and t = time of exposure in hours ($t = 48$):
Then C = rate of chemical corrosion expressed as inches penetrated per month:

$$C = \frac{24 \times 30 \times W}{(2.54)^3 ASt} \text{ or } \frac{43.9W}{ASt}$$

In order to calculate the pitting corrosion, let p = loss in weight (in grams) due to grinding out pits:

Then d = rate of penetration of metal by both normal corrosion over the entire surface and by local action due to pitting:

$$d = \frac{43.9(W + p)}{ASt}$$

Effect of Variables on Corrosion Test. As in other branches of engineering, it is necessary to apply a factor of safety to the results obtained, the factor varying with the degree of confidence in the applicability of the results. Ordinarily, a factor of from 3 to 10 might be considered normal.

Among the more important points which should be considered in attempting to base plant design of laboratory data are the following:

Effect of Oxygen. The test given above does not call for aëration, the reason being that the effect of oxygen is considered a specific one, to be taken into account for each problem. In general, its effect is to increase the rate of corrosion markedly, and, if it is to be present in the plant operation, its effect should be allowed for.

Electrolysis. This is a frequent source of trouble on the large scale. Not only is the use of different metals in the same piece of equipment dangerous, but the effect of cold working may be sufficient to establish potential differences of objectionable magnitude between different parts of the same piece of metal. **Riveting,** even when extreme precautions are taken to have the rivets of identical composition with the sheet, is very likely to establish potential differences. Even such slight working as threading without subsequent annealing has been known to cause rapid failure from electrolysis. The mass of metal in chemical apparatus is ordinarily so great, and the electrical resistance consequently so low, that a very small voltage can cause a very high current. It might also be noted that improper heat treatment of a solid-solution-type alloy may convert it from the solid-solution type to the non-homogeneous type, with the result that it fails even more rapidly than its poorest constituent. **Welding** also, if not properly done, may leave the weld of a different physical or chemical composition from the body of the sheet and cause the development of stray currents. A simple test for weld homogeneity is to treat a sample of the welded sheet containing a portion of both weld and sheet with a suitable solvent such as nitric acid or aqua regia, till about half the thickness of the metal has been dissolved; if heterogeneity exists, evidence of differential attack should be seen.

Velocity of Corrosive Liquid. Because of increased rate of removal of corrosion products and increased mechanical action, corrosion increases rapidly with increasing velocity of corrosive medium. This is especially true if the flow is turbulent, or if the liquid carries suspended matter. Frequently, slight alterations in design can be made to reduce this factor.

Local Concentration. Both local variations in temperature and crevices which permit the accumulation of corrosion products are capable of allowing the formation of concentration cells, with the result of accelerated local corrosion.

Temperature. In the laboratory, the temperature of the test specimen is that of the liquid in which it is immersed, and the measured temperature is actually that at which the reaction is taking place. In the plant, heat being supplied through the metal to the liquid in many cases, the temperature of the film of (corrosive) liquid on the inside of the vessel may be a number of degrees higher than that registered by the thermometer. As the relation between temperature and corrosion is a logarithmic one, the rate of increase is very rapid. Like other chemical reactions, the speed ordinarily increases two- to threefold for each 10° temperature rise, the actual relation being that of the equation: $\log K = A + \frac{B}{T}$, where K represents the rate of corrosion, and T absolute temperature.

Impurities. The effect of impurities in either structural material or corrosive material is so marked (while at the same time it may be either accelerating or decelerating) that for reliable results the actual materials it is proposed to use should be tested and not types of these materials. In other words, it is much more desirable to test the actual plant solution and the actual metal or non-metal than to rely upon a duplication of either. Since as little as 0.01 per cent of certain organic compounds will reduce the rate of solution of steel in sulfuric acid 99.5 per cent, and 0.05 per cent bismuth in lead will increase the rate of corrosion over 1000 per cent under certain conditions, it can be seen how difficult it would be to attempt to duplicate here all the significant constituents.

SPECIAL PROPERTIES OF CERTAIN MATERIALS

Metals. **Iron** withstands anhydrous acids and concentrated solutions of some acids but is attacked by dilute aqueous solutions. It resists alkaline solutions with the exception of very hot, highly concentrated solutions, though its resistance is sufficiently good in the latter case for extensive use.

Copper, as can be seen from its position in the electromotive series of the metals, has little tendency to dissolve in non-oxidizing acids. It is, however, very subject to oxidation and, as the oxide is readily soluble in most acids, is subject to marked attack in the presence of air or other oxidizing agents. Ammonia and amines similarly attack copper under oxidizing conditions. It is less attacked by *strong* caustic solutions than is iron.

Nickel possesses in many respects the characteristics of a seminoble metal. It is acted upon by dilute acids with relative slowness, so that it can frequently be used at a fairly low pH. It resists concentrated caustic solutions very well. It is less affected by ammonia and by the amines than copper and is less rapidly oxidized.

Lead, because of its low tensile strength and elastic limit, is used chiefly in the form of linings. If not carefully anchored in place it is very liable to creep slowly, with the resulting formation of buckles especially if alternately heated and cooled. It is attacked but slowly by most acids and hence is very extensively used in chemical-plant construction. Homogeneous lead lining overcomes the tendency of lead to creep and also improves the heat transfer as compared with loose linings.

Aluminum, while belonging to the class of metals which decompose water, is quite resistant to dilute and weak acids because of the presence of a surface film of the relatively inert hydrated oxide. Any material capable of removing the aluminum oxide, however, such as the halogen acids or alkalies, will cause very rapid corrosion of aluminum by exposing the highly reactive metal. Thermometers in aluminum equipment should be carefully protected, as the mercury from a broken bulb, if it comes in intimate contact with the metal, will cause rapid perforation.

Tantalum, while not extensively used in the past, possesses the valuable property of being unattacked by aqueous hydrochloric acid, either hot or cold, and by most other chemical reagents excepting hydrofluoric acid, fluorides, and sodium hydroxide solutions. Its relatively high cost restricts its use to linings for the most part but makes possible the construction of certain equipment in metal which would otherwise have to be built of a non-metal.

Silver also possesses a fair degree of resistance to the aqueous solutions of the halogen acids. In this case the resistance is due largely to the formation of a protective film of the insoluble halide, so that conditions which favor the recrystallization of the halide, such as the presence of ammonium salts or very high temperatures, materially reduce the life of the equipment. Silver is also widely used for handling and storage of organic acids, particularly at high temperatures.

Magnesium is one of the few materials not attacked by dilute aqueous solutions of hydrofluoric acid.

Antimony, while seldom used because of brittleness, also withstands well the action of aqueous hydrochloric acid.

Alloys. Chromium-iron-nickel alloys (stainless steels) belong to the solid-solution type of alloy when properly heat treated; when improperly heat treated, segregation takes place and the resulting electrolysis may reduce the resistance below that of ordinary steel. Roughly, the resistance to acids increases with increasing nickel content.

Bronzes tend to possess the acid resistance of copper with a lessened tendency to oxidation, so that their over-all resistance is somewhat greater than that of copper while the mechanical properties are much superior to those of copper.

Brasses, like bronzes, have mechanical properties and workability superior to those of copper. As a class they are subject to dezincification and consequent weakening on exposure to corrosive conditions.

Silicon alloys, such as **Corrosiron, Duriron,** and **Durichlor,** have the high resistance to be expected from the high silicon content and also the brittleness and hardness which are ordinarily associated with the compounds of silicon. The high-silicon alloys must be worked by casting and grinding.

Non-metals. One important difference between the metals and the non-metals, which greatly affects the design of equipment, is the low thermal conductivity of the latter class. It is frequently difficult or impossible to obtain the necessary heat transfer through the wall of the vessel when working with a non-metal, so that some form of internal heating or cooling must be resorted to, such as coils, electrical heating, or live steam.

Glass possesses the desirable quality of almost perfect resistance to acids of all strengths and at almost all temperatures, hydrofluoric acid being an exception to the rule. The difficulty of obtaining large sizes and the fragility of the material have somewhat limited its application, piping being the chief form of equipment ordinarily used. The making of tight joints also presents a problem, no material equally resistant being available for use as a gasket material, while fused joints increase the complication and fragility of the assembly. Glass is also sensitive to sudden changes of temperature and must be carefully protected from them.

Silica ware, while possessing the corrosion resistance of glass in a more marked degree, also possesses all the weaknesses of glass with the exception of its susceptibility to sudden temperature changes.

Enameled ware to a considerable degree avoids the objectionable mechanical features of glassware but at the expense of a reduction in chemical resistance. The glass used for coating iron or steel is less resistant to chemical action than the best grade of chemical glass, which because of its coefficient of expansion and melting point, however, cannot be used for enameling.

Stoneware largely avoids the mechanical fragility of glass and silica but substitutes a problem in heat transfer, the thickness of the material ordinarily used causing this to become a serious problem when considerable quantities of heat are to be supplied or removed.

Other ceramic materials present problems, in general, similar to those already mentioned, the design problems accompanying the use of porcelain or stoneware, tile or brick, of cementlike material in general, being accompanied by limitations as to temperature, pressure, heat transfer, size of equipment, etc., which are not present when working with the metals or alloys. Enameled equipment is the only type of this group in which the working surface is non-metallic, and the physical strength is supplied by a metal. In tile lining, difficulty is also encountered in obtaining satisfactory cement of resistance equal to the tile.

Hard and soft rubber are frequently used for the handling of acids, especially dilute aqueous solutions. The resistant properties of the two are similar, both being unattacked by dilute aqueous solutions except those of oxidizing agents and both being swelled by organic solvents. An important difference between the two lies in the fact that the soft rubber is used as a lining, usually for steel, while the hard rubber can be used alone.

Synthetic rubberlike polymers are becoming of increasing importance as engineering raw materials because of the superiority to natural rubber in many important properties such as resistance to oxidation, solvents, oil, and many chemicals.

Polymers such as neoprene and Perbunan can be compounded and cured to yield vulcanizates similar to natural rubber in their stress-strain and elastic properties. Other polymers such as Thiokol are swollen even less than neoprene or Perbunan by aliphatic hydrocarbons but have lower elongations and do not retain their properties as well at elevated temperatures. Plasticized polyvinyl chloride such as Koroseal can be used where less rubberlike properties are required.

Wood, while fairly inert chemically, is readily dehydrated by concentrated solutions and hence shrinks badly when subjected to the action of such solutions. It is also slowly hydrolyzed by acids and alkalies, especially when hot. In tank construction, if sufficient shrinkage once takes place to allow crystals to form between the staves, it becomes very difficult to make the tank tight again.

Structural carbon is available for use as tank linings, tower packing, absorption systems, and piping. It is suitable for use with most chemical materials except strong oxidizing agents. An impervious form, Karbate, is available; Karbate No. 2, impervious graphite, has good thermal conductivity and can be used in heat-transfer equipment.

RECOMMENDED MATERIALS OF CONSTRUCTION FOR DIFFERENT USES

Materials of Construction Used in Chemical Operations. (From "Corrosion, Causes and Prevention," pages 562 to 568, by Frank N. Speller.) This list gives the materials of construction commonly used in chemical operations. It represents the results of a survey of the literature, the edited

RECOMMENDED MATERIALS OF CONSTRUCTION FOR DIFFERENT USES—(*Continued*)

comments of manufacturers of chemicals and apparatus, and the experience of Whitman and Russell who compiled the list.[1] The table does not attempt to give the entire range over which a given material will resist corrosion or the entire list of materials which will withstand a given corroding agent, but rather it indicates the range over which the material usually finds application. The materials included are given in alphabetical order with no attempt to indicate which ones are superior. Where pipe or tubing is specified, valves, fittings, etc., should be constructed of the same material. Where monel metal is specified, other copper-nickel alloys are often suitable.

The concentrations and temperatures commonly encountered are given in parentheses or brackets, the concentrations being given first and the temperatures second. For example "[all concs.; temp. 158°F. (70°C.)]" means all concentrations at temperatures below 158°F. (70°C.).

1. **Acetic anhydride.**

Copper, enameled iron, fused silica, glass, high-silicon iron, steel, lead.

2. **Acid, acetic.**

Aluminum—distillation (glacial), condenser and piping, tanks (glacial).
Aluminum bronze—pumps (all concs.), condenser and piping.
Cast iron—stills for commercial acid.
Chromium-nickel stainless steels (all concs.; all temps.).
Copper—distillation (glacial), condenser and piping, tanks (glacial).
Enameled iron or steel—condensers and piping, tanks (glacial).
Hard rubber—tank lining, pumps, piping (all concs.; low temp.).
High-silicon iron—condensers and piping, pumps (all concs.).
Monel metal—pumps and tubing (all concs.).
Phosphor bronze—pumps (high concs.).
Silver—condensing tubes.
Slate—tanks (glacial).
Stoneware—condensers and piping; tanks and pumps (all concs.).
Wood-tanks (oak, cypress, fir) (glacial); for short-storage tanks (low concs.).

3. **Acid, benzoic** [vapor 482°F. (250°C.)].

Aluminum, nickel.
KA2SMo—high temperatures.
Silver.

4. **Acid, chloracetic** (plant concs.).

Copper, enameled iron, lead, nickel.

5. **Acids, fatty** (propionic, butyric, etc.).

Aluminum—condensers, pumps.
Bronze—pumps (low concs.).
Cast iron—stills.
Copper—pipe (all concs.; all temps.), condenser tubes.
Enameled iron—stills, condensers.
High-silicon iron—condensers, piping, pumps.
Monel metal—pumps, tubing, receiving eggs.
Steel—stills.
Stoneware—pumps.

---

[1] The helpful cooperation of the various manufacturers of chemicals and equipment who contributed to this table is acknowledged. The table of Calcott and Whetzel has been of especial assistance in this work.

RECOMMENDED MATERIALS OF CONSTRUCTION FOR DIFFERENT USES—*(Continued)*

6. Acid, hydrochloric.

Acid-brick—lining for tanks (all concs.; all temps.).
Cast iron—furnace pots.
Durichlor—(all concs.; all temps.).
Earthenware—gas fans (moist), gas pipe (moist, cool).
Fused silica—gas pipe (all concs.; all temps.), absorption systems.
Glass—absorption systems, pipes, carboys.
Hard lead—pumps (concs. from 1 to 5 per cent).
Hard rubber—lining for fans (moist, cool), pumps (concs. from 1 to 30 per cent), pipes (all concs.; all temps.), lining for tanks (all concs.; low temps.).
Hastelloy B—(all concs.; all temps.).
Haveg—(all concs. up to boiling point).
Karbate—pipe, linings, absorption systems (all concs.; all temps.).
Monel metal—pickling equipment (low concs. and temps.).
Pure gum rubber—tubing, lining for tanks, tank cars, fans (all concs.; low temps.).
Rubber—acid hose for piping (all concs.).
Stoneware—gas and liquor pipes (all concs.; all low temps.), absorption systems, tanks (above 5 per cent), pumps, lifts, fans.
Tantalum—aqueous, hot or cold.

7. Acid, lactic.

Bronze—pumps (all concs.).
Copper—evaporators, tanks, and pipe (all concs.; all temps.).
Enameled steel—evaporating pans, tanks, pipes, pumps (for edible products) (all concs.).

8. Acid, mixed.

Acid-brick—tanks.
Cast steel—tanks, pumps.
Enameled iron—nitrators.
High-chrome alloys—dipping baskets.
18-8—dipping baskets.
Lead—nitrators.
Steel—tanks, pipe, nitrators.

9. Acid, nitric.

Acid-brick—towers and packing (conc. 60 per cent; all temps.).
Fused silica—gas piping and condensers (all concs.).
High-chromium iron.
High-silicon iron—gas piping, condensers, pumps, fans, and liquor piping (all concs.).
Stainless steel—chromium-nickel-iron alloys.
Stoneware—gas piping, condensers, absorption systems, pumps, fans, and piping (all concs.).

10. Acid, sulfuric.

Acid-brick—absorption towers (concs. < 75 per cent; high temps.), absorption-tower packing, pickling tanks (6 per cent conc.).
Aluminum bronze—pickling apparatus (6 per cent conc.), pumps (low conc.).
Cast iron—absorbers and tanks (concs. from 85 to 98 per cent), pumps and lifts (concs. from 90 to 98 per cent).
Coke—absorption-tower packing.
Concrete—pickling tanks.
Hard rubber—pumps (concs. from 1° to 45°Bé.; low temps.), pipe, tanks.
High-silicon iron—chamber acid concentration (concs. from 70 to 90 per cent), fans, pumps (concs. from 10 to 70 per cent), piping.

RECOMMENDED MATERIALS OF CONSTRUCTION FOR DIFFERENT USES—*(Continued)*

Lead [temps. < 320°F. (150°C.)]—chambers, shell for absorption towers (only soft lead for 48° to 50°Bé.), lining for pumps (concs. < 70 per cent), lining for pickling tanks and agitators, hard lead for fans and pumps (low concs.).
Monel metal—pumps and tubing [concs. < 15 per cent; temps. < 392°F. (200°C.)], pickling apparatus (6 per cent conc.).
Quartz—absorption tower packing.
Stainless steel (KA2SMo)—0 to 10 per cent and 95 to 122.5 per cent.
Steel—absorbers and tanks (concs. from 90 per cent to fuming), pumps and lifts [90 per cent conc.; temps. < 149°F. (65°C.)].
Stoneware—absorption-tower packing, fans, pumps (all concs.), pickling apparatus and pickling tanks (6 per cent conc.).
Wood—pickling apparatus (6 per cent conc.), tanks (concs. < 20 per cent).

11. Acid, sulfurous.

Acid-brick—linings of digesters and absorption towers.
Aluminum bronze—relief gas lines [temps. < 320°F. (150°C.)], pumps [high conc.; temp. 212°F. (100°C.)], piping liquor [all concs.; temps. < 212°F. (100°C.)], SO_2 gas piping [temps. < 392°F. (200°C.)].
Cast iron—SO_2 burner gas [temps. < 392°F. (200°C.)].
Enameled iron—tanks [low concs.; temps. < 212°F. (100°C.)].
Hard lead—liquor piping [all concs.; temps. < 212°F. (100°C.)], SO_2 gas piping [temps. < 392°F. (200°C.)].
Lead—pumps [high concs.; temp. < 212°F. (100°C.)], lining for tanks [low concs.; temps. < 212°F. (100°C.)], pipe and pipe lining for liquor [(temp. < 212°F. (100°C.)], SO_2 gas piping [temps. < 392°F. (200°C.)].
Stainless steel (KA2S)—digesters.
Steel—shell for absorption system [temps. < 212°F. (100°C.)], shell for digesters (all concs.; all temps.), rotary burner shells.
Stoneware—absorption systems [all concs.; temps. < 212°F. (100°C.)], pumps [high concs.; temps. < 212°F. (100°C.)], piping for liquor [all concs.; temps. < 122°F. (50°C.)], SO_2 gas piping [temps. < 392°F. (200°C.)].
Wood—asphalted for tanks [low concs.; temps. < 212°F. (100°C.)] (yellow pine or cypress).

12. Alkaline brines.

Cast iron, low-carbon open-hearth steel, ingot iron, steel, wood.

13. Alums.

High-silicon iron—pumps (all concs.; all temps.).
Lead—pumps and lining for tanks (concs. < 20°Bé.).
Stoneware—tanks (concs. < 20°Bé.), pumps (all concs.; all temps.).
Wood—tanks (concs. < 20°Bé.).

14. Ammonia, anhydrous—liquid.

Monel metal—pump and valve parts.
Steel.

15. Ammonia, aqua.

Cast iron and steel (all concs.; all temps.).

16. Ammonia, vapor.

Aluminum (all concs.; high temps.).
Nickel—high temps.
Steel—ordinary temps.

RECOMMENDED MATERIALS OF CONSTRUCTION FOR DIFFERENT USES—(*Continued*)

17. Ammonium nitrate.

Acid-brick—lining for reaction tubes.
Cast iron—evaporating pans, coils.
High-silicon iron—NH_3 distributor in reaction tubs.
Stainless steel.
Steel—heater coils.
Wood—mixing tank.

18. Ammonium sulfate.

Acid-brick—tank linings (high temps.).
Aluminum bronze—ammonia saturators.
High-silicon iron—pipe and pumps (all concs.).
Lead—tanks and absorber linings, pumps, pipe.
Monel metal—filter cloth, ammonia saturators, extractor baskets.

19. Antimony trichloride.

Enameled iron, lead-lined steel, steel.

20. Bleach liquors.

Cast iron—piping, pumps.
Cement—tanks, bleaching tanks.
High-silicon iron—pumps, pipes.
Lead—pipe lining.
Monel metal—valves, tank linings, auxiliaries (concs. < 3 g. available chlorine per liter).

21. Calcium chloride.

Cast iron—evaporators, piping.
Steel—evaporators, piping, tanks.

22. Caustic alkalies.

Cast iron (low Si and 2.5 Mn best)—pots for fusions [concs. from 50 to 95 per cent; temps. < 482°F. (250°C.)], kettles and autoclaves (all concs.; all temps.), pumps and piping [all concs.; temp. < 176°F. (80°C.)], valves (all concs.; all temps.) (lubricated with graphite), salt catches.
Cast steel—kettles and autoclaves.
Copper—filtrations.
Monel metal—filter cloth, pumps, evaporator tubes, autoclave linings, salt catches.
Nickel—evaporator tubes.
Steel—evaporators, pumps, piping [all concs.; temps. < 212°F. (100°C.)], tanks (all concs.; all temps.).
Steel, low-carbon open hearth (seamless)—evaporator tubes.

23. Chlorine (gas).

Cast iron—piping [dry gas; temp. of about 212°F. (100°C.)], fans (moist gas; 1 per cent conc.; low temps.), chlorinators.
Earthenware—sewer-pipe piping [moist gas; temps. slightly $>$ 212°F. (100°C.)].
Enameled iron—chlorinators.
Everdur—valve bodies.
Fused silica—piping (any temperatures).
Glass—chlorinators.
Hard rubber—up to 10 lb. per sq. in. pressure (low temps.).
High-silicon iron—piping [dry gas; temp. 212°F. (100°C.)], fans.
Lead—lining for pipe [moist gas; temps. < 212°F. (100°C.)], chlorinators.

RECOMMENDED MATERIALS OF CONSTRUCTION FOR DIFFERENT USES—(*Continued*)

Monel metal—valves (dry and liquefied gas).
Monel—valve-stem seats.
Nickel—piping, high temp., 300 to 800°F.
Silver—reducing valves.
Steel—piping [dry gas; temp. to 300°F.)], chlorinators, high-pressure cylinders.
Stellite—valves.
Stoneware—piping (dry gas; low temps.) (flanged, with rubber gaskets).
Stainless steel—piping, high temp., 300 to 600°F.

24. **Chlorine (solutions).**

Enameled iron—pumps and piping.
Hard rubber—pumps and piping, valves (low temps.).
High-silicon iron—pipe, pumps.
Stellite—valves.
Stoneware—pumps, piping, valves, tanks, jars, etc.

25. **Electrolytic cells.**

Asbestos—diaphragm.
Concrete—bodies.
Filtros—diaphragm.
Steel—bodies.

26. **Filtrations (strong acids).**

Asbestos, Filtros, porous stoneware, sand, wool, porous carbon.
Vinyon—filter cloth (temp. below 130°F.).

27. **Filtrations (weak acids).**

Cotton, Filtros, ground quartz, monel-metal cloth, porous stoneware, wool, nitrocellulose.
Human hair, glass cloth, porous carbon, porous rubber, Vinyon (below 130°F.).

28. **Fruit juices.**

Aluminum—kettles, evaporators.
Enameled iron—evaporators, tanks, pipe.
Glass—piping (all concs.).
Hard rubber—pumps, piping (low temps.).
Monel metal—evaporators, tanks, pipes.
Silvered copper—evaporators, tanks, pipes.
Tinned copper—evaporators, tanks, pipes.
Wood—tanks.

29. **Phenol.**

Enameled iron—tanks, receivers.
Silver—still lining, condensers.

30. **Reductions.**

Cast iron, high-silicon irons, lead, monel metal.

31. **Soda pulp, black liquor.**

Cast iron—evaporators, pumps (all concs.; all temps.).
Monel metal—evaporator tubes.
Steel—tanks, evaporators (all concs.; all temps.).
Wrought iron—evaporators.

RECOMMENDED MATERIALS OF CONSTRUCTION FOR DIFFERENT USES—(*Continued*)

32. Sodium carbonate.

Cast iron, steel; wood (low concs.).

33. Sodium chloride.

Admiralty brass—sea-water condenser tubes.
Bronze—pumps.
Cast iron—pumps, piping (all concs.; all temps.).
Cement—tanks.
Copper—tubes for heaters, evaporators.
Enameled iron—tanks, piping.
Monel metal—pumps, tubing, filter cloth.
Naval bronze—pumps, piping.
Steel—piping.
Wood—tanks.
Wrought iron—piping.

34. Sodium hydrosulfite.

Lead—absorber and tank lining, also piping [20 per cent conc.; temp. 77°F. (25°C.)].

35. Sodium hypochlorite.

Concrete, soapstone, stoneware, wood [20 per cent conc.; temp. 77°F. (25°C.)].
See also Bleach liquors.

36. Sodium nitrate.

Cast iron, steel (all concs.; all temps.).

37. Sodium sulfate.

Cast iron—pumps (all concs.; all temps.).
Galvanized iron—crystallizing pans.
Steel—tanks (all concs.; all temps.).
Wood—filter presses, crystallizing pans [temps. $< 122°$F. (50°C.)].

38. Sodium sulfide.

Acid-brick—furnaces.
Inconel—solutions.
Lead—tank lining, piping [concs. 5 to 25 per cent; temps. $< 212°$F. (100°C.)].
Nickel—solutions.
Stainless steel—solutions.
Steel—tanks, piping, cooling pans (high concs.).
Stoneware—tanks, piping.

39. Sodium thiosulfate.

Hard lead, lead lining, monel metal, nickel, stoneware.

40. Sulfite pulp liquor.

Cast iron—evaporators, pumps.
Special alloys—evaporators.
Special alloys of chromium and nickel with copper and molybdenum—pumps, valves, evaporators.
Wood—tanks.

41. Sulfonations.

Brick-lined steel—sulfonators.

RECOMMENDED MATERIALS OF CONSTRUCTION FOR DIFFERENT USES—(*Continued*)

Cast iron, enameled iron, lead.
Steel—sulfonators.

42. Sulfur chloride.

Earthenware, glass, lead, steel.

43. Sulfuryl chloride.

Glass, lead.

44. Zinc chloride.

Cast iron—fusion kettles, hot solutions.
Enameled iron—concentration, fusion.
Hastelloy B—solutions, hot, concentrated.
Rubber-lined steel—storage or shipping (low conc., low temp.).
Steel—tanks (hot solutions if free from copper).
Stoneware—tanks, pipes.

Table 1. Concrete for Chemical-tank Construction*

Solutions	Applications	Treatment
Ammonium hydroxide	Storing strong ammoniacal liquors	Inside coating of tar
Bleach liquors	Bleach tanks, stuff chests and bleachers in paper plants	No special treatment necessary
Brines	Tanks for holding brine solutions in paper manufacturing, salt works, and vegetable-pickling plants	No special treatment necessary
Calcium chloride	Containers	No special treatment necessary
Coconut oil	Storage tanks	No special treatment necessary
Cottonseed oil	Storage tanks	No special treatment necessary
Fish oil	Storage tanks	No special treatment necessary
Fluosilicates	Depositing tanks in electrolytic refining of lead	Asphalt coating used
Glycerine	Storage tanks	Coating of rich mortar
Hemlock liquors	Storage tanks	Treated on inside with a brush coat of neat cement
Hydrochloric acid	Chemical-sediment tanks (dilute acid)	Lined inside and outside with special coating
Leaching bark	Storing and handling of leaching-bark solutions	Inner surface plastered with cement mortar
Mineral oils	Storage tanks	Coating of rich mortar
Molasses	Storage tanks	No special treatment necessary
Peanut oil	Storage tanks	No special treatment necessary
Quebracho extract	Handling of quebracho extracts	Inside plaster coat of 1:2 cement mortar
Sodium silicate	Storage tanks	
Soya-bean oil	Storage tanks	No special treatment necessary
Sulfuric acid	Digester tanks for boiling wood chips	Lined with brick laid in litharge
	Depositing tank in electrolytic refining of zinc	Asphalt coating used
	Storing of fuming acid	Lined with lead
	Storing 5% sulfuric acid	Lined with mastic
Tanning liquors	Vats for beam-house work in a tannery	No special treatment necessary
Water	Storage tanks	Coating of rich mortar
Zinc chloride	Storing concentrated solutions of zinc chloride	

* From *Chem. & Met. Eng.*, September, 1929.

Table 2. Resistance of Metals and Alloys to Corrosive Gases*

Part I. Typical Composition of Alloys for Resistance to Corrosive Gases (in Presence of Moisture)

Composition	Representative Trade Names or Type Number (See Tables 3 and 5)
Moist SO_2[a]	
14–18 Cr, bal. Fe, 0.12 max. C	Stainless type 430[f]
23–30 Cr, bal. Fe, 0.35 max. C	Stainless type 446
17.5–19 Cr, 8–9 Ni, bal. Fe, 0.08 max. C	Stainless type 304[g]
18–20 Cr, 9–10 Ni, bal. Fe, 0.08 max. C	Stainless type 304
22–26 Cr, 12–14 Ni, bal. Fe, 0.20 max. C	Stainless type 309
17–19 Cr, 7–9 Ni, 1–1.5 Cu, 1–1.5 Mo, bal. Fe, 0.15 max. C	Stainless type 315
18–20 Cr, 14 max. Ni, 3–4 Mo, bal. Fe, 0.10 max. C	Stainless type 317
19 Cr, 22 Ni, 3.5 Mo, 1.25 Si, 0.07 max. C, bal. Fe	Durimet
14.5 Si, 0.8 C, bal. Fe	Duriron
14 Cr, 58 Ni, 17 Mo, 5 W, 6 Fe	Hastelloy C
99.93 Pb, 0.06 Cu	Chemical lead
94 Pb, 6 Sb	Antimonial lead
100 Ta	Tantalum
Moist Nitrous Gases[b]	
14–18 Cr, bal. Fe, 0.12 max. C	Stainless type 430[f]
23–30 Cr, bal. Fe, 0.35 max. C	Stainless type 446
17.5–19 Cr, 8–9 Ni, bal. Fe, 0.08 max. C	Stainless type 304[g]
18–20 Cr, 9–10 Ni, bal. Fe, 0.08 max. C	Stainless type 304
22–26 Cr, 12–14 Ni, bal. Fe, 0.20 max. C	Stainless type 309
17–19 Cr, 7–9.5 Ni, 1–1.5 Cu, 1–1.5 Mo, bal. Fe, 0.15 max. C	Stainless type 315
18–20 Cr, 14 max. Ni, 3–4 Mo, bal. Fe, 0.10 max. C	Stainless type 317
14 Cr, 58 Ni, 17 Mo, 5 W, 6 Fe	Hastelloy C
55–60 Ni, 18–24 Cr, 5–7 Mo, 4–8 Fe, 1–8 Cu	Illium G
14.5 Si, 0.8 C, bal. Fe	Duriron
100 Ta	Tantalum
Moist Hydrochloric Acid Vapors[c]	
62 Ni, 30 Mo, 5 Fe	Hastelloy B
14 Cr, 58 Ni, 17 Mo, 5 W, 6 Fe	Hastelloy C[h]
14.5 Si, 3 Mo, bal. Fe	Durichlor
100 Ta	Tantalum
Moist Chlorine[d]	
14 Cr, 58 Ni, 17 Mo, 5 W, 6 Fe	Hastelloy C[h]
100 Ta	Tantalum
Moist Hydrofluoric Acid Vapors, Cold[e]	
55–60 Ni, 18–24 Cr, 5–7 Mo, 4–8 Fe, 1–8 Cu	Illium G
99.93 Pb, 0.06 Cu	Chemical lead
86 Cu, 10 Al, 4 Fe	Ampco

* Based on Chilton and Huey [*Ind. Eng. Chem.*, **24**, 125 (1932)]; "Tables of Chemical Compositions, Physical and Mechanical Properties and Corrosion-resistant Properties of Corrosion-resistant and Heat-resistant Alloys" [*Proc. Am. Soc. Testing Materials*, **30**, Part I, Suppl. (1930)]; "Modern Alloys" [*Chem. & Met. Eng.*, **39**, 497 (1932)]; and other sources.

[a] Alloys from column headed Moist Sulfurous Atmosphere, A.S.T.M. Tables, 1930.

[b] Alloys taken from column headed Nitric Acid, recommendations for all concentrations at room and some higher temperature.

[c] Alloys taken from column headed Hydrochloric Acid, A.S.T.M. Tables, 1930; same recommendations.

[d] Alloys taken from column headed Chlorine in Aqueous Solution; same recommendations.

[e] Alloys taken from column headed Hydrofluoric Acid, A.S.T.M. Tables, 1930, and are the only ones unqualifiedly recommended by the manufacturers; none are recommended for hot solutions.

[f] Nearest type number to compositions listed, A.S.T.M. Tables, 1930.

[g] Columbium-modified compositions also applicable.

[h] Low temperatures only.

Table 2. Resistance of Metals and Alloys to Corrosive Gases*—(*Continued*)

Part II. Typical Composition of Alloys for Resistance to Direct Gaseous Attack (in Absence of Moisture)

Attack by Air and Oxidizing Fuel Gases[a]

Max. temp. recommended, °C.	Composition	Representative trade names or type number (see Tables 3 and 5)
500	70 Cu, 29 Ni, 1 Sn	Adnic
500[b,c]	67 Ni, 30 Cu, 1.4 Fe, 1 Mn	Monel
540	14 Cr, 0.35 C, bal. Fe	Stainless type 420[d]
595	13 Cr, 2 Ni, 0.12 C, bal. Fe	Stainless type 414
700	72 Ni, 18 Co, 6.5 Fe, 2.5 Ti, 0.5 Al	Konel
700[b,c]	99.4 Ni	Nickel
700[b,c]	79.5 Ni, 13 Cr, 6.5 Fe	Inconel
760[b]	18–20 Cr, 9–10 Ni, 0.2 C, bal. Fe	Stainless Type 302
800[b]	58 Ni, 20 Cr, 20 Fe, 2 Mn	Hastelloy A
800[b]	85 Ni, 10 Si, 3 Cu, 2 Al	Hastelloy D
815	14–18 Cr, bal. Fe	Stainless type 440
815[b,c]	22–26 Cr, 12–14 Ni	Stainless type 309
815[b,c]	36 Ni, 11–15 Cr, bal. Fe	ATV-1
925[b]	23–30 Cr, 0.35 max. C, bal. Fe	Stainless type 446
950[b,c]	26.5 Ni, 14 Cr, 3.5 W, bal. Fe	ATV-3
1000	50 Co, 30 Cr, 15.5 W, bal. Fe	Stellite No. 1
1000	65 Co, 30 Cr, 4 W, bal. Fe	Stellite No. 6
1000	60 Co, 30 Cr, 8 W, bal. Fe	Stellite No. 12
1000[b,c]	58 Ni, 17 Mo, 14 Cr, 5 W, 6 Fe	Hastelloy C
1150[c]	80 Ni, 20 Cr	Nichrome V
	Attack by Sulfur Gases[e]	
540	18–20 Cr, 9–10 Ni, 0.2 C, bal. Fe	Stainless type 302
540	14 Cr, 0.35 C, bal. Fe	Stainless type 420[d]
815	14–18 Cr, bal. Fe	Stainless type 440
925	23–30 Cr, 0.35 max. C, bal. Fe	Stainless type 446
	Attack by Hydrogen, Nitrogen and Ammonia[f]	
480	18–20 Cr, 9–10 Ni, 0.2 C, bal. Fe	Stainless type 302
540	22–26 Cr, 12–14 Ni	Stainless type 309
705	18–20 Cr, 14 max. Ni, 3–4 Mo, bal. Fe, 0.10 max. C	Stainless type 317
785	20 Ni, 8 Cr, 1 Si, 0.4 C, bal. Fe	Cyclops No. 17
900	35 Ni, 18 Cr, bal. Fe	Chromax
1000	45 Ni, 55 Cu	Advance

* Based on Chilton and Huey [*Ind. Eng. Chem.*, **24**, 125 (1932); "Tables of Chemical Compositions, Physical and Mechanical Properties and Corrosion-resistant Properties of Corrosion-resistant and Heat-resistant Alloys," *Proc. Am. Soc. Testing Materials*, **30**, Part I, Suppl. (1930)]; "Modern Alloys," *Chem. & Met. Eng.*, **39**, 497 (1932); and other sources.

[a] Alloys are listed under the lowest maximum operating temperature recommended by any manufacturer for material of substantially the same composition (A.S.T.M. Tables, 1930). Manufacturers vary widely in their recommendations; some recommend alloys of the same or even lower alloy content for higher temperatures.

[b] Recommended for oxidizing fuel gases at the same temperature (A.S.T.M. Tables, 1930).

[c] Nickel alloys recommended only in absence of sulfur-containing gases.

[d] Nearest type number to compositions listed, A.S.T.M. Tables, 1930.

[e] Only those alloys recommended for H_2S, SO_2, and SO_3 are included; the temperature is the lowest recommended by any manufacturer for substantially the same composition (A.S.T.M. Tables, 1930).

[f] Taken from column with this heading in A.S.T.M. Tables, 1930; the temperatures given are the lowest recommended by any manufacturer for any of the three gases for substantially the same alloy composition. Note that the metals should not be used at high pressure at these temperatures; see Maxwell [*Trans. Am. Soc. Metals*, **24**, 213 (1936)].

Table 3. Materials of Construction—Base Metals. Manufacturers, Chemical Composition, Physical Properties, and Commercial Forms Available

No.	Material	Manufacturer	Essential nominal chemical composition,* per cent	Specific gravity	Mean coeff. therm. exp., 32–212°F. (multiply by 10^{-5})	Therm. conduct., C.G.S. units, room temp.	Forms available†
			Ferrous Alloys				
1	Abrasion Resisting	Lukens Steel Co., Coatesville, Pa.	Fe; C, 0.4–0.5; Mn, 1.80 max.	....	....	....	HR, P
2	Allegheny Metal 18-8	Allegheny Ludlum Steel Corp., Pittsburgh	Fe; Cr, 18–20; Ni, 8–10; C, 0.16 max.	7.9	0.95	0.04	C
3	Allegheny Metal 18-8 S	Allegheny Ludlum Steel Corp., Pittsburgh	Fe; Cr, 18–20; Ni, 8–10; C, 0.12 max.	7.9	0.95	0.04	C
4	Allegheny Metal 18-8 S	Allegheny Ludlum Steel Corp., Pittsburgh	Fe; Cr, 18–20; Ni, 8–10; C, 0.08 max.	7.9	0.95	0.04	C
5	Allegheny Metal 20-10 S	Allegheny Ludlum Steel Corp., Pittsburgh	Fe; Cr, 19–22; Ni, 10–12; C, 0.10 max.	7.9	0.95	0.04	C
6	Allegheny Metal 25-12	Allegheny Ludlum Steel Corp., Pittsburgh	Fe; Cr, 22–26; Ni, 12–14; C, 0.18 max.	7.9	0.9	0.04	C
7	Allegheny Metal 25-12 S	Allegheny Ludlum Steel Corp., Pittsburgh	Fe; Cr, 22–26; Ni, 12–14; C, 0.11 max.	7.9	0.9	0.04	C
8	Allegheny Metal 25-20	Allegheny Ludlum Steel Corp., Pittsburgh	Fe; Cr, 24–26; Ni, 19–21; C, 0.18 max.	7.9	0.9	0.04	C
9	Allegheny Metal 25-20 S	Allegheny Ludlum Steel Corp., Pittsburgh	Fe; Cr, 24–26; Ni, 19–21; C, 0.11 max.	7.9	0.9	0.04	C
10	Allegheny Metal 20-25 S	Allegheny Ludlum Steel Corp., Pittsburgh	Fe; Cr, 19–21; Ni, 24–26; C, 0.11 max.	7.9	0.9	0.04	C
11	Allegheny Metal 20-25 SM	Allegheny Ludlum Steel Corp., Pittsburgh	Fe; Cr, 19–21; Ni, 24–26; C, 0.11 max.; Mo, 2–3	7.9	0.9	0.04	C
12	Allegheny Metal 29-9	Allegheny Ludlum Steel Corp., Pittsburgh	Fe; Cr, 27–31; Ni, 8–10; C, 0.20 max.	7.9	0.9	0.04	C
13	Allegheny Metal 29-9 S	Allegheny Ludlum Steel Corp., Pittsburgh	Fe; Cr, 27–31; Ni, 8–10; C, 0.12 max.	7.9	0.9	0.04	C
14	Allegheny Metal 18-8 M	Allegheny Ludlum Steel Corp., Pittsburgh	Fe; Cr, 16–18; Ni, 14 max.; C, 0.16 max.; Mo, 2–3	7.9	0.95	0.04	C

15	Allegheny Metal 18-8 SM	Allegheny Ludlum Steel Corp., Pittsburgh	Fe; Cr, 16–18; Ni, 14 max.; C, 0.12 max.; Mo, 2–3	7.9	0.95	0.04	C
16	Allegheny Metal 19-10 M	Allegheny Ludlum Steel Corp., Pittsburgh	Fe; Cr, 18–20; Ni, 14 max.; C, 0.16 max.; Mo, 3–4	7.9	0.95	0.04	C
17	Allegheny Metal 19-10 SM	Allegheny Ludlum Steel Corp., Pittsburgh	Fe; Cr, 18–20; Ni, 14 max.; C, 0.12 max.; Mo, 3–4	7.9	0.95	0.04	C
18	Allegheny Metal 28-4	Allegheny Ludlum Steel Corp., Pittsburgh	Fe; Cr, 25–30; Ni, 3–5; C, 0.25 max.; N, 0.15	7.6	0.60	0.06	C
19	Allegheny Metal 15-35	Allegheny Ludlum Steel Corp., Pittsburgh	Fe; Cr, 14–16; Ni, 33–36; C, 0.25 max.	7.9	0.8	0.04	C
20	Allegheny Metal 18-8 C	Allegheny Ludlum Steel Corp., Pittsburgh	Fe; Cr, 17–20; Ni, 8–12; C, 0.10 max.; Cb, 8 × C	7.9	0.95	0.04	C
21	Allegheny Metal 18-11 C	Allegheny Ludlum Steel Corp., Pittsburgh	Fe; Cr, 17–18.5; Ni, 10 min.; C, 0.08 max.; Cb, 8 × C	7.9	0.95	0.04	C
22	Allegheny Metal 12 Cr.	Allegheny Ludlum Steel Corp., Pittsburgh	Fe; Cr, 10–13.5; Ni, 1 max.; C, 0.15	7.7	0.55	0.06	C
23	Allegheny Metal 17 Cr.	Allegheny Ludlum Steel Corp., Pittsburgh	Fe; Cr, 14–18; Ni, 1 max.; C, 0.15 max.	7.7	0.60	0.06	C
24	Alray C	Alloy Metal Wire Co., Prospect Park, Pa.	Fe; Cr, 12; Ni, 62	8.20	....	137	CR, D, R, W
25	Alray D	Alloy Metal Wire Co., Prospect Park, Pa.	Fe; Cr, 35; Ni, 15	7.95			
26	Amsco F-1	American Manganese Steel Co., Chicago Heights, Ill.	Fe; Cr, 15–17; Ni, 34–36; C, 0.35 max.; Si, 1.5; Mn 1	7.9–8.2	0.7	0.03	C
27	Amsco F-3	American Manganese Steel Co., Chicago Heights, Ill.	Fe; Cr, 27–29; Ni, 3 max.; C, 0.30; Si, 1.5 max.; Mn, 1 max.	7.5	1.0	0.059	C
28	Amsco F-5	American Manganese Steel Co., Chicago Heights, Ill.	Fe; Cr, 17–19; Ni, 65–68; C, 0.50 max.; Si, 1.5; Mn, 1 max.	8.0	0.9	0.035	C
29	Amsco F-6	American Manganese Steel Co., Chicago Heights, Ill.	Fe; Cr, 12–14; Ni, 59–62; C, 0.50 max.; Si, 1.5; Mn, 1 max.	....	....	....	C
30	Amsco F-8	American Manganese Steel Co., Chicago Heights, Ill.	Fe; Cr, 20–22; Ni, 8–10; C, 0.20 max.; Si, 1.5 max.; Mn, 1 max.	7.9	0.9	0.06	C
31	Amsco F-10	American Manganese Steel Co., Chicago Heights, Ill.	Fe; Cr, 26–28; Ni, 10–12; C, 0.40 max.; Si, 1.5 max.; Mn, 1 max.	7.8–8.1	0.7–0.9	0.03–0.04	C
32	Amsco Manganese Steel	American Manganese Steel Co., Chicago Heights, Ill.	Fe; C, 1.05–1.20; Mn, 12.5–13; Si, 0.4–0.6	7.81	1.8	0.027	

Copyright by *Chem. & Met. Eng.*, September, 1940.

* "Essential nominal chemical composition" refers to the principal constituents and the approximate proportion present. When Fe is shown without percentage, it may be assumed that it represents the balance of the composition.

† B = bar; C = casting; CR = cold rolled; D = drawn; HR = hot rolled; P = plates; R = rods; S = sheets; T = tubes; W = wire.

Table 3. Materials of Construction—Base Metals. Manufacturers, Chemical Composition, Physical Properties, and Commercial Forms Available—(*Continued*)

No.	Material	Manufacturer	Essential nominal chemical composition,* per cent	Specific gravity	Mean coeff. therm. exp., 32–212°F. (multiply by 10^{-5})	Therm. conduct., C.G.S. units, room temp.	Forms available†
33	Amsco Ni-Mn Welding Rod	American Manganese Steel Co., Chicago Heights, Ill.	Fe; C, 0.75–0.95; Mn, 13–15; Ni, 3.5–4.5	....	.....	....	Welding wire
34	B & W 5150	Babcock & Wilcox Co., New York	Fe; Cr, 1.4; C, 1.0; Mn, 0.4; Si, 0.25				
35	Bethlehem 235	Bethlehem Steel Co., Bethlehem, Pa.	Fe; C, 0.35–0.50; Mn, 1.20–1.75; Si, 0.15–0.25				
36	Bethlehem 300	Bethlehem Steel Co., Bethlehem, Pa.	Fe; C, 0.75–0.85; Mn, 0.7–0.9; Si, 0.15–0.25				
37	Buflokast Gray Iron	Buffalo Foundry & Machine Co., Buffalo, N. Y.	Fe; Cr, 3.2–3.6; Ni, 2 max.; Si, 1–2; Mn, 0.6–0.9	....	0.86	....	C
38	Calmar 18-8	Warman Steel Casting Co., Huntington Pk., Calif.	Fe; Cr, 17–20; Ni, 7–10; C, as specified	....	.....	....	C
39	Calmar 18-8 Cb	Warman Steel Casting Co., Huntington Pk., Calif.	Fe; Cr, 17–20; Ni, 7–10; C, as specified; Cb, 0.9–1.25	....	.....	....	C
40	Calmar 18-8 M	Warman Steel Casting Co., Huntington Pk., Calif.	Fe; Cr, 17–20; Ni, 7–10; C, as specified; Mo, 2.5–4	....	.....	....	C
41	Caloxo 18	Warman Steel Casting Co., Huntington Pk., Calif.	Fe; Cr, 16–20; C, as specified	....	.....	....	C
42	Caloxo 15-35	Warman Steel Casting Co., Huntington Pk., Calif.	Fe; Cr, 14–17; Ni, 33–37; C, as specified	....	.....	....	C
43	Caloxo 25-12	Warman Steel Casting Co., Huntington Pk., Calif	Fe; Cr, 24–27; Ni, 12–14; C, as specified	....	.....	....	C
44	Caloxo 25-20	Warman Steel Casting Co., Huntington Pk., Calif.	Fe; Cr, 23–27; Ni, 17–21; C, as specified	....	.....	....	C
45	Caloxo 28-10	Warman Steel Casting Co., Huntington Pk., Calif.	Fe; Cr, 27–30; Ni, 8–12; C, as specified	....	.....	....	C
46	Carbon-Molybdenum Steel	Lukens Steel Co., Coatesville, Pa.	Fe; C, 0.18–0.28; Mo, 0.4–0.6; Mn, 0.5–0.9	....	.....	....	P, S, HR
47	Causul Metal	Lunkenheimer Co., Cincinnati	Fe; Ni, 18–22; C, 2.2–2.8; Cu, 3.5–5.5; Cr, 1.4–1.9; Mn, 0.9–1.5	7.5–7.6	.....	....	C

48	Chromax	Driver Harris Co., Harrison, N. J.	Fe; Cr, 15; Ni, 35	7.8	0.70	0.036	C
49	Chromax	Driver Harris Co., Harrison, N. J.	Fe; Cr, 19; Ni, 35	7.9	0.75	0.031	B, CR, HR, P, S, R, W
50	Chromel C	Hoskins Manufacturing Co., Detroit	Fe; Cr, 16; Ni, 61; Cr, 0.10	8.24	.....	....	B, CR, HR, D, R , W
51	4-6 Chrome	Spang Chalfant Division, Pittsburgh	Fe; Cr, 4–6; C, 0.10 max.; Mn, 0.5 max.; Ti, 4–6 × C	....	.....	....	HR, T
52	4-6 Chrome	Spang Chalfant Division, Pittsburgh	Fe; Cr, 4–6; C, 0.20 max.; Mn, 0.5 max.	....	0.63	....	HR, T
53	Chrome-copper-Nickel Steel	Lukens Steel Co., Coatesville, Pa.	Fe; Cr, 0.65–0.85; Ni, 0.75 max.; C, 0.12 max.; Cu, 0.45–0.65	....	.....	....	HR, P, S
54	Chrome-Manganese Steel	Lukens Steel Co., Coatesville, Pa.	Fe; Cr, 0.50; Mn, 0.90; C, 0.40	....	.....	....	HR, P, S
55	Cimet	Driver Harris Co., Harrison, N. J.	Fe; Cr, 23–28; Ni, 10–13	8.0	0.89	0.04	C
56	Circle L 1	Lebanon Steel Foundry, Lebanon, Pa.	Fe; C, 0.30; Mn, 1.35; V., 0.10	....	.....	....	C
57	Circle L 2	Lebanon Steel Foundry, Lebanon, Pa.	Fe; Cr, 0.75; C, 0.30; Mn, 1.35; Mo, 0.35	....	.....	....	C
58	Circle L 3	Lebanon Steel Foundry, Lebanon, Pa.	Fe; Cr, 1.0; C, 0.45; Mo. 0.40	....	.....	....	C
59	Circle L 5	Lebanon Steel Foundry, Lebanon, Pa.	Fe; Cr, 0.8; C, 0.30; Ni, 2; Mo, 0.35	....	0.64	....	C
60	Circle L 10	Lebanon Steel Foundry, Lebanon, Pa.	Fe; Cr, 4–6; C, 0.20; Mo, 0.50	....	0.62	0.10	C
61	Circle L 11	Lebanon Steel Foundry, Lebanon, Pa.	Fe; Cr, 18; C, 0.25	....	.....	....	C
62	Circle L 12	Lebanon Steel Foundry, Lebanon, Pa.	Fe; Cr, 11.5–13; C, 0.12 max.	7.75	0.61	0.06	C
63	Circle L 13	Lebanon Steel Foundry, Lebanon, Pa.	Fe; Cr 12–14; C, 0.30	7.8	.....	....	C
64	Circle L 19	Lebanon Steel Foundry, Lebanon, Pa.	Fe; Ni, 2.75; C, 0.18	....	.....	....	C
65	Circle L 21	Lebanon Steel Foundry, Lebanon, Pa.	Fe; Cr, 18–20; Ni, 8–10; C, 0.07 max.; Cb, 10 × C	7.8	0.89	0.063	C
66	Circle L 22	Lebanon Steel Foundry, Lebanon, Pa.	Fe; Cr, 18–20; Ni, 8–10; C, 0.07 max.	7.8	0.89	0.063	C
67	Circle L 22 XM	Lebanon Steel Foundry, Lebanon, Pa.	Fe; Cr, 18–20; Ni, 8–11; C, 0.07 max.; Mo, 2–4	7.8	0.89	0.063	C

---

* "Essential nominal chemical composition" refers to the principal constituents and the approximate proportion present. When Fe is shown without percentage, it may be assumed that it represents the balance of the composition.

† B = bar; C = casting; CR = cold rolled; D = drawn; HR = hot rolled; P = plates; R = rods; S = sheets; T = tubes; W = wire.

Table 3. Materials of Construction—Base Metals. Manufacturers, Chemical Composition, Physical Properties, and Commercial Forms Available—(*Continued*)

No.	Material	Manufacturer	Essential nominal chemical composition,* per cent	Specific gravity	Mean coeff. therm. exp., 32–212°F. (multiply by 10^{-5})	Therm. conduct., C.G.S. units, room temp.	Forms available†
68	Circle L 23	Lebanon Steel Foundry, Lebanon, Pa.	Fe; Cr, 18–20; Ni, 8–10; C, 0.20 max.	7.8	0.89	0.063	C
69	Circle L 30	Lebanon Steel Foundry, Lebanon, Pa.	Fe; Cr, 24; Ni, 12; C, 0.25	7.85	0.85	0.030	C
70	Circle L 31	Lebanon Steel Foundry, Lebanon, Pa.	Fe; Cr, 28; Ni, 11; C, 0.30	7.9	0.78	0.025	C
71	Circle L 34	Lebanon Steel Foundry, Lebanon, Pa.	Fe; Cr, 20; Ni, 30; C, 0.07 max.; Mo, 3; Cu, 5	7.9	0.80	0.05	C
72	Colonial 610	Vanadium-Alloys Steel Co., Latrobe, Pa.	Fe; Cr, 16–18; Ni, 1; C, 0.12 max.; S, optional; Mn, 0.25–.4	7.70	0.65	0.058	HR, D, P, S, R, W, B
73	Cooper 14	Cooper Alloy Foundry Co., Elizabeth, N. J.	Fe; Cr, 11–14; C, 0.12–0.35	7.65	0.57	0.06	B, C, CR, HR, D, P, S, R, T, W
74	Cooper 16	Cooper Alloy Foundry Co., Elizabeth, N. J.	Fe; Cr, 14–18; C, 0.12–0.35	7.71	0.64	0.06	B, C, CR, HR, D, P, S, R, T, W
75	Cooper 17	Cooper Alloy Foundry Co., Elizabeth, N. J.	Fe; Cr, 17.5–19; Ni, 8–10; C, 0.08–0.20	7.85	0.89	0.069	B, C, CR, HR, D, P, S, R, T, W
76	Cooper 17 S	Cooper Alloy Foundry Co., Elizabeth, N. J.	Fe; Cr, 18–20; Ni, 8–10; C, 0.08-max.	7.85	0.89	0.069	B, C, CR, HR, D, P, S, R, T, W
77	Cooper 17 S Mo	Cooper Alloy Foundry Co., Elizabeth, N. J.	Fe; Cr, 16–20; Ni, 14 max.; C, 0.10 max.; Mo, 2–4	7.85	0.89	0.06	B, C, CR, HR, D, P, S, R, T, W
78	Cooper 17 S CB	Cooper Alloy Foundry Co., Elizabeth, N. J.	Fe; Cr, 18–20; Ni, 8–10; C, 0.08 max.; Cb, 10 × C	7.85	0.89	0.069	B, C, CR, HR, D, P, S, R, T, W
79	Cooper 18	Cooper Alloy Foundry Co., Elizabeth, N. J.	Fe; Cr, 10; Ni, 22; C, 0.20; Mn, 0.50	8.0	1.0	0.074	B, C, CR, HR, D, P, S, R, T, W
80	Cooper 19 A	Cooper Alloy Foundry Co., Elizabeth, N. J.	Fe; Cr, 26–30; Ni, 2–3; C, 0.20–0.50; Mn, 0.50	7.6	0.56	0.064	B, C, CR, HR, D, P, S, R, T, W
81	Cooper 19	Cooper Alloy Foundry Co., Elizabeth, N. J.	Fe; Cr, 26–30; C, 0.20–0.50; Mn, 0.50	7.6	0.56	0.064	B, C, CR, HR, D, P, S, R, T, W
82	Cooper 20	Cooper Alloy Foundry Co., Elizabeth, N. J.	Fe; Cr, 15–18; Ni, 35–38; C, 0.2–0.5; Mn, 0.50	7.95	0.75	0.028	C

83	*Cooper 21*	*Cooper Alloy Foundry Co., Elizabeth, N. J.*	*Fe; Cr, 15; Ni, 65; C, 0.50 max.; Mn, 0.50*	*8.20*	*0.76*	*0.033*	*C*
84	Cooper 21 A-B-C	Cooper Alloy Foundry Co., Elizabeth, N. J.	Fe; Cr, 15–20; Ni, 20–25; C, 0.07–0.10; Mo, 3; Si, 1.5	7.9	0.85	0.055	C
85	Cooper 22	Cooper Alloy Foundry Co., Elizabeth, N. J.	Fe; Cr, 24–28; Ni, 10–12; C, 0.20–0.50	7.90	0.90	0.035	B, C
86	Cooper 22 P-M	Cooper Alloy Foundry Co., Elizabeth, N. J.	Fe; Cr, 29; Ni, 9; C, 0.2–0.5; Mo, 3	7.90	0.90	0.035	C
87	Cooper KNC 3	Cooper Alloy Foundry Co., Elizabeth, N. J.	Fe; Cr, 24–26; Ni, 19–21; C, 0.25 max.	7.85	0.95	0.03	C
88	Corrosiron	Pacific Foundry Co., San Francisco	Fe; C, 0.8–1.0; Si, 14.50; Mn, 0.50	7.02	.....	....	C
89	Crane 18-8 Mo	Crane Co., Chicago	Fe; Cr, 18–20; Ni, 9–10; C, 0.10; Mo, 2–4	8.02	0.96	0.069	C
90	Crane 5 Cr-Mo Cast Steel	Crane Co., Chicago	Fe; Cr, 4–6; C, 0.30 max.; Mo, 0.55	.....	.....	....	C
91	Croloy 2	Babcock & Wilcox Tube Co., Beaver Falls, Pa.	Fe; Cr, 2; C, 0.15 max.; Mo, 0.50	7.70	0.745	....	B, C, HR, P, S, T, W
92	Croloy 5	Babcock & Wilcox Tube Co., Beaver Falls, Pa.	Fe; Cr, 4–6; C, 0.15 max.; Mo, 0.45–0.65	7.7	0.64	....	B, C, HR, D, P, S, R, T, W
93	Croloy 9	Babcock & Wilcox Tube Co., Beaver Falls, Pa.	Fe; Cr, 8–10; C, 0.15 max.; Mo, 1.2–1.5	7.80	0.63	....	B, C, P, S, R, T, W
94	Croloy 12	Babcock & Wilcox Tube Co., Beaver Falls, Pa.	Fe; Cr, 12–14; C, 0.15 max.; Ni, 0.50 max.	7.72	0.61	0.0595	B, C, HR, D, S, R, T, W
95	Croloy 16-13-3	Babcock & Wilcox Tube Co., Beaver Falls, Pa.	Fe; Cr, 17; Ni, 13; C, 0.10 max.; Mo, 2.75	8.02	0.96	0.038	B, C, CR, HR, P, S, R, T, W
96	Croloy 18	Babcock & Wilcox Tube Co., Beaver Falls, Pa.	Fe; Cr, 15–18; Ni, 0.50 max.; C, 0.12 max.	7.56	0.60	0.0582	B, C, P, S, R, T, W
97	Croloy 18-8	Babcock & Wilcox Tube Co., Beaver Falls, Pa.	Fe; Cr, 18; Ni, 9.5; C, 0.07 max.; Mn, 0.75 max.	7.92	0.92	0.039	B, C, CR, HR, P, S, R, T, W
98	Croloy 25-20	Babcock & Wilcox Tube Co., Beaver Falls, Pa.	Fe; Cr, 25; Ni, 20; C, 0.10 max.	7.96	0.92	0.039	B, C, HR, P, S, R, T, W
99	Croloy 27	Babcock & Wilcox Tube Co., Beaver Falls, Pa.	Fe; Cr, 27; Ni, 1 max.; C, 0.20 max.; N, 0.12–0.25	7.47	0.59	0.05	B, C, P, S, R, T, W
100	Dopploy	Sowers Manufacturing Co., Buffalo, N. Y.	Fe; Cr, 2.35; Ni, 18.50; C, 2.85; Mn, 1	7.49	.....	....	C
101	Duraloy 18-8 S	Duraloy Co., Scottdale, Pa.	Fe; Cr, 17–19; Ni, 7–9; C, 0.07 max.; Mn, 1	.....	.....	....	C
102	Duraloy 18-8 S Mo	Duraloy Co., Scottdale, Pa.	Fe; Cr, 17–19; Ni, 7–9; C, 0.07 max.; Mo, 2–4; Mn, 1	.....	.....	....	C
103	Duraloy 35-15	Duraloy Co., Scottdale, Pa.	Fe; Cr, 14–16; Ni, 34–36; C, 0.50	.....	0.75	....	C

---

* "Essential nominal chemical composition" refers to the principal constituents and the approximate proportion present. When Fe is shown without percentage, it may be assumed that it represents the balance of the composition.

† B = bar; C = casting; CR = cold rolled; D = drawn; HR = hot rolled; P = plates; R = rods; S = sheets; T = tubes; W = wire.

Table 3. Materials of Construction—Base Metals. Manufacturers, Chemical Composition, Physical Properties, and Commercial Forms Available—(*Continued*)

No.	Material	Manufacturer	Essential nominal chemical composition,* per cent	Specific gravity	Mean coeff. therm. exp., 32–212°F. (multiply by 10^{-5})	Therm. conduct., C.G.S. units, room temp.	Forms available†
104	Duraloy 25-20 Mo	Duraloy Co., Scottdale, Pa.	Fe; Cr, 24–26; Ni, 19–21; C, 0.20; Mo, 2–3	.....	.....	......	C
105	Duraloy A	Duraloy Co., Scottdale, Pa.	Fe; Cr, 27–30; C, 0.25 max.; Mn, 1	7.6	.....	......	C
106	Duraloy B	Duraloy Co., Scottdale, Pa.	Fe; Cr, 16–18; C, 0.20 max.; Mn, 1	7.7	.....	......	C
107	Duraloy N	Duraloy Co., Scottdale, Pa.	Fe; Cr, 23–25; Ni, 11–13; C, 0.35; Mn, 1	7.86	.....	......	C
108	Duraloy NS Mo	Duraloy Co., Scottdale, Pa.	Fe; Cr, 23–25; Ni, 11–13; C, 0.15 max.; Mo, 2–3	.....	.....	......	C
109	Durco D-18	Duriron Co., Dayton, Ohio	Fe; Cr, 17–21; C, 0.20	7.6	.....	......	C
110	Durco KA 2S	Duriron Co., Dayton, Ohio	Fe; Cr, 18–20; Ni, 8–10; C, 0.07 max.	7.80	0.9	0.05	C
111	Durco KA2S Mo	Duriron Co., Dayton, Ohio	Fe; Cr, 18–20; Ni, 8–10; C, 0.07 max.; Mo, 2–4	7.84	0.9	0.05	C
112	Durichlor	Duriron Co., Dayton, Ohio	Fe; C, 0.85; Si, 14.5; Mo, 3; Mn, 0.35	7.04	0.36	0.125	C
113	Durimet	Duriron Co., Dayton, Ohio	Fe; Cr, 19; Ni, 22; C, 0.07 max.; Mo, 3.5; Cu, 1	7.85	0.78	0.05	B, C, HR, D, P, S, R, W
114	Duriron	Duriron Co., Dayton, Ohio	Fe; C, 0.80; Si, 14.50; Mn, 0.35	7.0	0.36	0.125	C
115	Economet	General Alloys Co., So. Boston	Fe; Cr, 10; Ni, 30; C, ± .60; Mn, 1	7.70	.....	......	C
116	Economy Hardface-Self Hardening	American Manganese Steel Co., Chicago Heights, Ill.	Fe; C; Cr; Mo; Mn				
117	Elverite A	Babcock & Wilcox Co., New York	Fe; C, 3.00–3.5; Mn, 0.35; Si, 0.25–1.0				
118	Elverite B	Babcock & Wilcox Co., New York	Fe; Cr, 1–1.8; Ni, 3.75–4.75; C, 3–3.35; Si, 0.25–1				
119	Elverite C	Babcock & Wilcox Co., New York	Fe; Cr, 1–1.8; C, 3–3.5; Si, 0.25–1; Ni, 3.75–4.75				

120	Empire 8	Empire Steel Castings, Reading, Pa.	Fe; Cr, 1–1.25; C, 0.30–0.35; Mo, 0.25–0.3; Mn, .6–.8	.....	.....	......	C
121	Empire 11	Empire Steel Castings, Reading, Pa.	Fe; Cr, 1–1.25; Ni, 2.25–2.75; C, 0.25–0.3; Mo, .3–.35	.....	.....	......	C
122	Empire 12	Empire Steel Castings, Reading, Pa.	Fe; Cr, 4.5–6.0; C, 0.20–0.30; Mn, 0.40–0.70	.....	.....	......	C
123	Empire 14	Empire Steel Castings, Reading, Pa.	Fe; Cr, 12–14; C, 0.15–0.25; Mn, 0.40–0.70	.....	.....	......	C
124	Empire 15	Empire Steel Castings, Reading, Pa.	Fe; Cr, 16–20; Ni, 3.0 max.; C, 0.15–0.30; Mn, 0.40–0.70				
125	Empire 16	Empire Steel Castings, Reading, Pa.	Fe; Cr, 17–21; Ni, 7–9; C, 0.20 max.; Mn, 0.40–0.70	.....	.....	......	C
126	Empire 17	Empire Steel Castings, Reading, Pa.	Fe; Cr, 26–30; Ni, 3.0 max.; C, 0.15–0.30; Mn, 0.40–0.70	.....	.....	......	C
127	Empire 20	Empire Steel Castings, Reading, Pa.	Fe; Cr, 23–28; Ni, 10–13; C, 0.15–0.20; Mn, 0.40–0.70	.....	.....	......	C
128	Empire 23	Empire Steel Castings, Reading, Pa.	Fe; Cr, 13–17; Ni, 34–37; C, 0.15–0.25; Mn, 0.40–0.70	.....	.....	......	C
129	Esco 45	Electric Steel Foundry Co., Portland, Ore.	Fe; Cr, 19; Ni, 9; C, 0.15 max.; Mn, 1; Mo, 4.25	8.05	0.84	0.052	C
130	Fahrite C 7	Ohio Steel Foundry Co., Springfield, Ohio	Fe; Cr, 24–27; Ni, 10–13; C, 0.20 max.; Mn, 1.25 max.	7.8	0.82	0.04	B, C
131	Fahrite C 7 M	Ohio Steel Foundry Co., Springfield, Ohio	Fe; Cr, 24–27; Ni, 10–13; C, 0.20 max.; Mn, 1.25 max.; Mo, 2.5–3.5	7.9	0.82	......	B, C
132	Fahrite C 8 A	Ohio Steel Foundry Co., Springfield, Ohio	Fe; Cr, 18–21; Ni, 8–10; C, 0.07 max.; Mn, 1	7.7	0.85	......	C
133	Fahrite C 8 A M	Ohio Steel Foundry Co., Springfield, Ohio	Fe; Cr, 18–21; Ni, 8–10; C, 0.07 max.; Mn, 1 max.; Mo, 2.5–3.5	7.8	0.85	......	C
134	Fahrite C 12	Ohio Steel Foundry Co., Springfield, Ohio	Fe; Cr, 12–14; Ni, 1.0 max.; C, 0.16 max.; Mn, 1.0 max.	7.6	0.56	......	C
135	Fahrite C 73	Ohio Steel Foundry Co., Springfield, Ohio	Fe; Cr, 28–30; Ni, 8–10; C, 0.20–0.30; Mn, 1.0 max.	7.8	0.84	......	C
136	Genesee 212	Symington-Gould Corp., Rochester, N. Y.	Fe; Cr, 14; C, 0.15; Mn, 0.50; Mo, 0.20	7.72	.....	......	C
137	Genesee 255	Symington-Gould Corp., Rochester, N. Y.	Fe; Cr, 5; C, 0.20; Mn, 0.70; Mo, 0.50	7.78			
138	Genesee 304	Symington-Gould Corp., Rochester, N. Y.	Fe; Cr, 17; Ni, 35; C, 0.45; Mn, 0.65				
139	Genesee 305	Symington-Gould Corp., Rochester, N. Y.	Fe; Cr, 17; Ni, 65; C, 0.35; Mn, 0.60	.....	.....	......	C

---

* "Essential nominal chemical composition" refers to the principal constituents and the approximate proportion present. When Fe is shown without percentage, it may be assumed that it represents the balance of the composition.

† B = bar; C = casting; CR = cold rolled; D = drawn; HR = hot rolled; P = plates; R = rods; S = sheets; T = tubes; W = wire.

Table 3. Materials of Construction—Base Metals. Manufacturers, Chemical Composition, Physical Properties, and Commercial Forms Available—(*Continued*)

No.	Material	Manufacturer	Essential nominal chemical composition,* per cent	Specific gravity	Mean coeff. therm. exp., 32–212°F. (multiply by 10^{-5})	Therm. conduct., C.G.S. units, room temp.	Forms available†
140	Genesee 412	Symington-Gould Corp., Rochester, N. Y.	Fe; C, 1.10; Mn, 11–14	.....	1.0	0.027	C
141	Hard Facing Rod 217	American Manganese Steel Co., Chicago Heights, Ill.	Fe; C; Cr; Mn; W; Mo	.....	.....	....	Welding rod
142	Hard Facing Welding Rod 459	American Manganese Steel Co., Chicago Heights, Ill.	Fe; C; Cr; Mn; Mo	.....	.....	....	Welding rod
143	Hoskins 502	Hoskins Manufacturing Co., Detroit	Fe; Cr, 18.5; Ni, 35; C, 0.40; Mn, 1.5	7.9	0.89		
144	Ing Aclad Stainless Clad Steel 304	Ingersoll Steel & Disc Division, Borg Warner Corp., Chicago	Fe; Cr, 18–20; Ni, 8–10; C, 0.08 max.	7.92	.....	......	HR, P, S
145	Ing Aclad Stainless Clad Steel 309	Ingersoll Steel & Disc Division, Borg-Warner Corp., New Castle, Ind.	Fe; Cr, 22–26; Ni, 12–14; C, 0.20 max.; Mn, 2 max.	7.90	.....	......	HR, P, S
146	Ing Aclad Stainless Clad Steel 317	Ingersoll Steel & Disc Division, Borg-Warner Corp., New Castle, Ind.	Fe; Cr, 18–20; Ni, 10–14; C, 0.10 max.; Mn, 2 max.; Mo, 3–4	7.92	.....	......	HR, P, S
147	Ing Aclad Stainless Clad Steel 347	Ingersoll Steel & Disc Division, Borg-Warner Corp., New Castle, Ind.	Fe; Cr, 17–20; Ni, 8–12; C, 0.10 max.; Mn, 2 max.; Cb, 10 × C	7.92	.....	......	HR, P, S
148	Manganese-Molybdenum Steel	Lukens Steel Co., Coatesville, Pa.	Fe; C, 0.25 max.; Mn, 1.65 max.; Mo, 0.75 max.; Si, 0.25 max.	.....	.....	......	HR, P, S
149	Mayari R	Bethlehem Steel Co., Bethlehem, Pa.	Fe; Cr, 0.2–1.0; Ni, 0.25–0.75; C, 0.12 max.; Cu, 0.5–0.7; Mn, 0.5–1	.....	.....	......	B, CR, HR, P, S, R, T, W, shapes
150	Midvaloy 13	Midvale Co., Nicetown, Philadelphia	Fe; Cr, 12–14; C, 0.12 max.; Ni, 0.50 max.; Mn, 0.50 max.	7.77	0.6	0.06	B, C, HR, forgings
151	Midvaloy 18-8	Midvale Co., Nicetown, Philadelphia	Fe; Cr, 18–20; Ni, 8–10; C, 0.20 max.; Mn, 0.60 max.	7.86	0.91	0.04	B, C, HR, forgings

152	Midvaloy 18-8 Se	Midvale Co., Nicetown, Philadelphia	Fe; Cr, 17–19; Ni, 8.5–9.5; C, 0.12 max.; Se, 0.22–0.26; Mn, 0.8–1.2	7.86	0.91	0.052	B, HR
153	Midvaloy 1700	Midvale Co., Nicetown, Philadelphia	Fe; Cr, 15–18; C, 0.12 max.; Mn, 0.50 max.	7.71	0.57	0.06	B, C, HR
154	Midvaloy 17-35	Midvale Co., Nicetown, Philadelphia	Fe; Cr, 17; Ni, 35; C, 0.35	7.93	0.74	0.03	C
155	Midvaloy 25-12	Midvale Co., Nicetown, Philadelphia	Fe; Cr, 23–26; Ni, 11–13; C, 0.25 max.; Mn, 0.4–0.6	7.86	0.90	0.035	B, C, HR
156	Midvaloy 25–20	Midvale Co., Nicetown, Philadelphia	Fe; Cr, 24–26; Ni, 19–21; C, 0.25 max.	7.8	0.8	......	B, C, HR forgings
157	Midvaloy 2802	Midvale Co., Nicetown, Philadelphia	Fe; Cr, 25–30; Ni, 2.0 max.; C, 0.35 max.; Mn, 0.2–0.5	7.6	0.59	0.05	B, C, HR
158	Midvaloy ATV 3	Midvale Co., Nicetown, Philadelphia	Fe; Cr, 14; Ni, 26.5; C, 0.48; W, 3.5	8.12	0.78	......	B, C, forgings
159	Misco 18-8	Michigan Steel Casting Co., Detroit	Fe; Cr, 18–20; Ni, 8–10; C, 0.07–0.16;[1] Mo, or Cb	......	......	......	C
160	Misco C	Michigan Steel Casting Co., Detroit	Fe; Cr, 27–30; Ni, 8–10; C, 0.20–0.30	......	......	......	B, C, HR, R
161	Misco HN-1	Michigan Steel Casting Co., Detroit	Fe; Cr, 16–20; Ni, 66–70; C, 0.50–0.80	......	......	......	C
162	Misco HN-2[2]	Michigan Steel Casting Co., Detroit	Fe; Cr, 10–14; Ni, 58–62; C, 0.50–0.80	8.13	......	......	B, C, HR, R
163	Misco Metal[2]	Michigan Steel Casting Co., Detroit	Fe; Cr, 15–17; Ni, 35–37; C, 0.40–0.60	7.94	0.63	......	B, C, HR, S, R
164	Miscroms 1	Michigan Steel Casting Co., Detroit	Fe; Cr, 13–15; C, 0.25–0.35; Ni, 0.60 max.	......	......	......	C
165	Miscrome 4	Michigan Steel Casting Co., Detroit	Fe; Cr, 11.5–13; C, 0.12 max.; Ni, 0.80 max.	......	......	......	C
166	Nichrome	Driver Harris Co., Harrison, N. J.	Fe; Cr, 15; Ni, 60	8.25	0.76	0.03	B, CR, HR, P, S, R, T, W
167	Nichrome	Driver Harris Co., Harrison, N. J.	Fe; Cr, 11–13; Ni, 59–61	8.15	0.62	0.033	C
168	2% Nickel Steel	Lukens Steel Co., Coatesville, Pa.	Fe; Ni, 2.0 min.; C, 0.20 max.	......	......	......	HR, P, S
169	Ni-Hard	International Nickel Co., New York	Fe; Ni, 4.4–4.6; C, 2.75–3.6; Cr, 1.4–1.6; Si, 0.5–1.5	7.40	......	......	C
170	Nilstain	Wilber B. Driver Co., Newark, N.J	Fe; Cr, 18–20; Ni, 8–10; C, <0.20; Mn, 2 max.	7.9	0.95	0.048	HR, D, R, W

---

* "Essential nominal chemical composition" refers to the principal constituents and the approximate proportion present. When Fe is shown without percentage, it may be assumed that it represents the balance of the composition.

† B = bar; C = casting; CR = cold rolled; D = drawn; HR = hot rolled; P = plates; R = rods; S = sheets; T = tubes; W = wire.

[1] Depending on service.

[2] Composition given is for castings.

Table 3. Materials of Construction—Base Metals. Manufacturers, Chemical Composition, Physical Properties, and Commercial Forms Available—(*Continued*)

No.	Material	Manufacturer	Essential nominal chemical composition,* per cent	Specific gravity	Mean coeff. therm. exp., 32–212°F. (multiply by 10^{-5})	Therm. conduct., C.G.S. units, room temp.	Forms available†
171	Ni-Resist	International Nickel Co., New York	Fe: Ni, 13–15; Cu, 5.5–7; C, 2.95–3.10; Cr, 1.5–2.5; Mn, 1–1.75	7.4	1.03	0.095	C
172	Pyrocast	Pacific Foundry Co., San Francisco	Fe; Cr, 22–30; C, 1.75–2.0	7.54	.....	......	C
173	Q Alloy A	General Alloys Co., Boston	Fe; Cr, 17; Ni, 68; C, 0.60; Mn, 1.0	8.20	0.59	0.033	B, C, HR, P, S, R, W
174	Q Alloy B	General Alloys Co., Boston	Fe; Cr, 13; Ni, 60; C, 0.60; Mn, 1.0	8.13	0.64	0.033	C
175	Q Alloy CN-1	General Alloys Co., Boston	Fe; Ni, 12; C, 0.10 max.;[3] Cb, Se, or Ti if specified	7.81	0.90	0.055	B, C, HR, P, S, R, W
176	Q Alloy CN-1-H	General Alloys Co., Boston	Fe; Cr, 28; Ni, 10; C, 0.30[4]	7.65	0.88	......	C
177	Q Alloy CN-1-Mo	General Alloys Co., Boston	Fe; Cr, 24; Ni, 12; C, 0.10 max.;[3] Mo, 1–4; Cb, Se, or Ti	7.81	0.90	0.055	C, HT, P, S, R
178	Q Alloy CN-2	General Alloys Co., Boston	Fe; Cr, 18; Ni, 8; C, 0.10 max.;[3] Cb, Se, or Ti if specified	7.70	0.92	......	C, HR, D, P, S, R, T, W
179	Q Alloy CN-2 Mo	General Alloys Co., Boston	Fe; Cr, 18; Ni, 8; C, 0.10 max.;[3] Mo, 1–4; Cb, Se, or Ti	7.70	0.92	......	B, C, HR, D, P, S, R, T, W
180	O-4	Otis Elevator Co., Buffalo, N. Y.	Fe; Cr, 4–8; C, 0.15–0.35; Mo, 0.25–2; Mn, 0.45–0.85	7.78–7.82	0.62		
181	O-12	Otis Elevator Co., Buffalo, N. Y.	Fe; Cr, 11–14; C, 0.12–0.35; Mn, 0.4–0.6	7.65–7.75	0.61		
182	O-16	Otis Elevator Co., Buffalo, N. Y.	Fe; Cr, 14–18; C, 0.12 max.; Mn, 0.5 max.	....	.....	0.057	
183	O-18	Otis Elevator Co., Buffalo, N. Y.	Fe; Cr, 18–20; Ni, 8–10; C, 0.08–0.20; Mn, 0.5–0.6	7.86–7.94	0.96	0.069	C
184	O-24	Otis Elevator Co., Buffalo, N. Y.	Fe; Cr, 22–25; Ni, 10–13; C, 0.20 max.	7.86–7.94	0.9	0.039	

185	*G-30*	*Otis Elevator Co., Buffalo, N. Y.*	*Fe; Cr, 25–30; C, 0.25 max.; Mn, 1.0 max.*	....	.....	*0.059*	
186	Sivyer 5% Cr	Sivyer Steel Castings Co., Milwaukee, Wis.	Fe; Cr, 4.5–6.5; C, 0.25 max.; Mo, 0.45–0.65	....	.....	......	C
187	Sivyer 60	Sivyer Steel Castings Co., Milwaukee, Wis.	Fe; Cr, 18; Ni, 8; C, 0.12 max.; Mn, 0.60 max.	7.85	0.90	......	C
188	Sivyer 62	Sivyer Steel Castings Co., Milwaukee, Wis.	Fe; Cr, 23–25; Ni, 11–13; C, 0.20 max.; Mn, 0.60 max.	7.8	0.90	......	C
189	Sivyer 64	Sivyer Steel Castings Co., Milwaukee, Wis.	Fe; Cr, 26–28; Ni, 9–11; C, 0.20 max.; Mn, 0.60 max.	7.8	0.90	......	C
190	Sivyer 66	Sivyer Steel Castings Co., Milwaukee, Wis.	Fe; Cr, 12–14; C, 0.15 max.; Mn, 0.60 max.	7.7	0.58	......	C
191	Sivyer 70	Sivyer Steel Castings Co., Milwaukee, Wis.	Fe; Cr, 15–17; Ni, 34–36; C, 0.40	....	.....	......	C
192	Spang Chalfant 1	Spang Chalfant Division, National Supply Co., Pittsburgh	Fe; C, 0.15; Cr, 1.75–2.25; Mn, 0.3–0.6; Mo, 0.45–0.65	....	0.63	......	HR, T
193	Spang Chalfant 2	Spang Chalfant Division, National Supply Co., Pittsburgh	Fe; C, 0.1–0.2; Mn, 0.3–0.6; Si, 0.25 max.	....	0.61	......	HR
194	Spang Chalfant 3	Spang Chalfant Division, National Supply Co., Pittsburgh	Fe; C, 0.1–0.2; Mo, 0.45–0.65; Mn, 0.3–0.6; Si, 0.25 max.	....	0.60	......	HR
195	Stainless Steel 301		Fe; Cr, 16–18; Ni, 7–9; C, 0.09–0.20; Mn, 1.25 max.	7.9	0.96	0.05	CR, HR, P, S, B, D, W
196	Stainless Steel 302		Fe; Cr, 17.5–20; Ni, 8–10; C, > 0.08–0.20; Mn, 1.25 max.	7.93	0.95	0.048	B, CR, HR, P, S, D, W, R, T
197	Stainless Steel 302 B		Fe; Cr, 17.5–20; Ni, 8–10; C, > 0.08; Si, 2–3; Mn, 1.25 max.	7.93	.....	......	B, D, W, P, S, HR, CR
198	Stainless Steel 303	Note: Physical and chemical properties for those stainless steels in *wrought* form which have been assigned type numbers by the American Iron & Steel Institute are grouped herewith at the suggestion of	Fe; Cr, 17.5–20; Ni, 8–10; C, 0.2 max.; S or Se, 0.07 min, or Mo, 0.60 max.	7.93	0.93	0.042	B, HR, D, CR, B, S, P, W
199	Stainless Steel 304		Fe; Cr, 18–20; Ni, 8–10; C, 0.08 max.; Mn, 2.0 max.	....	0.96	0.039	B, CR, HR, D, P, S, R, T, W
200	Stainless Steel 308		Fe, Cr, 19–22; Ni, 10–12; C, 0.08 max.; Mn, 2.0 max.	7.90	0.96	0.039	B, CR, HR, D, P, S, R, T, W

---

* "Essential nominal chemical composition" refers to the principal constituents and the approximate proportion present. When Fe is shown without percentage, it may be assumed that it represents the balance of the composition.

† B = bar; C = casting; CR = cold rolled; D = drawn; HR = hot rolled; P = plates; R = rods; S = sheets; T = tubes; W = wire.

[2] Higher carbon if specified.

Table 3. Materials of Construction—Base Metals. Manufacturers, Chemical Composition, Physical Properties, and Commercial Forms Available—(*Continued*)

No.	Material	Manufacturer	Essential nominal chemical composition,* per cent	Specific gravity	Mean coeff. therm. exp., 32–212°F. (multiply by 10^{-5})	Therm. conduct., C.G.S. units, room temp.	Forms available†
201	Stainless Steel 309	the Stainless Steel Technical Committee of the producers. These stainless steels in *wrought* form are generally available from the following producers: Allegheny-Ludlum Steel Corp., Pittsburgh American Rolling Mill Co., Middletown, Ohio Alloy Metal Wire Co., Prospect Park, Pa. Babcock & Wilcox Tube Co., Beaver Falls, Pa. Bethlehem Steel Co., Bethlehem, Pa. Carpenter Steel Co., Reading, Pa. Cooper Alloy Foundry Co., Elizabeth, N. J. Crucible Steel Co., New York, N. Y. Firth Sterling Steel Co., McKeesport, Pa. Forging and Casting Corp., Ferndale, Mich. Henry Disston & Sons, Philadelphia, Pa. Ingersoll Steel & Disc Division, Borg-Warner Corp., New Castle, Ind.	Fe; Cr, 22–26; Ni, 12–14; C, 0.20 max.	7.88	0.83	0.035	CR, HR, P, S, D, R, W, B
202	Stainless Steel 309 S		Fe; Cr, 22–26; Ni, 12–14; C, 0.08 max.	7.80	0.87	......	B, CR, HR, D, P, S, W
203	Stainless Steel 310		Fe; Cr, 24–26; Ni, 19–21; C, 0.25 max.	7.98	0.92	0.039	B, HR, D, R, W, P, S, T, CR
204	Stainless Steel 311		Fe; Cr, 19–21; Ni, 24–26; C, 0.25 max.	7.84	.....	......	B, CR, HR, D, P, S, W
205	Stainless Steel 312		Fe; Cr. 27–31; Ni, 8–10; C, 0.25 max.	.....	.....	......	B, D, W, P, S
206	Stainless Steel 315		Fe; Cr, 17–19; Ni, 7–9.5; C, 0.15 max.; Cu, 1–1.5; Mo, 1–1.5	7.86	0.93	0.038	B, HR, D, P, S, R
207	Stainless Steel 316		Fe; Cr, 16–18; Ni, 10–14; C, 0.10 max.; Mo, 2–3	8.03	0.90	0.041	B, CR, HR, D, P, S, R, T, W
208	Stainless Steel 317		Fe; Cr, 18–20; Ni, 10–14; C, 0.10 max.; Mo, 3–4	8.03	0.96	0.038	B, CR, HR, P, S, R, T, W
209	Stainless Steel 321		Fe; Cr, 17–20; Ni, 7–10; C, 0.10 max.; Ti, min. 4 × C	7.98	.....	......	B, CR, HR, D, P, S, R, T, W
210	Stainless Steel 325		Fe; Cr, 7–10; Ni, 19–23; C, 0.25 max.; Cu, 1–1.5	8.0	0.93	0.074	B, CR, HR, D, P, S, W
211	Stainless Steel 327		Fe; Cr, 25–30; Ni, 3–5; C, 0.25 max.	.....	.....	......	B, CR, HR, D, P, S, W
212	Stainless Steel 329		Fe; Cr, 25–30; Ni, 3–5; C, 0.10 max.; Mo, 1–1.5	7.69	0.8	......	B, CR, HR, D, P, S, W
213	Stainless Steel 330		Fe; Cr, 14–16; Ni, 33–36; C, 0.25 max.	7.92	0.72	0.027	B, D, P, S, W, HR
214	Stainless Steel 347		Fe; Cr, 17–20; Ni, 8–12; C, 0.10 max.; Cb, 10 × C	8.02	0.90	0.038	B, CR, HR, D, P, S, R, T, W
215	Stainless Steel 403		Fe; Cr, 11.5–13; C, 0.15 max.	7.65	0.57	......	B, CR, HR, D, P, S, R, T, W

216	Stainless Steel 405	Jessop Steel Co., Washington, Pa. Latrobe Electric Steel Co., Latrobe, Pa. Midvale Co., Philadelphia, Pa. Spang Chalfant Division, National Supply Co., Pittsburgh Timken Roller Bearing Co., Canton, Ohio Republic Steel Corp., Cleveland, Ohio Rustless Iron & Steel Corp., Baltimore, Md. Superior Steel Corp., Carnegie, Pa. United States Steel Corp., Pittsburgh Universal Cyclops Steel Corp., Bridgeville, Pa. Vanadium Alloys Steel Co., Latrobe, Pa.	Fe; Cr, 11.5–13.5; C, 0.08; Al, 0.10–0.20	7.65	0.57	......	B, CR, HR, D, P, S, W
217	Stainless Steel 406		Fe; Cr, 12–14; C, 0.15 max.; Al, 4–4.5	.....	.....	......	B, CR, HR, D, P, S, W
218	Stainless Steel 410		Fe; Cr, 10–14; C, 0.15 max.	7.65	0.61	0.06	B, CR, HR, D, P, S, R, T, W
219	Stainless Steel 414		Fe; Cr, 10–14; Ni, 2 max.; C, 0.15 max.	7.76	.....	0.039	B, HR, D, P, S, R, W
220	Stainless Steel 416		Fe; Cr, 12–14; C, 0.15 max.; S or Se, 0.07 min. or Mo, 0.60 max.	7.73	0.57	0.046	B, HR, R, D, W, CR, P, S, T
221	Stainless Steel 418		Fe; Cr, 12–14; C, 0.15 max.; W, 2.5–3.5				
222	Stainless Steel 420		Fe; Cr, 12–14; C, >0.15	7.77	0.57	0.05	B, HR, D, R, W, CR, S, P
223	Stainless Steel 420 F		Fe; Cr, 12–14; C, >0.15; S or Se, 0.07 min. or Mo, 0.60 max.	7.75	.....	0.05	
224	Stainless Steel 430		Fe; Cr, 14–18; C, 0.12 max.	7.71	0.64	0.06	B, CR, HR, P, S, R, T, W
225	Stainless Steel 430 F		Fe; Cr, 14–18; C, 0.12 max.; S or Se, 0.07 min. or Mo, 0.60 max.	7.70	.....	......	B, CR, HR, D, R, W
226	Stainless Steel 431		Fe; Cr, 14–18; Ni, 2 max.; C, 0.15 max.	7.70	.....	......	B, CR, HR, D, R, W, S
227	Stainless Steel 440		Fe; Cr, 14–18; C, >0.12	7.72	0.57	......	B, CR, HR, D, R, W, S
228	Stainless Steel 441		Fe; Cr, 14–18; Ni, 2 max.; C, >0.15	7.65	0.65	0.058	B, CR, HR, D, P, S, R, W
229	Stainless Steel 442		Fe; Cr, 18–23; C, 0.35 max.	.....	.....	......	B, CR, HR, D, P, S, R
230	Superior Steel 446		Fe; Cr, 23–30; C, 0.35 max.	7.54	0.59	0.05	B, CR, HR, P, S, R, T, W, D
231	Stainless Steel 501		Fe; Cr, 4–6; C, >0.10	7.7	0.64	......	B, CR, HR, D, P, S, R, T, W
232	Stainless Steel 502		Fe; Cr, 4–6; C, 0.10 max.	7.83	.....	......	B, CR, HR, D, P, S, R, T, W
233	Standard Alloy H R 4	Standard Alloy Co., Cleveland, Ohio	Fe; Cr, 12–14; Ni, 59–62; C, 0.60–0.80	.....	.....	....	C, HR, P
234	Standard Alloy H R 3	Standard Alloy Co., Cleveland, Ohio	Fe; Cr, 15–17; Ni, 34–36; C, 0.50–0.60; Mn, 0.60–0.80	.....	.....	....	C, HR, P
235	Standard Alloy H R 6	Standard Alloy Co., Cleveland, Ohio	Fe; Cr, 24–26; Ni, 12–14; C, 0.40–0.50; Mn, 0.60–0.80	.....	0.8	....	B, C, HR, P, S, R, W
236	Symington Nirosta K A 2	Symington-Gould Corp., Rochester, N. Y.	Fe; Cr; Ni	7.93	.....	....	C
237	Thermalloy A	Electro-Alloys Co., Elyria, Ohio	Fe; Cr, 18–20; Ni, 62–65	.....	0.98	....	C
238	Thermalloy B	Electro-Alloys Co., Elyria, Ohio	Fe; Cr, 15–18; Ni, 38–40	.....	1.10	....	C

---

* "Essential nominal chemical composition" refers to the principal constituents and the approximate proportion present. When Fe is shown without percentage, it may be assumed that it represents the balance of the composition.

† B = bar; C = casting; CR = cold rolled; D = drawn; HR = hot rolled; P = plates; R = rods; S = sheets; T = tubes; W = wire.

Table 3. Materials of Construction—Base Metals. Manufacturers, Chemical Composition, Physical Properties, and Commercial Forms Available—(*Continued*)

No.	Material	Manufacturer	Essential nominal chemical composition,* per cent	Specific gravity	Mean coeff. therm. exp., 32–212°F. (multiply by 10^{-6})	Therm. conduct., C.G.S. units, room temp.	Forms available†
239	Thermalloy E	Electro-Alloys Co., Elyria, Ohio	Fe; Cr, 27–30; Ni, 8–10	.....	.....	.....	C
240	Thermalloy 72	Electro-Alloys Co., Elyria, Ohio	Fe; Cr, 12–15; Ni, 60–62	.....	0.98	.....	C
241	Timken 16-13-3	Timken Roller Bearing Co., Canton, Ohio	Fe; Cr, 15.5–17; Ni, 12.5–14.5; C, 0.13 max.; Mo, 2.5–3.25; Mn, 1.5 max.	.....	.....	.....	B, HR, D R, T, W
242	Timken 2512	Timken Roller Bearing Co., Canton, Ohio	Fe; Ni, 4.75-5.25; C, 0.20 max.	.....	.....	.....	T
243	Timken 2% Cr-0.5% Mo	Timken Roller Bearing Co., Canton, Ohio	Fe; Cr, 1.75–2.25; C, 0.15 max.; Mo, 0.45–0.65; Mn, 0.30–0.60	.....	.....	.....	B, HR, T
244	Timken 5% Cr-Mo	Timken Roller Bearing Co., Canton, Ohio	Fe; Cr, 4–6; C, 0.15 max.; Mn, 0.50 max.; Mo, 0.45–0.65	.....	.....	.....	B, HR, R, T
245	Timken DM	Timken Roller Bearing Co., Canton, Ohio	Fe; Cr, 1–1.5; C, 0.15 max.; Mn, 0.3–0.6; Mo, 0.45–0.65	.....	.....	.....	B, HR, R, T
246	Timken Sicromo 5S	Timken Roller Bearing Co., Canton, Ohio	Fe; Cr, 4–6; C, 0.15 max.; Si, 1–2; Mn, 0.5 max.; Mo, 0.45–0.65	.....	.....	.....	B, HR, R, T
247	Timken Sicromo 7	Timken Roller Bearing Co., Canton, Ohio	Fe; Cr, 6–8; C, 0.15 max.; Si, 0.5–1; Mn, 0.50 max.; Mo, 0.45–0.65	.....	.....	.....	B, HR, D R, T, W
248	Timken Sicromo 7M	Timken Roller Bearing Co., Canton, Ohio	Fe; Cr, 6–8; C, 0.15 max.; Mo, 0.9–1.1; Mn, 0.50 max.; Si, 0.5–1	.....	.....	.....	B, HR, D ,R, T, W
249	Timken Sicromo 9 M	Timken Roller Bearing Co., Canton, Ohio	Fe; Cr, 8–10; C, 0.15 max.; Mn, 0.50 max.; Mo, 0.9–1.1	.....	.....	.....	B, HR, D, R, T, W

250	*Tisco 41*	*Taylor Wharton Iron & Steel Co., High Bridge, N. J.*	*Fe; C, 0.5; Cr, 1; Mn, 0.7*	.....	.....	.....	*C*
251	Tisco 53	Taylor Wharton Iron & Steel Co., High Bridge, N. J.	Fe; Cr, 5; C, 0.25; Mn, 0.60; Mo, 0.50	.....	.....	.....	C
252	Tisco 80	Taylor Wharton Iron & Steel Co., High Bridge, N. J.	Fe; Ni, 2; Cr, 1.5; C, 0.45; Mn, 0.7; Mo, 0.35	.....	.....	.....	C
253	Tisco 102	Taylor Wharton Iron & Steel Co., High Bridge, N. J.	Fe; Cr, 18; Ni, 8; C, 0.07 max.; Mn, 1.0 max.; Si, 2 max.	.....	.....	.....	C
254	Tisco 108	Taylor Wharton Iron & Steel Co., High Bridge, N. J.	Fe; Cr, 25; Ni, 18; C, 0.20 max.; Mn, 1.0 max.; Si, 2 max.	.....	.....	.....	C
255	Tisco 130	Taylor Wharton Iron & Steel Co., High Bridge, N. J.	Fe, Cr, 28; C, 0.10 max.; Mn, 0.50; Si, 2.0	.....	.....	.....	C
256	Tisco 131	Taylor Wharton Iron & Steel Co., High Bridge, N. J.	Fe; Cr, 18; C, 0.10 max.; Mn, 0.50; Si, 2	.....	.....	.....	C
257	Tisco 160	Taylor Wharton Iron & Steel Co., High Bridge, N. J.	Fe; Cr, 1.5–4; Ni, 12–15; C, 3; Cu, 5–7; Mn, 1; Si, 1.5	.....	.....	.....	C
258	Tophet C	Wilber B. Driver Co., Newark, N. J.	Fe; Ni, 60; Cr, 15	8.20	0.76	.....	HR, D, R, W
259	Warman 5M	Warman Steel Casting Co., Huntington Pk., Calif.	Fe; Cr, 5–7; C as specified; Mo, 0.50–0.60	.....	.....	.....	C
260	Warman 13	Warman Steel Casting Co., Huntington Pk., Calif.	Fe; Cr, 11.5–14; C as specified	.....	.....	.....	C
261	Worthite	Worthington Pump & Machinery Corp., Harrison, N. J.	Fe; Cr, 20; Ni, 24; Cu, 1.75; C, 0.07; Si, 3.25, Mo, 3.0; Mn, 0.60	7.85	0.78	0.045	B, C, HR, R
262	X-7	General Alloys Co., So. Boston	Fe; Cr, 25; Ni, 12; C, 0.35; Mn, 0.75	7.73	0.87	.....	B, C, HR, P, S, R, W
263	X-ITE	General Alloys Co., So. Boston	Fe; Cr, 18; Ni, 38; C, 0.60; Mn, 1.0	8.05	0.64	0.031	B, C, HR, P, S, R, W
264	X-ITE-B	General Alloys Co., So. Boston	Fe; Cr, 15; Ni, 35; C, 0.60; Mn, 1.0	7.99	0.66	0.032	B, C, HR, P, S, R, W

---

* "Essential nominal chemical composition" refers to the principal constituents and the approximate proportion present. When Fe is shown without percentage, it may be assumed that it represents the balance of the composition.

† B = bar; C = casting; CR = cold rolled; D = drawn; HR = hot rolled; P = plates; R = rods; S = sheets; T = tubes; W = wire.

Table 3. Materials of Construction—Base Metals. Manufacturers, Chemical Composition, Physical Properties, and Commercial Forms Available—(*Continued*)

No.	Material	Manufacturer	Essential nominal chemical composition,* per cent	Specific gravity	Mean coeff. therm. exp., 32–212°F. (multiply by 10^{-5})	Therm. conduct., C.G.S. units, room temp.	Forms available†
			Non-ferrous Alloys				
265	Admiralty	Generally available[4]	Cu, 70; Sn, 1; Zn, 29	8.53	1.03	0.26	CR, D, P, S, T
266	Adnic	Scovill Manufacturing Co., Waterbury, Conn.	Cu, 70; Ni, 29; Sn, 1	8.88	0.91	0.07	CR, D, R, T, W
267	Advance	Driver Harris Co., Harrison, N. J.	Cu, 55; Ni, 45	8.9	0.82	0.05	B, CR, HR, P, S, R, W
268	Alcoa 2 S	Aluminum Company of America, Pittsburgh	Al, 99 min.	2.71	1.33	....	B, C, CR, HR, P, S, R, T, W
269	Alcoa 3 S	Aluminum Company of America, Pittsburgh	Al; Mn, 1.2	2.73	1.33	....	B, CR, HR, D, P, S, R, T, W
270	Alcoa 52 S	Aluminum Company of America, Pittsburgh	Al; Cr, 0.25; Mg, 2.5	2.67	....	....	B, CR, HR, D, P, S, R, T, W
271	Alcoa 53 S	Aluminum Company of America, Pittsburgh	Al; Mg, 1.3; Si, 0.7; Cr, 0.25	2.69	1.30	....	B, CR, HR, D, P, S, R, T, W
272	Alcoa Alclad (72 S) 3 S	Aluminum Company of America, Pittsburgh	(See footnote 5)	2.75	....	....	D, P, S, T
273	Alcoa 43	Aluminum Company of America, Pittsburgh	Al; Si, 5	2.66	1.22	....	Castings
274	Alcoa B 214	Aluminum Company of America, Pittsburgh	Al; Mg, 3.8; Si, 1.8	2.57	1.33	....	Castings
275	Alcoa 220-T 4	Aluminum Company of America, Pittsburgh	Al; Mg, 10	2.56	1.36	....	Castings
276	Alcoa 356	Aluminum Company of America, Pittsburgh	Al; Si, 7; Mg, 0.3	2.65	1.19	....	Castings
277	Alcoa 406	Aluminum Company of America, Pittsburgh	Al; Mn, 2	....	....	....	Castings
228	Alcumite	Duriron Co., Dayton, Ohio	Cu, 89.75; Al, 9.0; Fe, 1.25	7.75	0.93	....	Castings
279	40 Alloy	National Smelting Co., Cleveland	Al; Cu, 4	2.77	1.27	0.35	Castings

280	*Alray A*	*Alloy Metal Wire Co., Prospect Park, Pa.*	*Ni, 80; Cr, 20*	*8.4*	*0.73*	....	*CR, D, R, W*
281	Aluminum 99 +	National Smelting Co., Cleveland	Al, 99+	2.72	1.33	0.52	Castings
282	Aluminum 98–99	National Smelting Co., Cleveland	Al, 98–99	....	....	0.50	Castings
283	Aluminum Bronze	American Brass Co., Waterbury, Conn.	Cu, 82–95; Al, 5–9.5; Fe; Mn; Ni; Sn	7.58–8.17	....	0.09–0.19	C, CR, HR, D, P, S, R, T, W
284	Aluminum Bronze	Hills McCanna Co., Chicago	Cu, 85–89; Al, 9–10; Fe, 1–4; Mn, 0.5	....	0.95	....	Castings
285	Aluminum Bronze 5%	Revere Copper & Brass, New York, N. Y.	Cu, 95; Al, 5	8.1	....	0.198	T
286	Aluminum Brass	Generally available[4]	Cu, 76; Zn, 21.5–22; Al, 2–2.5	8.31–8.33	0.97–1.02	0.225–0.24	CR, D, T
287	Ambrac 850	American Brass Co., Waterbury, Conn.	Cu, 75; Ni, 20; Zn, 5	8.86	0.85	0.092	B, C, CR, HR, D, P, S, R, T, W
288	Ampco 18	Ampco Metal, Milwaukee, Wis.	Cu, 84.6; Al, 11.3; Fe, 3.7; Ni, 0.2	7.59	1.24	0.118	B, C
289	Ampcoloy	Ampco Metal, Milwaukee, Wis.	Cu; Al, 10–11; Fe, 1.25 max.	7.48	0.92	....	Castings
290	Antimonial Admiralty	Chase Brass & Copper Co., Waterbury, Conn.	Cu, 72; Zn, 27; Sb, 0.07; Sn, 1	8.52	1.05	0.26	S, T, W
291	Antimonial Lead	National Lead Co., New York	Pb, 94; Sb, 6	10.88	1.52	0.068	D, C, CR, S, R, T, W
292	Antimonial Lead	Northwest Lead Co., Seattle, Wash.	Pb, 93.02–93.06; Sb, 6; Cu, 0.04–0.08	10.8	1.52	0.068	B, C, S, T, W
293	Arsenical Copper	Generally available[4]	Cu; As, 0.15–0.75	8.91	0.98	....	D, T
294	Asarco Acid Lead	American Smelting & Refining Co., New York, N. Y.	Pb; Cu, 0.06; Bi, 0.02	11.33	1.6	....	C, CR, S, R, T, W
295	Beryllium Copper	American Brass Co., Waterbury, Conn.	Cu, 97.40; Be, 2.25; Ni, 0.35	8.22	....	0.25	B, C, CR, HR, D, P, S, R, W
296	Beryllium Copper	Riverside Metal Co., Riverside, N. J.	Cu, 97.90; Be, 2.10	8.22	1.65	....	B, CR, D, P, S, R, W
297	Cadmium-Copper	Phelps Dodge Copper Products Corp., New York	Cu, 99; Cd, 1	8.94	....	0.824	B, CR, HR, D, R, W
298	Chemical Lead	National Lead Co., New York	Pb, 99.93; Cu, 0.06	11.36	1.63	0.083	B, C, CR, S, R, T, W

---

* "Essential nominal chemical composition" refers to the principal constituents and the approximate proportion present. When Fe is shown without percentage, it may be assumed that it represents the balance of the composition.

† B = bar; C = casting; CR = cold rolled; D = drawn; HR = hot rolled; P = plates; R = rods; S = sheets; T = tubes; W = wire.

[4] Copper and copper alloys are available from such companies as the following: American Brass Co., Waterbury, Conn.; Bridgeport Brass Co., Bridgeport, Conn.; Bristol Brass Corp., Bristol, Conn.; Chase Brass & Copper Co., Waterbury, Conn.; Mueller Brass Co., Huron, Mich., New England Brass Co., Taunton, Mass.; Phelps Dodge Copper Products Corp., New York; Revere Copper & Brass, Inc., New York; Riverside Metal Co., Riverside, N. J.; Scovill Manufacturing Co., Waterbury, Conn.; Seymour Manufacturing Co., Seymour, Conn.; Wolverine Tube Co., Detroit.

[5] Surface layer of aluminum or aluminum alloy which is anodic to the core and will therefore protect the latter electrolytically.

Table 3. Materials of Construction—Base Metals. Manufacturers, Chemical Composition, Physical Properties, and Commercial Forms Available—(*Continued*)

No.	Material	Manufacturer	Essential nominal chemical composition,* per cent	Specific gravity	Mean coeff. therm. exp., 32–212°F. (multiply by 10^{-5})	Therm. conduct., C.G.S. units, room temp.	Forms available†
299	Chemical Lead	Northwest Lead Co., Seattle, Wash.	Pb, 99.92–99.96; Cu, 0.04–0.08	11.2	1.63	0.083	B, C, S, T, W
300	Chromel A	Hoskins Manufacturing Co., Detroit	Ni, 78; Cr, 20; Mn, 2 max., C, 0.06	8.4	....	....	B, CR, HR, D, R, W
301	Colmonoy 6	Wall-Colmonoy Corp., Detroit	Ni, 63–73; Cr, 15–22; B, 3–5	7.8	0.87	....	B, C, R
302	Chromium Copper	American Brass Co., Waterbury, Conn.	Cu, 99.05; Cr, 0.85; Si, 0.10	8.89	....	0.77	B, C, CR, HR, D, P, S, R, W
303	Commercial Bronze, F. C.	Chase Brass & Copper Co., Waterbury, Conn.	Cu, 89; Zn, 9; Pb, 2	8.84	0.98	0.42	R
304	20% Cupro-Nickel	Generally available[4]	Cu, 80; Ni, 20	8.85	0.91	0.08	P, T
305	30% Cupro-Nickel	Generally available[4]	Cu, 70; Ni, 30	8.94	0.83	0.07	B, C, CR, HR, D, P, S, R, T, W
306	Deoxidized Copper	Generally available[4]	Cu, 99.9+; P, 0.01–0.03	8.89–8.94	0.93–0.98	0.81–0.92	B, C, CR, HR, D, P, S, R, T, W
307	Durco D-10	Duriron Co., Dayton, Ohio	Ni, 57; Cr, 23; Cu, 8; Mo, 4; W, 2; Mn, 1	8.2	0.78	....	Castings
308	Everdur-1000	American Brass Co., Waterbury, Conn.	Cu, 94.9; Si, 4; Mn, 1.1	8.23	0.94	0.066	Castings
309	Everdur-1015	American Brass Co., Waterbury, Conn.	Cu, 98.25; Si, 1.5; Mn, 0.25	8.75	0.94	0.129	B, CR, HR, D, P, S, R, T, W
310	Gold	Baker & Co., Newark, N. J.	Au	19.3	0.80	0.707	B, C, CR, HR, D, P, S, R, T, W
311	Hastelloy A	Haynes Stellite Co., Kokomo, Ind.	Ni; Mo; Fe	8.80	0.61	0.04	B, C, HR, P, S, R, T, W
312	Hastelloy B	Haynes Stellite Co., Kokomo, Ind.	Ni; Mo; Fe	9.24	0.56	0.027	B, C, HR, P, S, R, T, W
313	Hastelloy C	Haynes Stellite Co., Kokomo, Ind.	Ni; Mo; Fe; Cr	8.94	0.63	0.03	B, C, P, S, T
314	Hastelloy D	Haynes Stellite Co., Kokomo, Ind.	Ni; Si; Cu	7.80	0.61	0.05	B, C
315	Hardware Bronze	Scovill Manufacturing Co., Waterbury, Conn.	Cu, 89; Zn, 8; Pb, 2; Ni, 1	8.88	1.02	0.43	O, R, W
316	Herculoy A	Revere Copper & Brass, New York	Cu, 96; Si, 3; Mn, 1; Sn, 0.5	8.5	....	0.086	B, CR, HR, D, P, S, R, W

317	*Herculoy B*	*Revere Copper & Brass, New York*	*Cu, 98; Si, 1.75; Sn, 0.25*	*8.85*	*........*	*0.117*	*B, CR, HR, D, P, S, R, T, W*
318	High Brass	Generally available[4]	Cu, 66; Zn, 34	8.46	1.05	0.29	
319	High Silicon Bronze	Phelps Dodge Copper Products Corp., New York	Cu, 97.3; Sn, 1.65; Si, 1.05	8.80	0.93	....	B, CR, HR, D, R, W
320	Illium G	Burgess-Parr Co., Freeport, Ill.	Ni, 55-60; Cr, 18-24; Cu, 5-8; Mo, 5-8; Fe, 5-8; Mn, 1.5; W; Si	8.21	....	0.029	B, C
321	Illium R	Burgess-Parr Co., Freeport, Ill.	Ni, 58-64; Cr, 18-24; Mo, 5-8; Fe, 5-8; Cu, 2-6; Mn, 0.5-1.75; W; Si	8.3	....	0.031	B, CR, R, T
322	Inconel	International Nickel Co., New York	Ni, 79.5; Cr, 13; Fe, 6.5; Mn, 0.25; Si, 0.25; C, 0.08; Cu, 0.20	8.51	0.64	0.036	B, C, CR, HR, D, P, S, R, T, W
323	Inconel Clad Steel	Lukens Steel Co., Coatesville, Pa.	Ni, 79.5; Cr, 13; Fe, 6.5; Mn, 0.25; Si, 0.25; C, 0.08; Cu, 0.20	....	....	....	HR, P, S
324	Indium Metal	American Smelting & Refining Co., New York	In	7.31	....	....	B, C, CR, S, R, T, W
325	Iridio Platinum	Baker & Co., Newark, N. J.	Pt; Ir, 5-30	21.4-21.5	....	....	B, C, CR, HR, D, P, S, R, W
326	10% Iridium Platinum	J. Bishop & Co., Malvern, Pa.	Pt, 90; Ir, 10	21.59	....	....	B, C, CR, HR, D, P, S, R, T, W
327	30% Iridium Platinum	J. Bishop & Co., Malvern, Pa.	Pt, 70; Ir, 30	21.78	....	....	B, C, CR, HR, D, P, S, R, T, W
328	40 M Alloy	National Smelting Co., Cleveland	Al; Mg, 4	2.63	1.33	0.32	Casting
329	Molybdenum	P. R. Mallory & Co., Indianapolis	Mo, 99.8-99.9	10.2	0.49	0.346	S, R, W
330	Monel	International Nickel Co., New York	Ni, 67; Cu, 30; Fe, 1.4; Mn, 1; C, 0.15; Si, 0.1	8.84	0.78	0.062	B, C, CR, HR, D, P, S, R, T, W
331	Monel Clad Steel	Lukens Steel Co., Coatesville, Pa.	Ni, 67; Cu, 30; Fe, 1.4; Mn, 1; C, 0.15; Si, 0.1	....	....	....	HR, P, S
332	Muntz	Generally available[4]	Cu, 60; Zn, 40	8.39	1.16	0.30	S, R, T, W
333	Naval Brass	Generally available[4]	Cu, 60; Zn, 39.25; Sn, 0.75	8.39	1.19	0.28	D, R, T, W
334	Nichrome V	Driver Harris Co., Harrison, N. J.	Ni, 80; Cr, 20	8.41	0.73	0.036	B, CR, HR, D, P, S, R, T, W
335	Nickel	International Nickel Co., New York	Ni, 99.4; C, 0.1; Cu, 0.1; Fe, 0.15; Mn, 0.2; Si, 0.05	8.89	0.72	0.145	B, C, CR, HR, D, P, S, R, T, W

---

[*] "Essential nominal chemical composition" refers to the principal constituents and the approximate proportion present. When Fe is shown without percentage, it may be assumed that it represents the balance of the composition.

[†] B = bar; C = casting; CR = cold rolled; D = drawn; HR = hot rolled; P = plates; R = rods; S = sheets; T = tubes; W = wire.

[4] Copper and copper alloys are available from such companies as the following: American Brass Co., Waterbury, Conn.; Bridgeport Brass Co., Bridgeport, Conn.; Bristol Brass Corp., Bristol, Conn.; Chase Brass & Copper Co., Waterbury, Conn.; Mueller Brass Co., Huron, Mich., New England Brass Co., Taunton, Mass.; Phelps Dodge Copper Products Corp., New York; Revere Copper & Brass, Inc., New York; Riverside Metal Co., Riverside, N. J.; Scovill Manufacturing Co., Waterbury, Conn.; Seymour Manufacturing Co., Seymour, Conn.; Wolverine Tube Co., Detroit.

Table 3. Materials of Construction—Base Metals. Manufacturers, Chemical Composition, Physical Properties, and Commercial Forms Available—(*Continued*)

No.	Material	Manufacturer	Essential nominal chemical composition,* per cent	Specific gravity	Mean coeff. therm. exp., 32–212°F. (multiply by 10^{-5})	Therm. conduct., C.G.S. units, room temp.	Forms available†
336	Nickel Clad Steel	Lukens Steel Co., Coatesville, Pa.	Ni, 99.4; C, 0.1; Cu, 0.1; Fe, 0.15; Mn, 0.2; Si, 0.05	....	....	....	HR, P, S
337	Nickel Silver 18%	Generally available[4]	Cu, 65; Ni, 18; Zn, 17	8.75	0.94	0.080	B, C, CR, D, P, S, R, T, W
338	Nickel Silver 18%	Generally available[4]	Cu, 55; Zn, 27; Ni, 18	8.69	....	....	B, C, CR, D, P, S, R, W
339	Nirex	Driver Harris Co., Harrison, N. J.	Ni, 80; Cr, 14; Fe, 6	8.55	0.70	0.034	B, CR, HR, P, S, R, W
340	O F H C	Scomet Engineering Co., New York, N. Y.	Cu, 99.985	8.93	0.922	0.923	C, CR, HR, D, P, S, R, T, W, B
341	Olympic Bronze	Chase Brass & Copper Co., Waterbury, Conn.	Cu, 96; Si, 3; Zn, 1	8.58	0.94	0.08	B, C, CR, HR, D, P, S, R, T, W
342	P D C P Copper	Phelps Dodge Copper Products Corp., New York	Cu, 99.985	8.94	0.933	0.921	B, C, DR, HR, D, R, T, W
343	Palladium	Baker & Co., Newark, N. J.	Pd, 100	12.0	1.16	0.166	B, C, CR, HR, D, P, S, R, T, W
344	Phosphor Bronze A	Generally available[4]	Cu, 94.8–95.5; Sn, 4.3–5.0; P	8.86	0.95	0.195	B, CR, D, P, S, R, T, W
345	Phosphor Bronze D	Generally available[4]	Cu, 89.5–96; Sn, 10–10.5; P	8.77	0.98	0.121	B, CR, D, P, S, R, T, W
346	Phosphor Bronze F. C.	Generally available[4]	Cu, 88; Zn, 4; Sn, 4; Pb, 4	8.86	....	0.133	B, CR, D, P, S, R
347	Pioneer	Pioneer Alloy Products Co., Cleveland	Ni, 30–35; Cr, 20–25; Mo, 3–5; C, 0.10–0.20	7.83			
348	Platinum	Baker & Co., Newark, N. J.	Pt, 100	21.4	0.88	0.166	B, C, CR, HR, D, P, S, R, T, W
349	Platinum	J. Bishop & Co., Malvern, Pa.	Pt, 99.9	21.4	0.89	0.166	B, C, CR, HR, D, P, S, R, T, W
350	Red X 10	National Smelting Co., Cleveland	Al; Si, 10; Cu, 1.5; Mn, 0.6; Mg, 0.5	2.70	1.13	0.30	Casting
351	Red Brass	Generally available[4]	Cu, 85; Zn, 15	8.64–8.75	0.98–1.04	0.37–0.39	B, CR, D, P, S, R, T, W
352	Rhodio Platinum	Baker & Co., Newark, N. J.	Pt; Rh, 5–40	20	....	....	B, C, CR, HR, D, P, S, R, T, W
353	Roofloy	Amer. Smelting & Refining Co., New York	Pb; Sn, 0.25; Ca, 0.02; Mg, 0.02; Bi, 0.02	11.3	1.6	....	CR, R, T
354	70 S Alloy	National Smelting Co., Cleveland	Al; Si, 7.5	2.65	1.2	0.40	Casting
355	Silver, Fine[6]	Baker & Co., Newark, N. J.	Ag, 99.9+	10.5	10.5	0.97	
356	Silver, Fine[7]	Handy & Harman, Bridgeport, Conn.	Ag, 99.9+	10.5	10.6	1.0	B, C, CR, D, P, S, R, T, W

357	*Stellite 1*	*Haynes Stellite Co., Kokomo, Ind.*	*Co; Cr; W*	*8.59*	*0.80*	....	*C, weld rod*
358	Stellite 6	Haynes Stellite Co., Kokomo, Ind.	Co; Cr; W	8.38	0.94	0.035	Weld rod
359	Super Nickel Clad Steel	Lukens Steel Co., Coatesville, Pa.	....	....	....	....	HR, P, S
360	Tantalum	Fansteel Metallurgical Corp., N. Chicago	Ta, 99.9+	....	0.357	0.130	D, S, R, T, W
361	Tellurium Lead	National Lead Co., New York	Pb, 99.88; Cu, 0.06; Te, 0.045	11.36	1.63	0.083	B, C, CR, S, R, T, W
362	Tellurium Lead	Northwest Lead Co., Seattle, Wash.	Pb, 99.88; Cu, 0.06; Te, 0.045	11.36	1.63	0.083	B, C, CR, S, R, T, W
363	Tophet A	Wilbur B. Driver Co., Newark, N. J.	Ni, 80; Cr, 20	8.40	0.73	....	HR, D, R, W
364	Tungsten	P. R. Mallory & Co., Indianapolis	W, 99.8–99.9	19.3	0.43	0.476	S, R, W
365	Wolverine Commercial Bronze	Wolverine Tube Co., Detroit	Cu, 90; Zn, 10	8.80	1.01	0.45	T
366	Y Alloy	National Smelting Co., Cleveland	Al; Cu, 4; Ni, 1.5; Mg, 1.5	2.73	1.3	0.35	Casting

---

[*] "Essential nominal chemical composition" refers to the principal constituents and the approximate proportion present. When Fe is shown without percentage, it may be assumed that it represents the balance of the composition.

[†] B = bar; C = casting; CR = cold rolled; D = drawn; HR = hot rolled; P = plates; R = rods; S = sheets; T = tubes; W = wire.

[4] Copper and copper alloys are available from such companies as the following: American Brass Co., Waterbury, Conn.; Bridgeport Brass Co., Bridgeport, Conn.; Bristol Brass Corp., Bristol, Conn.; Chase Brass & Copper Co., Waterbury, Conn.; Mueller Brass Co., Huron, Mich., New England Brass Co., Taunton, Mass.; Phelps Dodge Copper Products Corp., New York; Revere Copper & Brass, Inc., New York; Riverside Metal Co., Riverside, N. J.; Scovill Manufacturing Co., Waterbury, Conn.; Seymour Manufacturing Co., Seymour, Conn.; Wolverine Tube Co., Detroit.

[6] Coin, sterling, and clad are also available.

[7] Coin and sterling are also available.

Table 4. Materials of Construction—Base Metals. Physical Properties and Methods of Fabrication

No.	Material	Form for which tensile properties are recorded	Tensile strength, 1000 lb. per sq. in.	Yield point, 1000 lb. per sq. in.	Elongation, % in 2 in.	Reduction of area, %	Elastic modulus, lb. per sq. in. (multiply by 10^6)	Brinell hardness	Machining qualities	Methods of fabrication*
			Ferrous Alloys							
1	Abrasion Resisting	Plates	110	60	20	40	......	225	Fair	R
2	Allegheny Metal 18-8	Casting	70–87	36–46	50–75	45–60	28–30	....	....	R, W
3	Allegheny Metal 18-8 S	Casting	70–87	36–46	50–75	45–60	28–30	....	....	R, W
4	Allegheny Metal 18-8 S	Casting	70–85	35–46	50–75	45–65	28–30	....	....	R, W
5	Allegheny Metal 20-10 S	Casting	68–80	40–60	50–75	45–65	28–30	....	....	R, W
6	Allegheny Metal 25-12	Casting	75–90	40–60	25–40	25–45	28–30	....	....	R, W
7	Allegheny Metal 25-12 S	Casting	75–90	40–55	30–50	30–50	28–30	....	....	R, W
8	Allegheny Metal 25-20	Casting	60–75	40–55	40–60	40–60	28–30	....	....	R, W
9	Allegheny Metal 25-20 S	Casting	60–75	40–55	40–60	40–60	28–30	....	....	R, W
10	Allegheny Metal 20-25 S	Casting	60–75	40–55	40–60	40–60	28–30	....	....	R, W
11	Allegheny Metal 20-25 SM	Casting	60–75	50–70	40–60	40–60	28–30	....	....	R, W
12	Allegheny Metal 29-9	Casting	80–95	45–70	20–30	20–35	28–30	....	....	R, W
13	Allegheny Metal 29-9 S	Casting	75–95	36–46	20–40	20–45	28–30	....	....	R, W
14	Allegheny Metal 18-8 M	Casting	73–85	35–46	45–60	45–60	28–30	....	....	R, W
15	Allegheny Metal 18-8 SM	Casting	73–85	35–45	45–60	45–60	28–30	....	....	R, W
16	Allegheny Metal 19-10 M	Casting	70–80	35–45	45–60	45–60	28–30	....	....	R, W
17	Allegheny Metal 19-10 SM	Casting	70–80	40–60	45–60	45–65	28–30	....	....	R, W
18	Allegheny Metal 28-4	Casting	70–90	35–50	10–25	10–25	28–30	....	....	R, W
19	Allegheny Metal 15-35	Casting	50–75	40–56	40–60	40–60	28–30	....	....	R, W
20	Allegheny Metal 18-8 C	Casting	78–90	40–55	35–50	40–50	28–30	....	....	R, W
21	Allegheny Metal 18-11 C	Casting	75–90		35–50	40–50	28–30	....	....	R, W
22	Allegheny Metal 12 Cr.	Casting	Depends on heat treatment		....	.....	28–30	....	....	R, W
	Allegheny Metal 17 Cr.	Casting	Depends on heat treatment		....	.....	28–30	....	....	R, W
24	Alray C	Annealed	95–175	35	35–40	60	.....	....	....	W, B
25	Alray D	Annealed	75–160	....	35–40	60	.....	....	....	R, W, B
26	Amsco F-1	....	50–70	40–50	4–8	4–8	26	150–180	Fair	W
27	Amsco F-3	As cast	40–60	30–45	0–2	.....	.....	170	Fair	W

28	Amsco F-5	*As cast*	*50–70*	*35–45*	*24–40*	.....	.....	*180–200*	*Fair*	*W*
29	Amsco F-6	.....	.....	.....	.....	.....	.....	.....	.....	W
30	Amsco F-8	As cast	70–90	25–45	40–70	45–75	29	150–180	Good	W
31	Amsco F-10	As cast	80–100	40–55	30–50	30–50	23.5	160–190	Fair	W
32	Amsco Manganese Steel	Water quenched from 1850°F.	120–140	50–60	25–35	25–35	.....	185–200	.....	R, W
33	Amsco Ni-Mn Welding Rod									
34	B & W 5150	.....	.....	.....	.....	.....	.....	450–750		
35	Bethlehem 235	.....	.....	.....	.....	.....	.....		Fair	
36	Bethlehem 300	.....	.....	.....	.....	.....	.....	.....	Fair	
37	Buflokast Gray Iron	Bar	30–50	.....	.....	.....	.....	150–250	Good	
38	Calmar 18-8	Annealed casting	80–90	38–45	55–65	.....	.....	.....	Fair	
39	Calmar 18-8 Cb	Annealed casting	80–85	40–45	40–45	.....	.....	.....	Fair	
40	Calmar 18-8 M	Annealed casting	85–95	40–50	25–30	.....	.....	150	Fair	
41	Caloxo 18	Cast, heat treated	65–75	45–52	20–25	35–40	.....	.....	Fair	
42	Caloxo 15-35	.....	55–65	35–40	10–15	.....	.....	.....	Good	
43	Caloxo 25-12	Annealed casting	75–85	40–45	30–35					
44	Caloxo 25-20	Annealed casting	72–80	35–42	35–40	.....	.....	135–145	Fair	
45	Caloxo 28-10	Annealed casting	70–80	40–45	20–30					
46	Carbon-Molybdenum Steel	Plates	65–87	.....	.....	.....	.....	145	Good	F, R, W
47	Causul Metal	.....	32	.....	.....	.....	.....	140–180		
48	Chromax	Casting	60	40	2	3	.....	.....	Good	
49	Chromax	Wrought	70–150	30–40	30	.....	.....	.....	Good	DD, F, R, W, B
50	Chromel C	.....	105	60	25–35	55	31	.....	Good	F, R, W, B
51	4-6 Chrome	Drawn	66	39	45	.....	30	143	Good	F, W
52	4-6 Chrome	Annealed	67	26	39	81	30	128	Good	F, W
53	Chrome-Copper-Nickel Steel	.....	65	35	30	50	.....	135	Fair	F, R, W
54	Chrome-Manganese Steel	Plate	100	55	25.2	45	.....	200	Fair	F, R, W
55	Cimet	Casting	65	.....	0	.....	.....	.....	Good	
56	Circle L 1	Cast, normalized and drawn	90	60	25	50	.....	180	Good	W
57	Circle L 2	Cast, oil quenched and drawn	125	95	15	35	.....	250	Fair	W
58	Circle L 3	Cast, oil quenched and drawn	160	130	10	15	.....	350	Fair	W
59	Circle L 5	Cast, normalized and drawn	100	70	24	50	.....	190	Fair	W
60	Circle L 10	Cast, normalized and drawn	100	75	18	30	.....	215	Good	W
61	Circle L 11	Cast, normalized and drawn	100	75	7.5	8.5	.....	200	Fair	W
62	Circle L 12	Cast, normalized and drawn	85	55	22	40	.....	170	Good	W
63	Circle L 13	Cast, oil quenched and drawn	200	150	5	10	.....	500 max.	Good	W
64	Circle L 19	Cast, normalized and drawn	70	40	30	50	.....	150	Good	W
65	Circle L 21	Cast, water quenched	70	32	45	50	.....	150	Fair	W
66	Circle L 22	Cast, water quenched	70	32	45	50	.....	150	Fair	W
67	Circle L 22 XM	Cast, water quenched	75	35	45	50	.....	160	Fair	W
68	Circle L 23	Cast, water quenched	75	35	40	45	.....	160	Fair	W

Copyright by *Chem. & Met. Eng.*, September, 1940.
* B = brazing; DD = deep drawing; F = flanging; R = riveting; W = welding.

Table 4. Materials of Construction—Base Metals. Physical Properties and Methods of Fabrication—(*Continued*)

No.	Material	Form for which tensile properties are recorded	Tensile strength, 1000 lb. per sq. in.	Yield point, 1000 lb. per sq. in.	Elongation, % in 2 in.	Reduction of area, %	Elastic modulus, lb. per sq. in. (multiply by 10^6)	Brinell hardness	Machining qualities	Methods of fabrication*
69	Circle L 30	Cast, water quenched	80	40	30	30	.....	170	Fair	W
70	Circle L 31	Cast, water quenched	90	50	25	25	.....	180	Fair	W
71	Circle L 34	Cast, water quenched	65	30	30	30	.....	135	Fair	W
72	Colonial 610	Annealed	85	54	29.9	54	28	187	Fair	W, B
73	Cooper 14	Annealed cast bar	75	50	20	30	27	180	Good	DD, F, R, W, B
74	Cooper 16	Annealed cast bar	75	50	20	30	29	180	Good	DD, F, R, W, B
75	Cooper 17	Annealed cast bar	80	40	47	50	28	155	Good	DD, F, R, W, B
76	Cooper 17 S	Annealed cast bar	80	40	47	50	28	150	Good	DD, F, R, W, B
77	Cooper 17 S Mo	Annealed cast bar	85	43	47	50	28	160	Good	DD, F, R, W, B
78	Cooper 17 S CB	Annealed cast bar	80	40	47	50	28	150	Good	DD, F, R, W
79	Cooper 18	Annealed cast bar	79	46.5	23	24	28	140	Good	DD, F, R, W, B
80	Cooper 19 A	As cast bar	70	35	7	8	28	190	Good	DD, F, R, W, B
81	Cooper 19	As cast bar	60	35	3	4	28	170	Good	DD, F, R, W, B
82	Cooper 20	As cast bar	65	40	7	8	28	165	Good	
83	Cooper 21	Cast bar	60	45	3	2	28	170	Good	
84	Cooper 21 A-B-C	Cast bar	70	35	10	12	28	150	Good	
85	Cooper 22	Cast bar	90	60	30	35	28	180	Good	DD, F, R, W, B
86	Cooper 22 P-M	Cast bar	90	60	30	35	28	180	Good	F, R, W
87	Cooper KNC 3	Cast bar	65	50	15	12	28	180	Good	W
88	Corrosiron	Casting	16	....	0	0	.....	275–325	Fair	
89	Crane 18-8 Mo	Annealed cast bar	85	48	53	55	28.6	135–150	Poor	W, B
90	Crane 5 Cr-Mo Cast Steel	Cast, heat-treated bar	110	80	18	30	.....	220–245	Fair	W
91	Croloy 2	Annealed seamless tube	60 min.	25 min.	30 -	.....	.....	163 max.	Fair	DD, F, R, W
92	Croloy 5	Seamless	60 min.	25 min.	30	.....	29	163 max.	Fair	DD, F, R, W
93	Croloy 9	Seamless tubing	60 min.	30 min.	30 min.	.....	29	170 max.	Fair	DD, F, R, W
94	Croloy 12	Seamless tubing	65–78	35–45	25–35	.....	28	135–160	Fair	DD, F, R, W
95	Croloy 16-13-3	Seamless tubing	75 min.	30 min.	35	.....	28	135–175	Fair	DD, F, R, W
96	Croloy 18	Seamless tubing	70–85	40–60	27–38	.....	29	140–180		
97	Croloy 18-8	Seamless tubing	75 min.	30 min.	50	.....	28	150	Fair	DD, F, R, W
98	Croloy 25-20	Seamless tubing	75 min.	35 min.	35	.....	29.7	115–175	Fair	DD, F, R, W
99	Croloy 27	Seamless tubing	75–90	45–60	15–30	.....	29	160–195	Fair	F, W
100	Popploy	Cast	35	....	....	.....	.....	....	Good	
101	Duraloy 18-8 S	Heat-treated casting	65	35	35	40	.....	180	Good	

102	Duraloy 18-8 S Mo	Heat-treated casting	65	35	35	40	.....	180	Good	
103	Duraloy 35-15	As cast	60	35	5	6	24	170	Good	
104	Duraloy 25-20 Mo	Heat-treated casting	65	45	20	20	.....	185	Fair	
105	Duraloy A	Casting	50	40	1	.....	.....	....	Good	
106	Duraloy B	Casting	90	55	5	5	.....	....	Good	
107	Duraloy N	As cast	80	50	20	15	.....	190	Good	
108	Duraloy NS Mo	Heat-treated casting	75	35	25	30	.....	195	Fair	
109	Durco D-18	Sand casting	90	55	5	5	.....	180	Fair	W
110	Durco KA 2S	Sand casting	80–90	30–40	40–55	40–55	.....	130–150	Fair	W
111	Durco KA2S Mo	Sand casting	85–95	45–55	40–50	45–55	.....	140–160	Fair	W
112	Durichlor	Bar	17	....	....	.....	.....	....	Poor	W
113	Durimet	Sand casting	65–75	30–40	35–50	40–55	.....	130–150	Fair	W
114	Duriron	Bar	16–18	....	....	.....	.....	....	Poor	W
115	Economet	Casting	60	30	7	10	.....	180	Good	
116	Economy Hardface-Self Hardening	....	....	....	....	.....	.....	450–550	Poor	W
117	Elverite A	....	....	....	....	.....	.....	500		
118	Elverite B	....	....	....	....	.....	.....	650–700		
119	Elverite C	....	....	....	....	.....	.....	650–700		
120	Empire 8	Annealed casting	105	60	20–25	25–30	.....	225	Fair	W
121	Empire 11	Annealed casting	105	65	15–20	30–35	.....	250	Fair	W
122	Empire 12	Annealed casting	85	50	15–25	30–35	.....	200	Fair	W
123	Empire 14	Annealed casting	95	55	20–25	45–50	.....	225	Fair	W
124	Empire 15	....	....	....	....	.....	.....	....	....	W
125	Empire 16	Annealed casting	75	40	50–55	60–65	.....	165	Fair	W
126	Empire 17	Annealed casting	55	35	1–2	3–4	.....	180	Fair	W
127	Empire 20	Annealed casting	75	40	35–40	40–50	.....	175	Fair	W
128	Empire 23	Annealed casting	60	30	30–35	35–40	.....	180	Fair	W
129	Esco 45	....	95–100	55	40	40	30	217	Good	
130	Fahrite C 7	....	84	40	25	24	25	160	Fair	
131	Fahrite C 7 M	....	85	42	27	26.5	25	179	Fair	
132	Fahrite C 8 A	....	80	36	50	60	28	160	Fair	
133	Fahrite C 8 A M	....	90	45	50	60	28	170	Fair	
134	Fahrite C 12	....	100	45	20	35	29	179–400	Fair	
135	Fahrite C 73	....	80	45	10	10	29	200	Fair	
136	Genesee 212	....	100	80	25	50	.....	200	Good	
137	Genesee 255	....	110	80	20–25	50–55	.....	180–200	Fair	
138	Genesee 304	....	....	....	....	.....	.....	....	Fair	
139	Genesee 305	....	....	....	....	.....	.....	....	Fair	
140	Genesee 412	Cast	80–140	....	25–35	30–40	.....	170–200	Poor	
141	Hard Facing Rod 217	....	....	....	....	.....	.....	600–700	Poor	
142	Hard Facing Welding Rod 459	....	....	....	....	.....	.....	500–600	Poor	

* B = brazing; DD = deep drawing; F = flanging; R = riveting; W = welding.

Table 4. Materials of Construction—Base Metals. Physical Properties and Methods of Fabrication— *(Continued)*

No.	Material	Form for which tensile properties are recorded	Tensile strength, 1000 lb. per sq. in.	Yield point, 1000 lb. per sq. in.	Elongation, % in 2 in.	Reduction of area, %	Elastic modulus, lb. per sq. in. (multiply by 10^6)	Brinell hardness	Machining qualities	Methods of fabrication[*]
143	Hoskins 502									
	(a) Hot rolled (b) Casting	Hot rolled Casting	110 70	....	....	.....	.....	....	Good	
144	Ing Aclad Stainless Clad Steel 304	Sheets and plates	60–70	40–50	30–50	50–60	29	145–160	Fair	DD, F, R, W, B
145	Ing Aclad Stainless Clad Steel 309	Sheets and plates	60–70	40–50	30–50	50–60	29	150–200	Poor	DD, F, R, W, B
146	Ing Aclad Stainless Clad Steel 317	Sheets and plates	60–70	40–50	....	.....	29	145–175	Poor	DD, F, R, W, B
147	Ing Aclad Stainless Clad Steel 347	Sheets and plates	60–70	40–50	30–50	50–60	29	145–160	Poor	DD, F, R, W, B
148	Manganese-Molybdenum Steel	Plates	95	60	25	45	.....	200	Fair	F, R, W
149	Mayari R	Hot rolled	70 min.	50 min.	22	50	28–30	150	Good	F, R, W, B
150	Midvaloy 13	Heat-treated wrought bars	80–200	35–170	7–30	11–70	28	140–400	....	DD, F, R, W, B
151	Midvaloy 18-8	Annealed wrought and cast bars	70–145	30–110	20–60	30–70	28	130–280	Fair	
152	Midvaloy 18-8 Se	Annealed wrought bars	85	35	45	56	28	170	Good	
153	Midvaloy 1700	Annealed wrought and cast bars	65–90	45–55	20–32	40–60	29	145–210	Fair	
154	Midvaloy 17-35	Castings	55–65	40–45	5–12	10–20	.....	150–170	Fair	
155	Midvaloy 25-12	Bars and castings	70–115	40–60	9–45	9–50	28	140–230	Fair	
156	Midvaloy 25-20	Wrought and cast bars	85–100	40–50	25–40	24–50	.....	170–200	Fair	
157	Midvaloy 2802	Castings	75–100	60–80	20–30	40–60	29	150–200	Fair	
158	Midvaloy ATV 3	Wrought bars	100–112	45–70	24–33	40–45	.....	185–238	Good	
159	Misco 18-8	Room temperature, quench annealed	70	30	40	46	26	197 max.	Fair	
160	Misco C	Room temperature, as cast	85	50	15–25	15–25	21.9	229 max.	Fair	
161	Misco HN-1	....	60	30	3	3	.....	217 max.	Good	
162	Misco HN-2	Room temperature, as cast	60	30	3	3	22	217 max.	Good	
163	Misco Metal[1]	Room temperature	60	35	4	4	23.9	217 max.	Fair	
164	Miscrome 1	Room temperature Normalized and tempered	95	60	17	20	30	241 max.	Good	
165	Miscrome 4	Air quenched and tempered, room temperature	80	50	20	30	.....	217 max.	Good	
166	Nichrome	Wrought	95–175	60	35	50	.....	....	Good	DD, F, R, W, B
167	Nichrome	Casting	65	....	2	3.5	.....	....	Good	
168	2% Nickel Steel	Plates	....	....	....	.....	.....	140	Poor	F, R, W
169	Ni-Hard	Casting	30–40	....	0	0	.....	575–750	Fair	....

170	Nilstain	Annealed wire	85–350	35–300	5–60	0–75	27	140–425	Good	W
171	Ni-Resist	Casting	25–40	.....	.....	.....	12–17	130–175	Good	
172	Pyrocast	Casting	60–70	60–70	0	0	.....	300–600	Good	
173	Q Alloy A	Casting	68	34	5	6	24.9	180	Good	W
174	Q Alloy B	Casting	66	33	4.5	5.5	23.6	176	Good	W
175	Q Alloy CN-1	Casting	85	43	50	60	26	190	Good	
176	Q Alloy CN-1-H	Casting	80	40	16	17	23	177	Fair	W
177	Q Alloy CN-1-Mo	Casting	85	43	50	60	26	190	Fair	W
178	Q Alloy CN-2	Casting	75	37	50	60	35	180	Fair	W
179	Q Alloy CN-2 Mo	Casting	75	37	50	60	35	180	Good	DD, F, W
180	O-4	Cast bar	100–110	65–80	18 min.	30 min.	30	190–230	Good	
181	O-12	Cast bar, heat treated	75–130	39–90	25–10	55–25	30	140–250	Good	
182	O-16	Heat-treated casting	70–80	39–43	20–15	30–20	28	140–175	Good	
183	O-18	Casting water quenched, 2000°F.	78–85	39–42	60–65	60–70	28.6	135–160	Fair	
184	O-24	Heat-treated casting	80–85	40–43	50–45	55–45	29	140–150	Fair	
185	O-30	Casting, stress relieved	45–70	35–60	2–3	2–3	29	170–190	Good	
186	Sivyer 5% Cr	Heat-treated casting	100–110	75–90	18–22	45–55	.....	229	Good	
187	Sivyer 60	Heat-treated casting	70–80	35–45	45–55	45–60	.....	156	Fair	
188	Sivyer 62	Casting	75–85	35–45	35–45	35–45	.....	163	Fair	
189	Sivyer 64	Casting	85–95	40–50	20–30	20–30	.....	197	Fair	
190	Sivyer 66	Heat-treated casting	75–150	45–125	8–25	15–50	.....	175–325	Good	
191	Sivyer 70	Casting	60–70	.....	2–7	2–8	.....	200–225	Fair	
192	Spang Chalfant 1	Annealed	66.4	40.6	36.5–40.3	74.1	30	131	Good	F, W
193	Spang Chalfant 2	Hot finished	62.4	42	36	67.5	30	126	Good	F, W
194	Spang Chalfant 3	Normalized and drawn	64.1	32.5	37	62.5	30	126	Good	F, W
195	Stainless Steel 301	Annealed sheet	90	45	55	.....	29	143–170	Fair	DD, F, R, W, B
196	Stainless Steel 302	Annealed sheet	90	45	55–60	65–55	29	130–162	Fair	DD, F, R, W, B
197	Stainless Steel 302 B	Annealed sheet	80–100	40–50	60–50	70–60	.....	150–180	Fair	
198	Stainless Steel 303	Annealed sheet	80–90	35–45	60–50	70–55	.....	130–150	Fair	W, B
199	Stainless Steel 304	Annealed bar, plate	80–95	35–45	55–60	55–65	29	135–185	Good	DD, F, R, W
200	Stainless Steel 308	Annealed bar	80–90	30–40	55–65	60–70	29	135–185	Poor	DD, F, R, W
201	Stainless Steel 309	Annealed sheet	95	45	45	65–50	29	170	Fair	DD, F, R, W, B
202	Stainless Steel 309 S	Annealed	85	35	40	.....	29	150		
203	Stainless Steel 310	Annealed sheet	90–110	40–60	55–45	60–50	.....	146–210	Fair	DD, F, R, W
204	Stainless Steel 311	Annealed	90–110	45–50	30–40	35–45	29	175	Good	DD, W, R, F
205	Stainless Steel 312									
206	Stainless Steel 315	As rolled	101	56	40	62	28.5	196	Fair	W, B
207	Stainless Steel 316	Annealed sheet	85	40	45–55	75–60	28	135–185	Fair	DD, F, R, W, B
208	Stainless Steel 317	Annealed sheet	85	40	40–50	.....	28	150–200	Fair	DD, F, R, W, B
209	Stainless Steel 321	Annealed sheet	80–90	35–45	60–55	65–55	.....	135–185	Fair	DD, F, R, W
210	Stainless Steel 325	Annealed	85–95	45–55	30–40	45–55	28–30	160–190	....	W, R
211	Stainless Steel 327									

---

[*] B = brazing; DD = deep drawing; F = flanging; R = riveting; W = welding.

[1] Composition given is for castings.

Table 4. Materials of Construction—Base Metals. Physical Properties and Methods of Fabrication— *(Continued)*

No.	Material	Form for which tensile properties are recorded	Tensile strength, 1000 lb. per sq. in.	Yield point, 1000 lb. per sq. in.	Elongation, % in 2 in.	Reduction of area, %	Elastic modulus, lb. per sq. in. (multiply by 10^6)	Brinell hardness	Machining qualities	Methods of fabrication*
212	Stainless Steel 329	Annealed	100	75	20	.....	29	....	Good	DD, F, R, W, B
213	Stainless Steel 330	Rolled bars	60–110	40-50	40	50	.....	190–220	....	F, R, B
214	Stainless Steel 347	Annealed sheet	85	40	45–55	65–55	29	140–180	Fair	DD, F, R, W, B
215	Stainless Steel 403	Annealed	65–85	35–45	25–35	60–65	28	135–165	Fair	DD, F, R, W
216	Stainless Steel 405	Annealed	75	48	31	65	.....	143		
217	Stainless Steel 406									
218	Stainless Steel 410	Annealed sheet	65	35	30	65–60	29	135–160	Fair	DD, F, R, W, B
219	Stainless Steel 414	Quenched in oil from 1750°F.	128	113	20	53	28	269	....	DD, F, W, B
220	Stainless Steel 416	Annealed sheet	70–85	40–60	35–25	65–60	.....	145–185	Good	Welding, forging
221	Stainless Steel 418	Annealed	87	64	29	64				
222	Stainless Steel 420									
	(*a*) Annealed	Annealed	105	55	23	55	29.5	207	Fair	R, W, B, DD, F
	(*b*) Tempered	Tempered	225	185	8	20	.....	430		
223	Stainless Steel 420 F	Quenched in oil	134	76	6.5	7.6	29	278	Good	R, W, B
224	Stainless Steel 430	Annealed sheet	75	45	27	55–40	29	146	Fair	DD, F, R, W, B
225	Stainless Steel 430 F	Annealed sheet	70–90	40-55	30–20	55–40	28	145–185	Good	R, W, B
226	Stainless Steel 431	Oil quenched 1750°F.	195	165	17	57	.....	418		
227	Stainless Steel 440									
	(*a*) Annealed	Annealed	100	60	23	40	29	197	Fair	R, W, B
	(*b*) Tempered	Tempered	250	190	3	6	.....	495		
228	Stainless Steel 441	Annealed	235	186	6	14	29.5	430	....	W, B
229	Stainless Steel 442	Annealed	80	55	35	55	.....	171	Fair	DD, F, R, W, B
230	Stainless Steel 446	Annealed sheet	80	50	23	50–40	29	175	Fair	F, R, W, B
231	Stainless Steel 501	Seamless tubes	60	25	30	.....	29	163	Fair	DD, F, R, W
232	Stainless Steel 502	Seamless tubing	60	25	30	.....	29	163	Fair	DD, F, R, W
233	Standard Alloy H R 4	....	....	....	....	.....	.....	....	Fair	DD, F, R, W
234	Standard Alloy H R 3	....	....	....	....	.....	.....	....	Fair	R, W
235	Standard Alloy H R 6	....	90	50	50	50	30	200	Fair	R, W
236	Symington Nirosta K A 2	....	65	35	35–40	55–65	.....	145–160	Good	
237	Thermalloy A	....	....	....	....	.....	.....	170–240		
238	Thermalloy B	....	....	....	....	.....	.....	170–240		
239	Thermalloy E	....	....	....	....	.....	.....	170–240		
240	Thermalloy 72	....	....	....	....	.....	.....	170–240		
241	Timken 16-13-3	Annealed	80 min.	30 min.	40 min.	.....	.....	200	Fair	DD, F, R, W
242	Timken 2512	Annealed tubing	90 min.	50 min.	30 min.	.....	.....	163 max.	Good	

243	Timken 2% Cr-0.5% Mo	Annealed	65.9	40.6	40.3	74.1	.....	137	Good	F, W
244	Timken 5% Cr-Mo	Annealed	66.6	26.3	39	80	.....	146	Good	F, W
245	Timken DM	Annealed	66.5	35.2	36.5	72.7	.....	149	Good	F, W
246	Timken Sicromo 5S	....	82.2	45	38	76	.....	163	Fair	F, W
247	Timken Sicromo 7	Annealed	60 min.	25 min.	30 min.	.....	.....	179	Good	F, DD, R, W
248	Timken Sicromo 7M	Annealed	60 min.	25 min.	30	.....	.....	179 max.	Good	DD, F, R, W
249	Timken Sicrome 9 M	Annealed	60 min.	25 min.	30 min.	.....	.....	179	Good	DD, F, R, W
250	Tisco 41	Casting	115-130	60-80	6-12	8-20	.....	220-235	....	W
251	Tisco 53	Casting	130-140	115-120	10-15	30-35	.....	280-300	Good	W
252	Tisco 80	Casting	....	....	....	.....	.....	300-600	Fair	
253	Tisco 102	Casting	65-70	25-30	50-60	45-60	.....	....	Poor	
254	Tisco 108	Casting	80-90	35-45	15-25	15-25	.....	160-180	Fair	
255	Tisco 130	Casting	40-60	30-40	0-3	0-5	.....	200-220	Good	
256	Tisco 131	Casting	75-100	70-90	5-12	6-15	.....	190-210	Good	
257	Tisco 160	Casting	20-35	....	....	.....	.....	120-170	Good	
258	Tophet C	Annealed wire	110	60	25					
259	Warman 5M	Casting	100	65	20-25	60-65	.....	....	Good	W
260	Warman 13	Heat-treated casting	85-95	45-50	25-30	50-60	.....	....	Good	
261	Worthite	Rolled	97	48	45	57	30	170	Good	W
		Sand casting	72	33	40	40	30	150		
262	X-7	Casting	75.4	41.1	17	17.5	24.2	172	Fair	W
263	X-ITE	Casting	62.5	35.8	5.5	7.2	25.9	169	Good	W
264	X-ITE-B	Casting	60	34	5.8	7.6	25	165	Good	W

Non-ferrous Alloys

265	Admiralty	Sheet, soft to hard	52-85	....	6-60	.....	15	....	Fair	DD, F, R, W, B
266	Adnic	Annealed strip	55	25	50	.....	21	....	Fair	DD, F, R, W, B
267	Advance	Wrought	60-100	25-75	30-35	50-65	.....	....	Good	DD, F, R, W, B
268	Alcoa 2 S[2]	A. S. T. M. specimen	13-24	5-21	15-45	60-80	10.3	....	Good	DD, F, R, W, B
269	Alcoa 3 S	A. S. T. M. specimen	16-29	6-25	10-40	45-75	10.3	....	Good	DD, F, R, W, B
270	Alcoa 52 S	A. S. T. M. specimen	29-41	14-36	8-30	40-65	10.3	....	Good	DD, F, R, W, B
271	Alcoa 53 S	A. S. T. M. specimen	16-39	7-33	20-35	50-70	10.3	....	Good	DD, F, R, W, B
272	Alcoa Alclad (72 S) 3 S	A. S. T. M. specimen	16-29	6-25	4-30	.....	.....	....	Good	DD, F, R, W
273	Alcoa 43	A. S. T. M. specimen	19-29	9-13	6	.....	10.3	....	Fair	R, W, B
274	Alcoa B 214	A. S. T. M. specimen	17-19	12-13	....	.....	10.3	....	Good	R, W
275	Alcoa 220-T 4	A. S. T. M. specimen	45	25	14	.....	10.3	....	Good	R, W
276	Alcoa 356	A. S. T. M. specimen	25-32	16-22	2-6	.....	10.3	....	Good	R, W, B

---

*B = brazing; DD = deep drawing; F = flanging; R = riveting; W = welding.

[2] In cast form known as Alcoa 100.

Table 4. Materials of Construction—Base Metals. Physical Properties and Methods of Fabrication— *(Continued)*

No.	Material	Form for which tensile properties are recorded	Tensile strength, 1000 lb. per sq. in.	Yield point, 1000 lb. per sq. in.	Elongation, % in 2 in.	Reduction of area, %	Elastic modulus, lb. per sq. in. (multiply by 10^6)	Brinell hardness	Machining qualities	Methods of fabrication*
277	Alcoa 406	A. S. T. M. specimen	16	....	6.5	.....	10.3	....	Good	R, W, B
278	Aleumite	Sand castings	65-70	23-25	30-35	30-35	16.5	120-140	Good	W, B
279	40 Alloy	Heat treated	30-40	15-35	1-10	.....	10	50-110	Good	
280	Alray A	Soft annealed	80-200	....	35-40	60	.....	....	Poor	F, R, W, B
281	Aluminum 99 +	Casting	12	....	40	.....	.....	18	Poor	
282	Aluminum 98-99	Casting	12	....	....	.....	.....	18	Poor	
283	Aluminum Bronze	Sheet, soft to hard	55-105	....	7-65	.....	.....	65-200	Fair	DD, F, R, B
284	Aluminum Bronze	As cast, bar	70-80	32-35	17-22	19-25	.....	120-140	Good	
285	Aluminum Bronze 5%	....	57-71	10-50	30-65	.....	16	....	Fair	
286	Aluminum Brass	Annealed tube	52-100	15-76	10-70	.....	15	....	Fair	DD, F, R, B
287	Ambrac 850	Sheet and rod, soft to hard	50-82	....	5-40	.....	19	58-145	Fair	DD, F, R, W, B
288	Ampco 18	Cast bar	77-84	35-42	10-14	6-10	14.4	159-183	Good	R, W, B
289	Ampcoloy	Cast bar	70-80	38-45	12-20	14-20	15	131-143	Good	R, W, B
290	Antimonial Admiralty	Annealed tube	50	....	70	.....	.....	....	Fair	
291	Antimonial Lead	Rolled sheet	4.5	2.2	50	.....	2	7.5	Good	DD, F, W
292	Antimonial Lead	Rolled	4	3	39	.....	2	9	Good	F, W
293	Arsenical Copper	Annealed tube	35	12	40	.....	15	....	Fair	F, R, W, B
294	Asarco Acid Lead	Cold rolled	2.3	....	40	100	2	4.5	Fair	DD, F, W
295	Beryllium Copper	Heat-treated sheet	175	....	5	.....	19	350	Fair	DD, F
296	Beryllium Copper	Cold rolled annealed	60-80	....	3.5	.....	18-19	....	Fair	DD, F, R, W, B
297	Cadmium-Copper	Rod	35-70	20-60	16-70	70-85	20	....	....	DD, F, B
298	Chemical Lead	Rolled sheet	2.3	1.2	50	.....	1.5	4.7	Good	DD, F, W
299	Chemical Lead	Rolled	1.9-2.3	0.7	50	.....	1-2	4.5-5.5	Good	F, W
300	Chromel A	Annealed	110	63	25-35	55	31	....	Good	F, R, W, B
301	Colmonoy 6	....	42	....	....	.....	....	587	Poor	W, B
302	Chromium Copper	Heat-treated rod	72	61	25	.....	.....	125	Fair	DD, F
303	Commercial Bronze F. C.	Rod half hard	45	40	30	.....	17	....	Good	
304	20% Cupro-Nickel	Annealed tube	55	....	45	.....	.....	....	Fair	DD, F, R, W, B
305	30% Cupro-Nickel	Sheet, soft to hard	55-77	22-70	5-40	.....	20	70-140	Fair	DD, F, R, W, B
306	Deoxidized Copper	Tube	31-55	12-44	6-56	60-72	16	42-111	Tough	DD, F, R, W, B
307	Durco D-10	Sand castings	65	45	5	5	.....	175	Fair	
308	Everdur-1000	Castings	50	20	25	25	15	80	Fair	W, B
309	Everdur-1015	Sheets, soft to hard	40-65	15-50	8-46	.....	15	45-125	Fair	DD, F, R, W, B

310	*Gold*	*Annealed*	*14*	....	*45*	.....	*11.3*	*35*	*Poor*	*DD, F, R, W, B*
311	Hastelloy A	Rolled annealed	110–120	47–52	40–48	40–54	27	200–215	Good	DD, F, W
312	Hastelloy B	Rolled annealed	130–140	60–65	40–45	40–45	30.7	210–235	Fair	DD, F, W
313	Hastelloy C	Casting	72–80	48–48	10–15	11–16	28.5	175–215	Fair	DD, F, W
314	Hastelloy D	Casting	36–40.5	....	0	0	28.8	....	Poor	W
315	Hardware Bronze	Annealed	38	16	35	.....	15	....	Good	R, B
316	Herculoy A	Hard to soft sheet and strip	67–113	30–104	5–60	.....	16	....	Fair	DD, F, R, W, B
317	Herculoy B	Hard to soft rod	42–88	14–60	20–70	75–85	15	....	Fair	W, B
318	High Brass	Sheet	46–90	15–55	3–64					
319	High Silicon Bronze	Rod	44–95	13–74	16–70	61–80	15	....	Fair	DD, F, B
320	Ilium G	Casting	60–70	50–60	....	.....	.....	160–210	Fair	W
321	Ilium R	Rolled annealed	100	50	30–45	.....	27.5	175–240	Good	DD, R, W
322	Inconel	Hot-rolled rods to cold-drawn wire	80–185	35–165	2–45	.....	31	....	Fair	DD, F, R, W, B
323	Inconel Clad Steel	Plate	55	30	30	50	.....	120	Fair	F, R, W
324	Indium Metal	Cold-rolled sheet	0.5	....	....	100	.....	1.0	Poor	DD, F, W
325	Iridio Platinum	Annealed	40–160	....	....	.....	.....	90–280	Good	DD, F, R, W, B
325	10% Iridium Platinum	Cold-drawn annealed wire	65	....	27	.....	.....	140	Fair	DD, F, R, W, B
327	30% Iridium Platinum	Cold-drawn annealed wire	145	....	....	.....	.....	240	Good	DD, F, R, W, B
328	40 M Alloy	Casting	26	13	9	.....	.....	50	Fair	
329	Molybdenum	Drawn wire	390	....	....	65	51.4	147	Poor	B
330	Monel	Hot-rolled rods to cold-drawn wire	80–170	40–130	2–45	.....	26	....	Fair	DD, F, R, W, B
331	Monel Clad Steel	Plate	55	30	30	50	.....	....	Fair	F, R, W
332	Muntz	Sheet	50–80	....	9.5–48	.....	12.8	76–172	Good	DD, F, B
333	Naval Brass	Annealed wire	55	23	40	60	.....	....	Fair	
334	Nichrome V	Wrought	100–200	60	35	55	.....	....	Good	DD, F, R, W, B
335	Nickel	Hot-rolled rods to cold-drawn wire	65–165	20–130	2–45	.....	30	....	Fair	DD, F, R, W, B
336	Nickel Clad Steel	Plates	55	30	30	50	.....	110	Fair	F, R, W
337	Nickel Silver 18%	Sheet soft to hard	58–85	....	4–40	.....	18	75–130	Fair	DD, F, R, W, B
338	Nickel Silver 18%	Sheet soft to hard	60–99	....	4–45	.....	18	75–175	Fair	DD, F, R, W, B
339	Nirex	Wrought	80–175	40	40	50	.....	....	Good	DD, F, R, W, B
340	O F H C	Rod and wire	32–65	12–45	5–50	60–90	18	30–120	Fair	DD, F, R, W, B
341	Olympic Bronze	Sheet	55–110	20–87	5–65	.....		....	....	DD, F, R, W, B
342	P D C P Copper	Rod or wire	30–65	15–45	5–50	60–90	16–18	30–120	Fair	DD, F, R, W, B
343	Palladium	Annealed	20		40	.....	13.8	46	Poor	DD, F, R, W, B
344	Phosphor Bronze A	Sheet, soft to hard	48–80	....	8–50	.....	15	60–140	Fair	DD, F, R, W, B
345	Phosphor Bronze D	Sheet, soft to hard	66–102	....	12–65	....	15	75–190	Fair	DD, F, R, W, B
346	Phosphor Bronze F. C.	Rod	60	....	20	....	15	120	Good	B
347	Pioneer	Casting	74	36.5	42	.....	.....	150	Good	
348	Platinum	Annealed wire	17	....	44	.....	21.4	42	Poor	DD, F, R, W, B
349	Platinum	Annealed wire	20	....	40	.....	21.5	40	Poor	DD, F, R, W, B
350	Red X 10	Heat treated	33–48	30–45	1	.....	10	100–140	Good	
351	Red Brass	Sheet	40–83	14–58	3–47	.....	.....	50–120	Fair	DD, F, R, W, B

* B = brazing; DD = deep drawing; F = flanging; R = riveting; W = welding.

Table 4. Materials of Construction—Base Metals. Physical Properties and Methods of Fabrication—(*Continued*)

No.	Material	Form for which tensile properties are recorded	Tensile strength, 1000 lb. per sq. in.	Yield point, 1000 lb. per sq. in.	Elongation, % in 2 in.	Reduction of area, %	Elastic modulus, lb. per sq. in. (multiply by 10^6)	Brinell hardness	Machining qualities	Methods of fabrication*
352	Rhodio Platinum	Annealed	30–100	......	......	.....	.....	70–150	Good	DD, F, R, W, B
353	Roofloy	Cold-rolled sheet	4	......	25	100	2	8.5	Fair	
354	70 S Alloy	Heat treated	28–40	26–38	1–5	.....	.....	70–90	Fair	
355	Silver, Fine[3]	....	......	......	......	.....	10.3			
356	Silver, Fine[4]	Annealed sheet 800 °F.	18–23	12	50	.....	10.3	25–35	Fair	DD, F, R, W, B
357	Stellite 1	Weld deposit	47	......	0	.....	34.9	....	Poor	W, B
358	Stellite 6	Casting	105	......	1	.....	30.4	....	Poor	W, B
359	Super Nickel Clad Steel	Plate	55	30	30	50	.....	....	Fair	F, R, W
360	Tantalum	Sheet, wire	42–178	......	......	.....	27	75–125	Poor	DD, F, W
361	Tellurium Lead	Rolled sheet	3.2	1.6	40	.....	1.5	6.0	Good	DD, F, W
362	Tellurium Lead	Rolled sheet	3.2	1.6	40	.....	1.5	6.0	Good	DD, F, W
363	Tophet A	Annealed wire	120	70	25					
364	Tungsten	Drawn wire	590	......	......	65	51.4	360	Poor	B
365	Wolverine Commercial Bronze	Tube	36–63	14–48	8–60	25–60	15	....	Fair	DD
366	Y Alloy	Casting	30–50	28–45	0–3	.....	10	80–150	Good	

* B = brazing; DD = deep drawing; F = flanging; R = riveting; W = welding.

[3] Coin, sterling, and clad are also available.

[4] Coin and sterling are also available.

Table 5. Comparative Prices of Stainless Steels

One of the determining factors in the selection of a material from which to construct equipment for use in the chemical industries is price. When several materials are of equal suitability from all other considerations, the engineer naturally chooses the one that is obtainable at the lowest cost. For the purpose of assisting in the selection, the current prices (September, 1940) of some of the alloy steels are given in the accompanying table. While these prices may change slightly from time to time, it is probable that their relation, one to another, will continue as at present.

It is essential to keep in mind that these are base prices of the steels in the form of plates, sheets, cold-rolled strip, etc. While the prices are comparable and therefore offer a useful guide, when selecting materials of construction other factors enter into the delivered cost of the alloy steel sheet, bar, wire, or other form. To the base price of the steel must be added extra charges that are due to such factors as quantity of the material ordered, the degree of surface finish specified and whether one or both sides are polished, width, length, and thickness. It is sometimes necessary to specify other requirements that add to the cost. Among those that should be mentioned are: close tolerance, testing, packing, machining, perforation, cutting to size, and, in the case of bars, shape. While this list of extras may appear to be long the average added cost for extras rarely prohibits the use of the steel.

Standard Type Numbers and Analyses Applicable Thereto*

Type* No.	Carbon	Chrome	Nickel	Other elements	Bars, drawn wire structurals, cts./lb.†	Plates, cts./lb.†	Sheets, cts./lb.†	Hot-rolled strip, cts./lb.†	Cold-rolled strip, cts./lb.†	Forging billets, cts./lb.†
301	0.09–0.20	16.00–18.00	7.00– 9.00	Mn 1.25 max.	24	27	34	20¼	25½	20.40
302	Over 0.08–0.20	17.50–20.00	8.00–10.00	Mn 1.25 max.	24	27	34	21½	28	20.40
302B	Over 0.08	17.50–20.00	8.00–10.00	Si 2.00–3.00; Mn 1.25 max.	24	27	34	21½	28	20.40
303	0.20 max.	17.50–20.00	8.00–10.00	S or Se 0.07 min. or Mo. 0.60 max.	26	29	36	27	33	22.10
304	0.08 max.	18.00–20.00	8.00–10.00	Mn 2.00 max.	25	29	36	23½	30	21.25
308	0.08 max.	19.00–22.00	10.00–12.00	Mn 2.00 max.	29	34	41	28½	35	24.65
309	0.20 max.	22.00–26.00	12.00–14.00	....	36	40	47	37	47	30.60
309S	0.08 max.	22.00–26.00	12.00–14.00	....	40	44	51	41	51	34.00
310	0.25 max.	24.00–26.00	19.00–21.00	....	49	52	53	48¾	56	41.65
311	0.25 max.	19.00–21.00	24.00–26.00	....	49	52	53	48¾	56	41.65
312	0.25 max.	27.00–31.00	8.00–10.00	....	36	40	49	....	....	30.60
315	0.15 max	17.00–19.00	7.00– 9.50	Cu 1.00–1.50; Mo 1.00–1.50	36	42	46	....	....	......
316	0.10 max.	16.00–18.00	10.00–14.00	Mo 2.00–3.00	40	44	48	40	48	34.00
317	0.10 max.	18.00–20.00	10.00–14.00	Mo 3.00–4.00	50	54	58	50	58	42.50

When carbon content 0.11 or under is specified or required in Types 301, 302, and 302B, the price of Type 304 applies.

No specific composition limits within the above ranges may be placed on Types 301, 302, 302B, and 303 except that carbon may be specified to a four-point range within the above limits.

* American Iron and Steel Institute type numbers and compositions, June, 1939.

† *Chemical & Metallurgical Engineering*, September, 1940.

Table 5. Comparative Prices of Stainless Steels—(*Continued*)

Type* No.	Carbon	Chrome	Nickel	Other elements	Bars, drawn wire structurals, cts./lb.†	Plates, cts./lb.†	Sheets, cts./lb.†	Hot-rolled strip, cts./lb.†	Cold-rolled strip, cts./lb.†	Forging billets, cts./lb.†
321	0.10 max.	17.00–20.00	7.00–10.00	Ti min. 4 × C	29	34	41	29¼	38	24.65
325	0.25 max.	7.00–10.00	19.00–23.00	Cu 1.00–1.50	26	30	37	26¼	34	22.10
327	0.25 max.	25.00–30.00	3.00– 5.00	....	31	36	42	30¾	41	26.35
329	0.10 max.	25.00–30.00	3.00– 5.00	Mo 1.00–1.50	36	40	44	36	45	30.60
330	0.25 max.	14.00–16.00	33.00–36.00	....	49	52	53	....	....	41.65
347	0.10 max.	17.00–20.00	8.00–12.00	Cb 10 × C	33	38	45	33	42	28.05
403	0.15 max.	11.50–13.00	....	Turbine quality	21½	24½	29½	21¼	27	18.275
405	0.08 max.	11.50–13.50	....	Al 0.10–0.20	20	23	28	19¾	25½	17.00
406	0.15 max.	12.00–14.00	....	Al 4.00–4.50	23	26	31	22¾	31	19.55
410	0.15 max.	12.00–14.00	....	....	18½	21½	26½	17	22	15.725
414	0.15 max.	10.00–14.00	2.00 max.	....	18½	21½	26½	17	22	15.725
416	0.15 max.	10.00–14.00	....	S or Se 0.07 min. or Mo 0.60 max.	19	22	27	18¼	23½	16.15
418	0.15 max.	12.00–14.00	....	W 2.50–3.50	Prices quoted upon application					
420	Over 0.15	12.00–14.00	....	....	24	28½	33½	23¾	36½	20.40
420F	Over 0.15	12.00–14.00	....	S or Se 0.07 min. or Mo 0.60 max.	24½	....	....	....	....	......
430	0.12 max.	14.00–18.00	....	....	19	22	29	17½	22½	16.15
430F	0.12 max.	14.00–18.00	....	S or Se 0.07 min. or Mo 0.60 max.	19½	22½	29½	18¾	24½	16.575
431	0.15 max.	14.00–18.00	2.00 max.	....	19	22	29	17½	22½	16.15
438	0.12 max.	16.00–18.00	....	W 2.50–3.50	Prices quoted upon application					
439	0.50–0.65	8.00	....	W 8.00	Prices quoted upon application					
440	Over 0.12	14.00–18.00	....	....	24	28½	33½	23¾	36½	20.40
441	Over 0.15	14.00–18.00	2.00 max.	....	24	28½	33½	23¾	36½	20.40
442	0.35 max.	18.00–23.00	....	....	22½	25½	32½	24	32	19.125
446	0.35 max.	23.00–30.00	....	....	27½	30½	36½	35	52	23.375
501	Over 0.10	4.00– 6.00	....	....	8	12	15¾	12	17	7.20
502	0.10 max.	4.00– 6.00	....	....	9	13	16¾	13	18	8.10

When carbon content 0.11 or under is specified or required in Types 301, 302, and 302B, the price of Type 304 applies.

No specific composition limits within the above ranges may be placed on Types 301, 302, 302B, and 303 except that carbon may be specified to a four-point range within the above limits.

* American Iron and Steel Institute type numbers and compositions, June, 1939.

† *Chemical & Metallurgical Engineering*, September, 1940.

Table 6. S.A.E. Steel Numbering System*

Compositions that do not conform to the S.A.E. compositions or that are not included in the S.A.E. Standard, should not bear the prefix "S.A.E."

A numeral index system is used to identify the compositions of the S.A.E. steels, which makes it possible to use numerals on shop drawings and blueprints that are partially descriptive of the composition of material covered by such numbers. The first digit indicates the type to which the steel belongs: thus 1– indicates a carbon steel, 2– a nickel steel, and 3– a nickel chromium steel. In the case of the simple alloy steels the second digit generally indicates the approximate percentage of the predominant alloying element. Usually the last two or three digits indicate the average carbon content in "points," or hundredths of 1 per cent. Thus 2340 indicates a nickel steel of approximately 3 per cent nickel (3.25 to 3.75) and 0.40 per cent carbon (0.35 to 0.45); and 71360 indicates a tungsten steel of about 13 per cent tungsten (12 to 15) and 0.60 per cent carbon (0.50 to 0.70).

In some instances, in order to avoid confusion, it has been found necessary to depart from this system of identifying the approximate alloy composition of a steel by varying the second and third digits of the number. An instance of such departure is the steel numbers selected for several of the corrosion and heat-resisting alloys.

The basic numerals for various types of S.A.E. steel are

Type of Steel	Numerals (and Digits)
Carbon steels	1xxx
Plain carbon	10xx
Free cutting (screw stock)	11xx
Free cutting, manganese	X13xx
High manganese	T13xx
Nickel steels	2xxx
0.50 % nickel	20xx
1.50 % nickel	21xx
3.50 % nickel	23xx
5.00 % nickel	25xx
Nickel chromium steels	3xxx
1.25 % nickel, 0.60 % chromium	31xx
1.75 % nickel, 1.00 % chromium	32xx
3.50 % nickel, 1.50 % chromium	33xx
3.00 % nickel, 0.80 % chromium	34xx
Corrosion and heat-resisting steels	30xxx
Molybdenum steels	4xxx
Chromium	41xx
Chromium nickel	43xx
Nickel	46xx and 48xx
Chromium steels	5xxx
Low chromium	51xx
Medium chromium	52xxx
Corrosion and heat resisting	51xxx
Chromium vanadium steels	6xxx
Tungsten steels	7xxx and 7xxxx
Silicon manganese steels	9xxx

Prefixes:

The prefix X is used in numerous instances to denote variations in the range of elements.

The prefix T is used with the manganese steels (1300 series) to avoid confusion with steels of somewhat different manganese range that have been identified by the same numerals but without the prefix.

---

* Reprinted from the S.A.E. Handbook, 1940 edition, by permission of the Society of Automotive Engineers, Inc.

Table 6. S.A.E. Steel Numbering System—(*Continued*)

Chemical Compositions

Carbon Steels

S.A.E. No.	Carbon range	Manganese range	Phosphorus, max.	Sulfur, max.
1010	0.05–0.15	0.30–0.60	0.045	0.055
1015	0.10–0.20	0.30–0.60	0.045	0.055
X1015	0.10–0.20	0.70–1.00	0.045	0.055
1020	0.15–0.25	0.30–0.60	0.045	0.055
X1020	0.15–0.25	0.70–1.00	0.045	0.055
1025	0.20–0.30	0.30–0.60	0.045	0.055
X1025	0.20–0.30	0.70–1.00	0.045	0.055
1030	0.25–0.35	0.60–0.90	0.045	0.055
1035	0.30–0.40	0.60–0.90	0.045	0.055
1040	0.35–0.45	0.60–0.90	0.045	0.055
X1040	0.35–0.45	0.40–0.70	0.045	0.055
1045	0.40–0.50	0.60–0.90	0.045	0.055
X1045	0.40–0.50	0.40–0.70	0.045	0.055
1050	0.45–0.55	0.60–0.90	0.045	0.055
X1050	0.45–0.55	0.40–0.70	0.045	0.055
1055	0.50–0.60	0.60–0.90	0.040	0.055
X1055	0.50–0.60	0.90–1.20	0.040	0.055
1060	0.55–0.70	0.60–0.90	0.040	0.055
1065	0.60–0.75	0.60–0.90	0.040	0.055
X1065	0.60–0.75	0.90–1.20	0.040	0.055
1070	0.65–0.80	0.60–0.90	0.040	0.055
1075	0.70–0.85	0.60–0.90	0.040	0.055
1080	0.75–0.90	0.60–0.90	0.040	0.055
1085	0.80–0.95	0.60–0.90	0.040	0.055
1090	0.85–1.00	0.60–0.90	0.040	0.055
1095	0.90–1.05	0.25–0.50	0.040	0.055

Free Cutting Steels

S.A.E. No.	Carbon range	Manganese range	Phosphorus range	Sulfur range
1112	0.08–0.16	0.60–0.90	0.09–0.13	0.10 –0.20
X1112	0.08–0.16	0.60–0.90	0.09–0.13	0.20 –0.30
1115	0.10–0.20	0.70–1.00	0.045 max.	0.075–0.15
1120†	0.15–0.25	0.60–0.90	0.045 max.	0.075–0.15
X1314	0.10–0.20	1.00–1.30	0.045 max.	0.075–0.15
X1315	0.10–0.20	1.30–1.60	0.045 max.	0.075–0.15
X1330	0.25–0.35	1.35–1.65	0.045 max.	0.075–0.15
X1335	0.30–0.40	1.35–1.65	0.045 max.	0.075–0.15
X1340	0.35–0.45	1.35–1.65	0.045 max.	0.075–0.15

† The Iron and Steel Division contemplates discontinuing S.A.E. 1120 at an early date in view of its gradual displacement in industry by S.A.E. 1115. (Note—Dec. 7, 1937.)

Table 6. S.A.E. Steel Numbering System—(*Continued*)

Chemical Compositions

Manganese Steels†

S.A.E. No.	Carbon range	Manganese range	Phosphorus, max.	Sulfur, max.
T1330	0.25–0.35	1.60–1.90	0.040	0.050
T1335	0.30–0.40	1.60–1.90	0.040	0.050
T1340	0.35–0.45	1.60–1.90	0.040	0.050
T1345	0.40–0.50	1.60–1.90	0.040	0.050
T1350	0.45–0.55	1.60–1.90	0.040	0.050

Nickel Steels†

S.A.E. No.	Carbon range	Manganese range	Phosphorus, max.	Sulfur, max.	Nickel range
2015	0.10–0.20	0.30–0.60	0.040	0.050	0.40–0.60
2115	0.10–0.20	0.30–0.60	0.040	0.050	1.25–1.75
2315	0.10–0.20	0.30–0.60	0.040	0.050	3.25–3.75
2320	0.15–0.25	0.30–0.60	0.040	0.050	3.25–3.75
2330	0.25–0.35	0.50–0.80	0.040	0.050	3.25–3.75
2335	0.30–0.40	0.50–0.80	0.040	0.050	3.25–3.75
2340	0.35–0.45	0.60–0.90	0.040	0.050	3.25–3.75
2345	0.40–0.50	0.60–0.90	0.040	0.050	3.25–3.75
2350	0.45–0.55	0.60–0.90	0.040	0.050	3.25–3.75
2515	0.10–0.20	0.30–0.60	0.040	0.050	4.75–5.25

Nickel Chromium Steels†

S.A.E. No.	Carbon range	Manganese range	Phosphorus, max.	Sulfur, max.	Nickel range	Chromium range
3115	0.10–0.20	0.30–0.60	0.040	0.050	1.00–1.50	0.45–0.75
3120	0.15–0.25	0.30–0.60	0.040	0.050	1.00–1.50	0.45–0.75
3125	0.20–0.30	0.50–0.80	0.040	0.050	1.00–1.50	0.45–0.75
3130	0.25–0.35	0.50–0.80	0.040	0.050	1.00–1.50	0.45–0.75
3135	0.30–0.40	0.50–0.80	0.040	0.050	1.00–1.50	0.45–0.75
3140	0.35–0.45	0.60–0.90	0.040	0.050	1.00–1.50	0.45–0.75
X3140	0.35–0.45	0.60–0.90	0.040	0.050	1.00–1.50	0.60–0.90
3145	0.40–0.50	0.60–0.90	0.040	0.050	1.00–1.50	0.45–0.75
3150	0.45–0.55	0.60–0.90	0.040	0.050	1.00–1.50	0.45–0.75
3215	0.10–0.20	0.30–0.60	0.040	0.050	1.50–2.00	0.90–1.25
3220	0.15–0.25	0.30–0.60	0.040	0.050	1.50–2.00	0.90–1.25
3230	0.25–0.35	0.30–0.60	0.040	0.050	1.50–2.00	0.90–1.25
3240	0.35–0.45	0.30–0.60	0.040	0.050	1.50–2.00	0.90–1.25
3245	0.40–0.50	0.30–0.60	0.040	0.050	1.50–2.00	0.90–1.25
3250	0.45–0.55	0.30–0.60	0.040	0.050	1.50–2.00	0.90–1.25
3312	max. 0.17	0.30–0.60	0.040	0.050	3.25–3.75	1.25–1.75
3325	0.20–0.30	0.30–0.60	0.040	0.050	3.25–3.75	1.25–1.75
3335	0.30–0.40	0.30–0.60	0.040	0.050	3.25–3.75	1.25–1.75
3340	0.35–0.45	0.30–0.60	0.040	0.050	3.25–3.75	1.25–1.75
3415	0.10–0.20	0.30–0.60	0.040	0.050	2.75–3.25	0.60–0.95
3435	0.30–0.40	0.30–0.60	0.040	0.050	2.75–3.25	0.60–0.95
3450	0.45–0.55	0.30–0.60	0.040	0.050	2.75–3.25	0.60–0.95

---

† Silicon range of all S.A.E. basic open-hearth alloy steels shall be 0.15–0.30. For electric and acid open-hearth alloy steels, the silicon content shall be 0.15 minimum.

Table 6. S.A.E. Steel Numbering System—(*Continued*)

Chemical Compositions

Molybdenum Steels†

S.A.E. No.	Carbon range	Manganese range	Phosphorus, max.	Sulfur, max.	Chromium range	Nickel range	Molybdenum range
4130	0.25–0.35	0.50–0.80	0.040	0.050	0.50–0.80	....	0.15–0.25
X4130	0.25–0.35	0.40–0.60	0.040	0.050	0.80–1.10	....	0.15–0.25
4135	0.30–0.40	0.60–0.90	0.040	0.050	0.80–1.10	....	0.15–0.25
4140	0.35–0.45	0.60–0.90	0.040	0.050	0.80–1.10	....	0.15–0.25
4150	0.45–0.55	0.60–0.90	0.040	0.050	0.80–1.10	....	0.15–0.25
4320	0.15–0.25	0.40–0.70	0.040	0.050	0.30–0.60	1.65–2.00	0.20–0.30
4340	0.35–0.45	0.50–0.80	0.040	0.050	0.50–0.80	1.50–2.00	0.30–0.40
X4340	0.35–0.45	0.50–0.80	0.040	0.050	0.60–0.90	1.50–2.00	0.20–0.30
4615	0.10–0.20	0.40–0.70	0.040	0.050	....	1.65–2.00	0.20–0.30
4620	0.15–0.25	0.40–0.70	0.040	0.050	....	1.65–2.00	0.20–0.30
4640	0.35–0.45	0.50–0.80	0.040	0.050	....	1.65–2.00	0.20–0.30
4815	0.10–0.20	0.40–0.60	0.040	0.050	....	3.25–3.75	0.20–0.30
4820	0.15–0.25	0.40–0.60	0.040	0.050	....	3.25–3.75	0.20–0.30

Chromium Steels†

S.A.E. No.	Carbon range	Manganese range	Phosphorus, max.	Sulfur, max.	Chromium range
5120	0.15–0.25	0.30–0.60	0.040	0.050	0.60–0.90
5140	0.35–0.45	0.60–0.90	0.040	0.050	0.80–1.10
5150	0.45–0.55	0.60–0.90	0.040	0.050	0.80–1.10
52100	0.95–1.10	0.20–0.50	0.030	0.035	1.20–1.50

Chromium Vanadium Steels†

S.A.E. No.	Carbon range	Manganese range	Phosphorus, max.	Sulfur, max.	Chromium range	Vanadium	
						Min.	Desired
6115	0.10–0.20	0.30–0.60	0.040	0.050	0.80–1.10	0.15	0.18
6120	0.15–0.25	0.30–0.60	0.040	0.050	0.80–1.10	0.15	0.18
6125	0.20–0.30	0.60–0.90	0.040	0.050	0.80–1.10	0.15	0.18
6130	0.25–0.35	0.60–0.90	0.040	0.050	0.80–1.10	0.15	0.18
6135	0.30–0.40	0.60–0.90	0.040	0.050	0.80–1.10	0.15	0.18
6140	0.35–0.45	0.60–0.90	0.040	0.050	0.80–1.10	0.15	0.18
6145	0.40–0.50	0.60–0.90	0.040	0.050	0.80–1.10	0.15	0.18
6150	0.45–0.55	0.60–0.90	0.040	0.050	0.80–1.10	0.15	0.18
6195	0.90–1.05	0.20–0.45	0.030	0.035	0.80–1.10	0.15	0.18

---

† Silicon range of all S.A.E. basic open-hearth alloy steels shall be 0.15–0.30. For electric and acid open-hearth alloy steels, the silicon content shall be 0.15 minimum.

Table 6. S.A.E. Steel Numbering System—(*Continued*)

Chemical Compositions

Tungsten Steels†

S.A.E. No.	Carbon range	Manganese, max.	Phosphorus, max.	Sulfur, max.	Chromium range	Tungsten range
71360	0.50–0.70	0.30	0.035	0.040	3.00–4.00	12.00–15.00
71660	0.50–0.70	0.30	0.035	0.040	3.00–4.00	15.00–18.00
7260	0.50–0.70	0.30	0.035	0.040	0.50–1.00	1.50– 2.00

Silicon Manganese Steels

S.A.E. No.	Carbon range	Manganese range	Phosphorus, max.	Sulfur, max.	Silicon range
9255	0.50–0.60	0.60–0.90	0.040	0.050	1.80–2.20
9260	0.55–0.65	0.60–0.90	0.040	0.050	1.80–2.20

Corrosion and Heat-resisting Alloys

S.A.E. No.	Carbon, max.	Manganese, max.	Silicon, max.	Phosphorus, max.	Sulfur, max.	Chromium range	Nickel range
30905	0.08	0.20–0.70	0.75	0.030	0.030	17.00–20.00	8.00–10.00
30915	0.09–0.20	0.20–0.70	0.75	0.030	0.030	17.00–20.00	8.00–10.00
51210	0.12	0.60	0.50	0.030	0.030	11.50–13.00	
X51410	0.12	0.60	0.50	0.030	0.15–0.50	13.00–15.00	
51335	0.25–0.40	0.60	0.50	0.030	0.030	12.00–14.00	
51510	0.12	0.60	0.50	0.030	0.030	14.00–16.00	
51710	0.12	0.60	0.50	0.030	0.030	16.00–18.00	

† Silicon range of all S.A.E. basic open-hearth alloy steels shall be 0.15–0.30. For electric and acid open-hearth alloy steels, the silicon content shall be 0.15 minimum.

Table 7. Properties of High-chromium Steels at High Temperatures

Cr	Composition per cent					Scaling temp., °F.	Strength at elevated temps., lb. per sq. in.			Creep strength, lb. per sq. in. 10,000 hr. with 1% elong.		
	Ni	Mo	Si	Cb	C		1300°F.	1500°F.	1700°F.	1000°F.	1300°F.	1500°F.
14–16....	..	..	..	..	0.12	1300	15,000	8,000	8,000	10,000	1,400	
16–18....	..	..	..	..	0.12	1500	15,000	8,000	5,000	8,500	1,400	
23–30....	..	..	..	..	0.25	1900	20,000	10,000	6,000	....	800	
18.......	8	..	..	..	.....	1600	42,000	28,000	18,000	17,000	4,000	850
18.......	8	..	..	1.0*	0.10	1600	56,000	32,000	18,000	18,000	4,500	850
18.......	8	3	..	..	0.12	1650	60,000	40,000	18,000	25,000	10,000	3,000
18.......	8	..	3	..	0.12	1700	60,000	32,000	24,000			
25.......	12	..	2	..	0.20	2000	54,000	32,000	16,000	17,000	4,800	850
25.......	20	..	2	..	0.25	2000	45,000	28,000	14,000	17,000	5,200	1,000

Copyright by *Chem. & Met. Eng.*, November, 1938.

* Or 0.60% Ti.

Table 8. Noble Metals and Their Alloys

No.	Material	Manufacturer	Essential nominal chemical composition, per cent	Sp. gr.	M.P., °F.	Form for which tensile prop. are recorded	Tensile strength, 1000 lb. per sq. in.	Elongation, % in 2 in.	Brinell hardness	Machining qualities	Abrasion resistance	Forms available*
1	Fine Gold	Baker & Co., Newark, N.J.	Au, 99.99	19.3	1945	Annealed	14	45	35	Poor	Poor	B, C, CR, HR, D, P, S, R, T, W
2	Gold-Platinum	Baker & Co., Newark, N.J.	Au, 70; Pt, 30	19.9	2160–2590	Annealed	65		135	Good	Good	
3	Iridioplatinum	Baker & Co., Newark, N.J.	Pt, 70; Ir, 30	21.7	3620	Annealed	160		280	Good	Good	B, C, CR, HR, S, R, W
4	Iridio Platinum	J. Bishop & Co., Malvern, Pa.	Pt, 75; Ir, 25	20.9		Annealed wire	133	20	185	......	......	B, C, CR, HR, D, P, S, R, T, W
5	Iridioplatinum	Baker & Co., Newark N.J.	Pt, 80; Ir, 20	21.6	3410	Annealed	100	26	200	Good	Good	B, C, CR, HR, D, P, S, R, W
6	20% Iridio Platinum	J. Bishop & Co., Malvern, Pa.	Pt, 80; Ir, 20	21.7	3353	Annealed wire	103	20	150	......	......	B, C, CR, HR, D, P, S, R, T, W
7	15% Iridio Platinum	J. Bishop & Co., Malvern, Pa.	Pt, 85; Ir, 15	21.6	3344	Annealed wire	80	20	103	......	......	B, C, CR, HR, D, P, S, R, T, W
8	Iridioplatinum	Baker & Co., Newark, N.J.	Pt, 90; Ir, 10	21.5	3300	Annealed	55	30	130	Good	Good	B, C, CR, HR, D, P, S, R, T, W
9	10% Iridio Platinum	J. Bishop & Co., Malvern, Pa.	Pt, 90; Ir, 10	21.5	3290	Annealed wire	67	25	92	......	......	B, C, CR, HR, D, P, S, R, T, W
10	5% Iridio Platinum	J. Bishop & Co., Malvern, Pa.	Pt, 95; Ir, 5	21.6	3218	Annealed wire	39	32	56	......	......	B, CR, D, P, S, R, T, W
11	Iridium	Baker & Co., Newark, N.J.	Ir, 99.8	22.4	4450	Annealed			172	Fair	Good	B, C, S
12	Osmium	Baker & Co., Newark, N.J.	Os, 99.5+	22.5	4900					Poor	Good	B, C
13	Palladium	Baker & Co., Newark, N.J.	Pd, 99.5+	12	2830	Annealed	20	40	49	Poor	Poor	B, CR, HR, D, P, S, R, T, W, C
14	Palladioplatinum	Baker & Co., Newark, N.J.	Pt, 80; Pd, 20	18.3	3145	Annealed	50		94	Fair	Fair	B, CR, HR, D, P, S, R, T, W, C
15	Platinum, C. P.	Baker & Co., Newark, N.J.	Pt, 99.99	21.5	3223	Annealed	18	44	47	Poor	Fair	B, CR, HR, D, P, S, R, T, W, C
16	Platinum, C. P.	J. Bishop & Co., Malvern, Pa.	Pt, 99.9	21.4	3191	Annealed	23	50	35	......	......	B, CR, HR, D, P, S, R, T, W
17	Platinum, Comm.	Baker & Co., Newark, N.J.	Pt, 99.0+	21.4	3223	Annealed	21	37	50	Poor	Fair	B, C, CR, HR, D, P, S, R, T, W
18	Platinum-Gold	Baker & Co., Newark, N.J.	Pt, 90; Au, 10	21.3	2950–3125	Annealed	80	16	150	Good	Good	B, C, S, R, W
19	Rhodioplatinum	Baker & Co., Newark, N.J.	Pt, 90; Rh, 10	20.0	3350	Annealed	45	40	90	Good	Fair	B, C, CR, HR, D, P, S, R, T, W

20	Rhodium	Baker & Co., Newark, N.J.	Rh, 99.8+	12.4	3570	Annealed	135	....	...	Poor	Fair	B, C, CR, HR, S, R, W
21	10% Rhodium	J. Bishop & Co., Malvern, Pa.	Pt, 90; Rh, 10	20.2	3310	Annealed wire	47	37	69	......	......	B, C, CR, HR, D, P, S, R, T, W
22	Ruthenium	Baker & Co., Newark, N.J.	Ru, 99.5	12.2	4440	Cast	....	....	220	Poor	Good	B, C

No.	Material	Manufacturer	Essential nominal chemical composition, per cent	Sp. gr.	M.P., °F.	Form for which tensile prop. are recorded	Tensile strength, 1,000 lb. per sq. in.	Yield point, 1,000 lb. per sq. in.	Elongation, % in 2 in.	Rockwell B hardness
23	Coin Silver	Handy & Harman, New York American Platinum Works, Newark, N.J. Baker & Co., Newark, N.J. J. Bishop & Co., Malvern, Pa.	Ag, 90; Cu, 10	10.35	1435–1615	Hard Annealed	70–75 35–40	60–65 25–30	5–7 30–35	80–90 30–40
24	Fine Silver	Handy & Harman, New York American Platinum Works, Newark, N.J. Baker & Co., Newark, N.J. J. Bishop & Co., Malvern, Pa.	Ag, 99.9+	10.53	1762	Hard Annealed	41 23	35–40 15–20	6–10 40–50	50–56 5–15
25	Sterling Silver	Handy & Harman, New York American Platinum Works, Newark, N.J. Baker & Co., Newark, N.J. J. Bishop & Co., Malvern, Pa.	Ag, 92.5; Cu, 7.5	10.40	1435–1635	Hard Annealed	67–72 35–40	55–60 20–25	4–6 30–35	75–85 25–40
26	Silver-Clad Copper	Handy & Harman, New York	Tensile properties same as copper							
27	Silver-Clad Nickel	Handy & Harman, New York	Tensile properties slightly greater than nickel portion							
28	Silver-Clad Steel	Handy & Harman, New York	Ag, 10–16; bal., SAE 1020 steel. Tensile properties slightly greater than steel portion							

Copyright by *Chem. & Met. Eng.*, November, 1938.

* Forms available: B, bars; C, casting; CR, cold rolled; HR, hot rolled; D, drawn; P, plates; S, sheets; R, rods; T, tubes; W, wire.

Table 9A. Glass-lined and Enameled Steel*

a. Coefficients of Heat Transfer for Typical Chemical Enamels.†
[In B.t.u./(sq. ft.) (hr.) per (°F. temperature difference)]

1. Dry dust enamel for very severe chemical service (Pfaudler No. 17)
   Steam to water being heated.. U = 60
   Steam to boiling water.. U = 80
2. Dry dust enamel for very severe chemical service (Pfaudler No. 11)
   Steam to water being heated.. U = 50
   Steam to boiling water.. U = 70
3. Spray enamel for organic acids and less corrosive mineral acids (Pfaudler No. 71)
   Steam to water being heated.. U = 125
   Steam to boiling water.. U = 150
4. Spray dust enamel for mineral acids in general storage service, usually for equipment of large capacities (Pfaudler No. 186).
   Steam to water being heated.. U = 90
   Steam to boiling water.. U = 100

b. Other Physical and Mechanical Properties.

Such physical properties of enamel as the coefficient of expansion, tensile strength, etc., are generally meaningless unless considered in connection with the steel on which the lining is applied. Ability to withstand rapid thermal change is tested in an automatic machine by intermittently heating enamel plates to 450°F. and then dipping them into ice water. However, both temperature and corrosion conditions to be met in each service application should be definitely ascertained and this information supplied to the manufacturer.

---

[*] From *Chem. & Met. Eng.*, December, 1932.

[†] Agitation and size of enameled piece as related to agitation are determining factors in establishing accurate coefficients of heat transfer for equipment of this kind. Good agitation may increase values by as much as 20 to 25 per cent.

Table 9B. Glass, Glass-lined, and Fused Silica Equipment

Physical Properties of Low-expansion Glasses, Fused Quartz and Fused Silica

Material	Specific gravity	Specific volume, cu. in. per lb.	Tensile strength, lb. per sq. in.	Modulus of elasticity, lb. per sq. in. (multiply by 10^5)	Hardness*	Thermal expansion per °C. (multiply by 10^{-5})	Thermal cond., cal. per sec., cm.², °C., cm. (multiply by 10^4)	Specific heat, cal. per °C., gm.	Softening point, °F.	Breakdown voltage, 60 cycles, volts per mil	Dielectric constant, 60 cycles	Refractive index, n_{AD}	Transparency†	Forms available‡
Borosilicate glass	2.23	12.4	10,000	98	...	0.32	24.5	0.20	1,505	3,200 (0.1 in.)	4.6	1.47	T, TL	S, R, T, other
96% silica glass	2.18	12.7	....	....	...	0.080	....	....	2,750 ±90	3,000	4.0	1.458	T	R, T, other
Fused quartz	2.20	12.6	4,000	105–126	4.9	0.054	35	0.25	2,600	500 (¼ in.)	Approx. 3.8	1.459	T	S, R, T, other
Fused silica	2.07	13.4	400–800	94–114	...	0.054	25	....	2,600	250 (¼ in.)	3.7	.....	TL, O	S, R, T, other

Copyright by *Chem. & Met. Eng.*, September, 1940.

* Hardness: 2.5 mm. ball, 25 kg. load, depth in 1/200 mm.

† T = transparent; TL = translucent; O = opaque.

‡ S = sheets; R = rods; T = tubes.

Makers of Glass, Glass-lined, and Fused Silica Equipment

Manufacturer	Composition, forms available	Manufacturer	Composition, forms available
Alsop Engineering Co., Milldale, Conn.	Glass-lined steel tanks and mixers	Hanovia Chemical & Mfg. Co., Newark, N. J.	Transparent fused quartz in all shapes
Amersil Corp., Hillside, N. J.	Fused silica ware such as pans, pipes, gas coolers, absorbers	Metal-Glass Products Co., Belding, Mich.	Glass-enameled steel equipment
Corning Glass Works, Corning, N. Y.	Special heat- and corrosion-resisting borosilicate glass supplied in various forms: pipe, columns, etc. Also 96 per cent high silica glassware now available for laboratory use	Owens-Corning Fiberglas Corp., Toledo, Ohio.	Fibrous glass filter cloths and dust filters
Ertel Engineering Co., New York	Glass-enameled tanks	The Pfaudler Co., Rochester, N. Y.	Wide variety of standard and special glass-enameled steel equipment—various formulas
General Electric Co., Schenectady, N. Y.	Transparent fused quartz in various small sized articles	A. O. Smith Corp., Milwaukee, Wis.	Glass-enameled steel equipment
Glascote Products, Inc., Euclid, Ohio	Glass-enameled steel equipment	The Thermal Syndicate, Brooklyn, N. Y.	Fused silica (non-transparent) supplied in various large forms; fused quartz (transparent) in smaller sizes
		Vitreous Enameling & Stamping Co., New York	Enameled specialties, tanks

Copyright by *Chem. & Met. Eng.*, September, 1940.

Table 10. Chemical Brick, Stoneware, and Cements

Physical Properties of Chemical Stoneware

The accompanying table, which has been prepared by the General Ceramics Co., gives the physical properties of an average grade of chemical stoneware. It should be emphasized here that "chemical stoneware" is not the name of a definite material, such as an alloy, but a generic term applied to a wide variety of ceramic compositions and hence that, in any particular composition designed to give optimum properties in one respect, it will ordinarily be impossible to secure optimum properties in all other respects.

Specific gravity	2.2	Modulus of elasticity, lb. per sq. in.	8×10^6
Hardness, scleroscope	100	Specific heat	0.2
Ultimate tensile strength, lb. per sq. in.	2,000	Thermal conductivity, B.t.u. per hr., sq. ft., (°F./in.)	0.833
Ultimate compressive strength, lb. per sq. in.	80,000	Linear thermal expansion, per °F.	2×10^{-6}
Modulus of rupture, lb. per sq. in.	5,000	Water absorption, per cent	0–4

Makers of Acid-proof Brick and Chemical Stoneware

Manufacturer	Materials
Acme Brick Co., Fort Worth, Tex.	Acid-proof brick
Alabama Clay Products Co., Birmingham, Ala.	Acid-proof brick
Alberene Stone Corp. of Va., New York	Acid-proof stone
Atlas Mineral Products Co., Mertztown, Pa.	Acid-proof brick construction
Belden Brick Co., Canton, Ohio	Acid-proof brick
Buckeye Pottery Co., Macomb, Ill.	Acid-proof ceramics
Charlotte Chemical Labs., Charlotte, N.C.	Acid-proof brick, rings
Claycraft Co., Columbus	Acid-proof brick
Custodis Construction Co., New York	Acid-proof brick construction, towers, tanks
Electro-Chemical Supply & Engineering Co., Paoli, Pa.	Acid-proof brick and masonry construction
Filtros, Inc., East Rochester, N.Y.	Acid-proof mineral as plates, cylinders, etc.
General Ceramics Co., New York	Chemical stoneware of all types
General Refractories Co., Philadelphia	Acid-proof tower packing, brick
Harbison-Walker Refractories Co., Pittsburgh	Acid-proof brick
B. Mifflin Hood Co., Daisy, Tenn.	Acid-proof tower packings and flooring tiles
Ironton Fire Brick Co., Ironton, Ohio	Acid-proof brick
Keagler Brick Co., Steubenville, Ohio	Acid-proof brick
Kewaunee Mfg. Co., Kewaunee, Wis.	Karcite carbon-filled acid-proof ceramic
Maurice A. Knight, Akron, Ohio	Chemical stoneware of all types
Laclede-Christy Clay Prod. Co., St. Louis, Mo.	Acid-proof brick
McLain Fire Brick Co., Pittsburgh	Acid-proof brick
McLeod & Henry Co., Troy, N.Y.	Acid-proof brick
Metropolitan Paving Brick Co., Canton, Ohio	Acid-proof brick
National Carbon Co., Inc., Cleveland	Carbon brick
Nukem Products Corp., Buffalo, N. Y.	Acid-proof brick
Parker-Russell Mining & Mfg. Co., St. Louis	Acid-proof brick
Patterson Foundry & Machine Co., East Liverpool, Ohio	Acid-proof lining blocks and grinding balls
Quigley Co., New York	Acid-proof brick
Robinson Clay Product Co. of N.Y., New York	Acid-proof and vitrified sewer tile
Southern Clay Mfg. Co., Chattanooga, Tenn.	Acid-proof brick
Thornton Fire Brick Co., Clarksburg, W.Va.	Acid-proof brick
Uhl Pottery Co., Huntington, Ind.	Acid-proof ceramics
United States Stoneware Co., Akron, Ohio	Chemical stoneware of all types

Table 10. Chemical Brick, Stoneware, and Cements—(*Continued*)
Makers of Cements and Putties for Acid-proof Brick and Stoneware

Manufacturer	Trade names	Compositions, applications, types
Anti-Hydro Waterproofing Co., Newark, N.J.	Anti-Hydro	Water-, acid-, alkali-, oil-resisting concrete mix
Atlas Lumnite Cement Co., New York	Lumnite cement	Water-, acid-, alkali-, oil-resisting concrete mix
Atlas Mineral Products Co., Mertztown, Pa.	Tegul-Vitrobond,-Mineralead, Tileset, Korez, G. K., others	Thiokol-sulfur-base chemical-setting silicate and resin-base and other cements for all acid-proof construction
Charlotte Chemical Labs., Charlotte, N.C.	Charlab, Acidpruf, Carolina	Standard and chemical-setting silicate cements; acid-proof putty
Chemical Construction Corp., New York	Acipruf	Acid-proof cement
Custodis Const. Co., New York	Penchlor, Asplit, Kabe mortar	See under Pennsylvania Salt Mfg. Co.
Electro-Chemical Supply & Engineering Co., Paoli, Pa.	Duro Standard, Special, Triple X	Silicate cements for all acid conditions; also water and steam
Filtros, Inc., East Rochester, N.Y.	Filtros	Acid-proof cement
General Ceramics Co., New York	Acidproof Nos. 1, 6, 7, 8	Silicate cements and linseed oil- and asphalt-base putties
B. F. Goodrich Rubber Co., Akron, Ohio	Plastikon	Rubber-base putty
The Haveg Corp., Newark, Del.	Havegit 41, 43	Self-hardening phenolic resin cements for acids
Johns-Manville Corp., New York	J-M	Caulking compound
M. W. Kellogg Co., New York		Acid-proof cement
Maurice A. Knight, Akron, Ohio	Knight	Silicate cements for strong acids
Nukem Products Co., New York	Basolit, Nu Mastic, Plasul-Basolit	Sulfur-base cements for acids
Paraffine Cos., San Francisco		Acid-proof cement
Patterson Foundry & Machine Co., East Liverpool, Ohio	Porox Cement	Silicate cement for strong acids
Pecora Paint Co., Philadelphia	Acitite, Acichlor, Cushion Putty	Slow- and quick-drying cements and elastic putties for acids
Pennsylvania Salt Mfg. Co., Philadelphia.	Penchlor, Asplit	Chemical-setting silicate cement; self-hardening resin cement
Philadelphia Quartz Co., Philadelphia.	"S" Brand Sodium Silicate	1:3.86 ratio sodium silicate for acid-proof cements
Quigley Co., New York	Acidproof Nos. 1 and 2	Silicate cements for acid gases and mineral acids
The Sullivan Co., Memphis, Tenn.	Acidol, Sulsilo	Pouring cements and premixed silicate cements for strong acids
Sauereisen Cements Co., Sharpsburg, Pa.	Insa-Lute, Acidproof	Standard and quick-setting silicate cements, etc.
United States Stoneware Co., Akron, Ohio.	U.S. Standard, Pre-Mixt, Calktite and others	Silicate cements of all types, resin cements, putties, etc.

Copyright by *Chem. & Met. Eng.*, September, 1940.

Table 11. Refractory Materials

Physical Properties of Refractory Materials

Complete revision of earlier Chem. & Met. Eng. data, compiled by L. J. Trostel, General Refractories Co., Baltimore, with additional material on kaolin super duty and insulating refractories supplied by the Babcock & Wilcox Co., New York.

Type of brick	Silica	High heat duty (No. 1) fireclay	Super duty fireclay	Super duty kaolin	Alumina-diaspore, 70 per cent Al_2O_3	Sillimanite (mullite)	Chrome	Unburned chrome[1]	Magnesite	Unburned magnesite[1]	Bonded silicon carbide (grade A)	Bonded fused alumina	Kaolin insul. refr. (2600 °F.)
Typical composition, %:													
SiO_2	96	50–57	52	52	22–26	35	6	5	3	5	7-9	8–10	57.7
Fe_2O_3	1	1.5–2.5	1	0.6	1–1.5	0.5	....	....	6	8.5	0.3-1	1-1.5	2.4
FeO	....	....	....	....	....	....	15	12					
Al_2O_3	1	36–42	43	45.4	68–72	62	23	18	2	7.5	2–4	85–90	36.8
TiO_2	....	1.5–2.5	2	1.7	3.5	1.5	....	....	....	....	1	1.5–2.2	1.5
CaO	2	....	....	0.1	....	....	....	....	3	2	....	....	0.6
MgO	....	....	....	0.2	....	....	17	32	86	64	....	....	0.5
Cr_2O_3	....	....	....	....	....	....	38	30	....	10			
SiC	....	....	....		....	....	....	....	....	....	85–90		
Flux[2]	....	1–3.5	2	....	1–1.5	0.5	....	....	....	....	1.5	0.8–1.3	
P.C.E. (with approx. equivalent temp., °F.)[3]	31–32 (3056–3092°)	31–33 (3056–3173°)	33–34 (3173–3200°)	34 (3200°)	36 (3290°)	37–38 (3308–3335°)	41+ (3578°+)	41+ (3578°+)	41+ (3578°+)	41+ (3578°+)	39 (3389°)	39+ (3389°+)	29–30 (2984–3002°)
Deformation under load,[4] % (at lb. per sq. in. and temp., °F., shown)	Shears (25 p.s.i. 2900°)	2.5–10* (25 p.s.i. 2460°)	2–4† (25 p.s.i. 2640°)	0.5† (25 p.s.i. 2640°)	1–4† (25 p.s.i. 2640°)	0.0–0.5† (25 p.s.i. 2640°)	Shears (28 p.s.i. 2740°)	Shears (28 p.s.i. 2955°)	Shears (28 p.s.i. 2765°)	Shears (28 p.s.i. 2940°)	0–1 (50 p.s.i. 2730°)	1 (50 p.s.i. 2730°)	0.3 (10 p.s.i. 2200°)
Resistance to spalling, % loss in appropriate A.S.T.M. panel test	Poor	5–20	0–4	No loss	No loss	No loss	Poor	Fair	Poor	Fair	Good	Good	Good
Permanent linear change on reheating[5] (after 5 hr. at temp., °F., shown)	(+)0.5–.8 (2640°)	(±)0–1.5 (2550°)	(±)0–1.5 (2910°)	(−)0.75–1 (2910°)	(−)2–4 (2910°)	(−)0–0.8 (2910°)	(−)0.5–1 (3000°)	(−)0.5–1.0 (3000°)	(−)1–2 (3000°)	(−)0.5–1.5 (3000°)	(+)2[6] (2910°)	(+)0.5 (2910°)	(−)0.2 (2600°)
Porosity (as open pores), %	20–30	15–25	12–15	18	34–38	20–25	20–26	10–12	20–26	10–12	13–28	20–26	75

Weight per brick (std. 9 in. straight), lb.	6.5	7.5	8.5	7.7	7.5	8.5	11	11.3	10	10.7	8-9.3	9-10.6	2.25
Specific heat (60-1200°F.)	0.23	0.23	0.23	0.22	0.23	0.23	0.20	0.21	0.27	0.26	0.20	0.20	0.22
Relative slag resistance:[7]													
Acid steel slag	Good	Fair	Fair	Fair	Good	Good	Poor	Poor	Poor	Poor	Good	Good	Poor
Basic steel slag	Poor	Poor	Poor	Poor	Fair	Fair	Good	Good	Good	Good	Good	Good	Poor
Mill scale	Fair	Poor	Fair	Good	Fair	Fair	Good	Good	Good	Good	Fair	Fair	Good
Coal ash slag	Poor	Fair	Fair	Fair	Fair	Fair	Fair	Fair	Good	Good	Good	Good	Poor

---

[1] Made by hydraulic pressing.

[2] Includes CaO + MgO + alkalies.

[3] Pyrometric cone equivalent; terms "fusion," "softening," "deformation," and melting points heretofore loosely used.

[4] Data marked (*) are from A.S.T.M. test C 16-36 with high heat duty time-temperature schedule; those marked (†) are from same test with super duty time-temperature schedule; others determined by other commonly used tests.

[5] (+) means expansion; (−) means shrinkage.

[6] Oxidizing atmosphere.

[7] Ratings affected somewhat by varying temperatures and type of atmosphere prevailing. Resistance to coal ash slag affected by furnace temperature as well as analysis and fusion point of slag.

Table 11. Refractory Materials—(*Continued*)
Representative Makers of Refractories and High Temperature Mortars

Manufacturer	Principal types
Acme Brick Co., Ft. Worth, Tex.	Firebrick and clay, high-temperature cements, plastic refractories
Alberene Stone Corp. of Va., New York	Refractory linings
American Crucible Co., Shelton, Conn.	Graphite crucibles, silica and mullite refractories
Armstrong Cork Co. Lancaster, Pa.	Refractory insulating brick, high-temperature mortars, 1800 deg. insulating block and cement
Atlas Lumnite Cement Co., New York	Cement for refractory, heat-resisting, and insulating concrete
Babcock & Wilcox Co., New York	Glass plant refractories, high-temperature mortars, plastic refractories, insulating and kaolin refractories
Bartley Crucible & Refr. Co., Trenton, N.J.	Graphite crucibles, firebrick, magnesite refractories
Betson Plastic Fire Brick Co., Buffalo	High-temperature mortars, plastic refractories
Botfield Refractories Co., Philadelphia	Chrome, firebrick, plastic refractories, high-temperature mortars
Philip Carey Co., Lockland, Ohio	High-temperature mortars, insulations
Carborundum Co., Perth Amboy, N.J.	Silicon carbide, aluminum oxide, mullite and fused cast refractories and high-temperature mortars
Champion Spark Plug Co., Detroit	Sillimanite plastic refractories, electric furnace refractories
Corhart Refractories Co., Louisville, Ky.	High-temperature mortars, electro-cast mullite refractories
Corundite Refractories, Inc., Massillon, Ohio	Firebrick, high-temperature mortars, plastic refractories, alumina, silica and mullite refractories
Denver Fire Clay Co., Denver	Firebrick, diaspore and sillimanite refractories, high-temperature mortars, plastic refractories, fireclays
W. S. Dickey Clay Mfg. Co., Kansas City, Mo.	Fireclay refractories
Joseph Dixon Crucible Co., Jersey City, N.J.	Graphite crucibles
Ehret Magnesia Mfg. Co., Valley Forge, Pa.	High-temperature mortar
Electro Refrs. & Alloys Corp., Buffalo	Mullite, fused alumina, silicon carbide and magnesia refractories
Emsco Refractories Co., Vernon, Calif.	Firebrick, glass plant refractories, high-temperature mortars
The Exolon Co., Blasdell, N.Y.	Silicon carbide, alumina refractories
General Abrasive Co., Niagara Falls, N.Y.	Alumina and silicon carbide
General Ceramics Co., New York	Special refractories
General Refractories Co., Philadelphia	Fired and unfired chrome and magnesite, firebrick, high-temperature mortars, plastic and silica refractories
Gladding, McBean & Co., Los Angeles	Firebrick, insulating brick and plastics, high-temperature mortars, plastic refractories, fireclays
A. P. Green Fire Brick Co., Mexico, Mo.	Firebrick, insulating firebrick, high-temperature mortars, plastic and castable refractories, fireclays
Harbison-Walker Refrs. Co., Pittsburgh	Refractories of most types including regular and super fireclay, high-alumina, silica, chrome, magnesite, Fosterite brick; clays; insulating firebrick and mortars; high-temperature mortars
Haws Refractories Co., Johnstown, Pa.	Firebrick of all kinds, silica brick, fireclays
Illinois Clay Products Co., Joliet, Ill.	Firebrick, high-temperature mortars, insulating cements coatings and brick
Ironton Fire Brick Co., Ironton, Ohio	Fireclay refractories, refractory cements, fireclays
Johns-Manville, New York	Bonding mortars, castables, ramming mixtures, plastic and insulating refractories
Laclede-Christy Clay Prod. Co., St. Louis	Firebrick, high-temperature mortars, plastic refractories, glass plant refractories, fireclays
E. J. Lavino & Co., Philadelphia	Chrome and magnesite refractories, high-temperature mortars, silica refractories, fireclays
Massillon Refractories Co., Massillon, Ohio	Firebrick, high-temperature mortars. plastic refractories, special compositions
McLain Fire Brick Co., Pittsburgh	Various

Table 11. Refractory Materials—(*Continued*)

Representative Makers of Refractories and High-temperature Mortars

Manufacturer	Principal types
McLeod & Henry Co., Troy, N.Y.	Firebrick, high-temperature mortars, plastic-refractories, fireclays
Mullite Refractories Co., Shelton, Conn.	High-temperature mortars, plastic refractories, mullite refractories
National Carbon Co., Inc., Cleveland	Carbon refractories
Niles Fire Brick Co., Niles, Ohio	Firebrick, insulating refractories
North American Refrs. Co., Cleveland	Fireclay, super, insulating and silica brick, high-temperature mortars, plastic refractories, fireclays
Norton Co., Worcester, Mass.	High-temperature mortars, silicon carbide, fused alumina and magnesia, raw materials, cements, refractory shapes
Pacific Clay Products Co., Los Angeles	Plastic refractories
Pyro Clay Products Co., Oak Hill, Ohio	Glass plant refractories
Quigley Co., Inc., New York	Firebrick, insulating refractories, super firebrick, high-temperature mortars, plastic refractories
Ramtite Co., Chicago	High-temperature mortars
Refractory & Insulation Corp., New York	High-temperature mortars
Robinson Clay Product Co. of N.Y., New York	High-temperature mortars, firebrick and clay, insulating refractories
Ross Tacony Crucible Co., Philadelphia	Graphite crucibles and stopper heads, magnesite refractories
St. Louis Fire Brick & Insulation Co., Huntington Park, Calif.	Various
Seaboard Refrs. Co., Perth Amboy, N.J.	Firebrick, high-temperature mortars, plastic and insulating refractories, silicon carbide and mullite refractories.
Chas. Taylor Sons Co., Cincinnati	Firebrick, glass plant and insulating refractories, sillimanite
The United States Stoneware Co., Akron, Ohio	Plastic and castable refractories
M.D. Valentine & Bro. Co., Woodbridge, N.J.	Firebrick
Vitrefrax Corp., Los Angeles	Glass plant refractories firebrick, high-temperature mortars, plastic refractories, fireclays

Copyright by *Chem. & Met. Eng.*, September, 1940.

Table 12. Structural Carbon and Graphite

Physical Characteristics of Carbon and Graphite Products

Material and Form	Apparent density	Weight, lb./cu. ft.	Strength, lb. per sq. in.			Elastic modulus, lb./sq. in. (multiply by 10^5)	Specific resistance, ohms./in.^{3}	K (thermal expansion) (see Note)	Thermal conductivity, B.t.u./hr./sq. ft./°F./per ft.
			Tensile	Compressive	Transverse				
Carbon cylinders:									
8 in. dia.	1.54	96.0	660	2,920	1,320	5.5	0.0013	13	6.0
10–14 in. dia. inc.	1.525	95.0	470	2,120	950	5.4	0.0013	12	6.0
17–24 in. dia. inc.	1.54	96.0	400	2,200	790	5.4	0.0014	13	6.0
30–40 in. dia. inc.	1.54	96.0	400	1,910	810	4.3	0.0026	12	6.0
Carbon blocks:									
4 × 4 in. to 6 × 6 in. inc.	1.57	97.8	840	4,100	1,670	9.4	0.0018	14	4.0
6 × 6 in. to 20 × 20 in. inc.	1.55	96.7	500	2,140	990	7.1	0.0016	15	4.0
15 × 30, 24 × 30 & 24 in. sq.	1.54	96.0	400	1,910	810	4.3	0.0026	12	4.0
Carbon tubes:									
½–4 in. i. d. inc.	1.51	94.2	885	10,200	2,700	21.0	0.0014	15	3.0
5–10 in. i. d. inc.	1.49	93.0	980	8,140	2,550	17.0	0.0016	21	3.0
Carbon brick:									
Dependent on application	1.56	96.7–97.8	970–1,530	5,340–8,320	1,950–3,070	8.9–10.3	0.0015–0.0016	13–14	3.0
Graphite cylinders:									
To 5⅛ in. dia. inc.	1.56	97.3	760	3,050	1,750	8.8	0.00036	5–12	84.0
6–12 in. dia. inc.	1.55	96.7	610	3,420	1,810	8.0	0.00037	6–12	79.0
14 in. dia.	1.53	95.3	580	3,180	1,490	6.7	0.00039	8–12	70.0
16 & 18 in. dia.	1.53	95.3	500	3,180	1,490	6.7	0.00040	8–12	70.0
20 in. dia.	1.53	95.3	440	3,180	1,490	6.7	0.00040	8–12	70.0
Graphite squares and slabs:									
To 5 in. thick inc.	1.56	97.3	700	3,050	1,750	8.8	0.00036	5–12	94.0
6 in. thick to 144 sq. in.	1.55	96.7	700	3,420	1,810	8.0	0.00037	6–12	84.0
Over 144 sq. in. section	1.53	95.3	570	3,180	1,490	6.7	0.00039	8–12	79.0
Graphite tubes:									
½–4 in. i. d. inc.	1.68	104.7	780	4,550	2,820	14.0	0.0003	12	94.0
5–10 in. i. d. inc.	1.67	104.0	870	5,100	2,980	13.0	0.0003	12	84.0
Graphite brick, standard sizes	1.56	97.3	700	3,050	1,750	8.8	0.00036	5–12	84.0
Karbate No. 1 (impervious carbon):									
Tubes ½–2 in. i. d. inc.	1.77	110.0	1,700	10,500	4,170	29.0	0.00164	27	3.0
Over 2 in. i. d.	1.76	110.0	2,000	10,500	4,640	26.0	0.0016	33	2.8
Karbate No. 2 (impervious graphite):									
Tubes ½–2 in. i. d. inc.	1.86	116.0	2,600	8,900	4,650	23.0	0.00034	23	85.0
Over 2 in. i. d.	1.91	119.0	2,350	10,500	4,980	21.0	0.00033	24	75.0
Carbocell (porous carbon):*									
Grade C (finest)	1.34	84.0	500	1,530	2,700	>1.2	0.0020	6	3.0
Grade 60	1.05	69.0	190	600	850	>1.2	0.0070	27	1.5
Grade 50	1.05	69.0	180	500	830	>1.2	0.0070	27	1.4
Grade 40	1.04	69.0	120	320	900	>1.2	0.0057	27	1.0
Grade 30	1.04	69.0	100	250	770	>1.2	0.0070	27	1.0
Grade 20	1.03	68.0	90	240	700	>1.2	0.0070	27	1.0
Grade 10	1.03	68.0	80	160	300	>1.2	0.0080	27	1.0
Graphicell (porous graphite):*									
Grade C (finest)	1.35	84.0	600	1,080	1,680	....	0.00045	6	60.0
Grade 60	1.05	69.0	110	250	500	....	0.0012	21	50.0
Grade 50	1.05	69.0	110	250	500	....	0.0012	21	45.0
Grade 40	1.04	69.0	100	190	500	....	0.0013	21	45.0
Grade 30	1.04	69.0	80	200	520	....	0.0017	21	40.0
Grade 20	1.03	68.0	60	140	310	....	0.0020	21	30.0
Grade 10	1.03	68.0	50	140	270	....	0.0020	22	20.0

Note: Coefficient of thermal expansion per degree: To temperature t°F = $[K + 0.0039t\ (°F)]\ 10^{-7}$; to temperature t°C = $[1.8\ K + 0.007t\ (°C)]\ 10^{-7}$. Carbon graphite products are resistant to most acids and alkalies.

* See following table for additional data.

Table 12. Structural Carbon and Graphite—(*Continued*)

Physical Properties of Porous Carbon and Porous Graphite

Material and Form	Porosity, per cent	Average pore diameter		Filter action, minimum diameter of particle retained, in.	Average water* permeability at 5 lb./sq. in. pressure, gal./sq. ft./min.	Average air† permeability at 2 in. H_2O pressure, cu. ft./sq. ft./min.
		Inches	Microns			
Carbocell						
Grade C	36	0.0002	5	....	0.30	
Grade 60	48	0.0013	33	0.00047	14.0	
Grade 50	48	0.0019	48	0.00079	30.0	
Grade 40	48	0.0027	69	0.00098	45.0	4.0
Grade 30	48	0.0039	99	0.00173	80.0	8.5
Grade 20	48	0.0055	140	0.00300	120.0	17.0
Grade 10	48	0.0075	190	0.00590	175.0	33.0
Graphicell						
Grade C	36	0.0002	5	....	0.30	
Grade 60	48	0.0013	33	0.00047	14.0	
Grade 50	48	0.0019	48	0.00079	30.0	
Grade 40	48	0.0027	69	0.00098	45.0	4.0
Grade 30	48	0.0039	99	0.00173	80.0	8.5
Grade 20	48	0.0055	140	0.00300	120.0	17.0
Grade 10	48	0.0075	190	0.00590	175.0	33.0

Note: Carbocell can be treated so as to be wettable for use in caustic filtration. Both are resistant to most acids and alkalies.

* Water at 70°F., 1 in. thick plate.

† Air at 70°F. and 760 mm. Hg. pressure, 15 per cent relative humidity, 1 in. thick plate.

Makers of Structural Carbon and Graphite Products

Manufacturer	Products
Acheson Graphite Corp., New York	Graphite electrodes and various shapes
International Graphite & Electrode Corp., St. Mary's, Pa.	Graphite electrodes and various shapes
National Carbon Co., Inc., Carbon Sales Div., Cleveland	Industrial carbon and graphite products such as: brushes, brick, pipe and fittings, heat exchangers, tubes, tower sections, lighting carbons, welding electrodes, etc.
Speer Carbon Co., St. Mary's, Pa.	Carbon and graphite brick, plates, blocks, tubes, cylinders, bushings, shapes
Stackpole Carbon Co., St. Mary's, Pa.	Various carbon and graphite products

Note: Manufacturers of graphite crucibles are listed under "Refractories."

Table 13. Plastic Molding Materials

Representative Makers of Molding Powders and Other Plastic Materials

Material	Manufacturer	Type
Aladdinite	Aladdinite Co., Newark, N.J.	Casein
Alvar (and others)	Shawinigan Products Corp., New York	Vinyl resin
Ameroid	American Plastics Corp., New York	Casein
Aqualite	National Vulcanized Fibre Co., Wilmington, Del.	Laminated phenolic
Atlastiseal	Atlas Mineral Prods. Co. of Pa., Mertztown, Pa.	Tank lining
Bakelite	Bakelite Corp., New York;* unit of Union Carbide & Carbon Corp.	Molding and cast phenolics, urea-form., styrene, cell. ac.
Beckacite (others)	Reichhold Chemicals, Inc., Detroit	Lacquer resins
Beetle	Beetle Prods. Div., American Cyanamid Co., New York*	Urea-formaldehyde
Benalite, -loid	Masonite Corp., Laurel, Miss.	Lignin plastics
Butacite	E. I. du Pont de Nemours & Co., Wilmington, Del.*	Vinyl resin
Catalin	Catalin Corp., New York	Molding and cast phenolics
Celeron	Continental Diamond Fibre Co., Newark, Del.	Molding phenolic
Celluloid	Celluloid Corp., New York	Cellulose nitrate
Crystalite	Rohm & Haas Co., Philadelphia	Acrylic resin
Cumar	Barrett Co., New York	Coumarone-indene
Dilecto	Continental Diamond Fibre Co., Newark, Del.	Laminated phenolic
Duraloy	Detroit Paper Prods. Corp., Detroit	Laminated phenolic
Durez	General Plastics, Inc., No. Tonawanda, N.Y.*	Molding phenolic
Durite	Durite Plastics, Inc., Philadelphia	Phenolic, phenol-furfural
Duroid	Atlas Mineral Prods. Co. of Pa., Mertztown, Pa.	Bulk and sheet plastics
Ethocel, -foil	Dow Chemical Co., Midland, Mich.	Ethyl cellulose
Formica	Formica Insulation Co., Cincinnati	Laminated phenolic
Harvel	Irvington Varnish & Insulator Co., Irvington, N.J.	Cashew nut derivative
Haveg	Haveg Corp., Newark, Del.	Phenolic-base chemical equipment
Hercules	Hercules Powder Co., Wilmington, Del.*	Ethyl cellulose, cellulose nitrate, acetate
Heresite	Heresite & Chemical Co., Manitowoc, Wis.*	Molding phenolic
Indur	Reilly Tar & Chemical Co., Indianapolis	Molding phenolic
Insurok	Richardson Co., Chicago	Phenolic and urea-formaldehyde
Lamicoid (and others)	Mica Insulator Co., New York	Laminated phenolic
Lauxite	I. F. Laucks, Inc., Lockport, N.Y.	Plywood resin bond
Lucite	E. I. du Pont de Nemours & Co., Wilmington, Del.	Methacrylate resin
Lumarith	Celluloid Corp., New York	Cellulose acetate
Makalot	Makalot Corp., Boston*	Molding phenolic
Marblette	Marblette Corp., Long Island City, N.Y.	Cast phenolic
Masuron	John W. Masury & Son, Brooklyn, N.Y.	Cellulose acetate
Micarta	Micarta Div., Westinghouse Elec. & Mfg. Co., Trafford, Pa.	Laminated phenolic and urea-formaldehyde
Monsanto	Plastics Div., Monsanto Chem. Co., Springfield, Mass.*	Cast phenolic, cellulose acetate and nitrate, vinyl, arochlor resins
Neville	Neville Co., Neville Is. P. O., Pittsburgh	Hydrocarbon and coumarone-indene resins
Nixonite, -oid	Nixon Nitration Works, Nixon, N.J.	Cellulose acetate, nitrate
Ohmoid	Wilmington Fibre Specialty Co., Wilmington, Del.	Laminated phenolic
Panelyte	Panelyte Corp., New York	Laminated phenolic
Phenolite	National Vulc. Fibre Co., Wilmington, Del.	Laminated phenolic
Plaskon	Plaskon Co., Toledo, Ohio	Urea-formaldehyde
Plastacele	E. I. du Pont de Nemours & Co., Plastics Dept., Arlington, N.J.	Cellulose acetate
Plexiglas, -gum	Rohm & Haas Co., Philadelphia	Acrylic resins
Pyralin	E. I. du Pont de Nemours & Co., Wilmington, Del.	Cellulose nitrate
Rauzene (and others)	U. S. Industrial Alcohol Co., New York	Varnish resin

* Also makes varnish and lacquer resins.

Table 13. Plastic Molding Materials—(*Continued*)
Representative Makers of Molding Powders and Other Plastic Materials

Material	Manufacturer	Type
Resinox	Resinox Corp., New York	Molding phenolic
Resoglaz	Advance Solvents & Chem. Co., New York	Styrene resin
Spauldite	Spaulding Fibre Co., Tonawanda, N.Y.	Laminated phenolic
Styron	Dow Chemical Co., Midland, Mich.	Styrene resin
Synthane	Synthane Corp., Oaks, Pa.	Laminated phenolic
Tego	Resinous Products & Chem. Co., Philadelphia*	Phenolic plywood bond
Tenite	Tennessee Eastman Corp., Kingsport, Tenn.	Cellulose acetate
Textolite	General Electric Co., Plastics Dept., Pittsfield, Mass.*	Molding, laminated phenolics
Ucinite	United-Carr Fastener Corp., Cambridge, Mass.	Laminated phenolic
Unyte	Plaskon Co., Toledo, Ohio	Urea-formaldehyde
Victron	Naugatuck Chemical Co., New York	Styrene resin
Vinylite	Carbide & Carbon Chem. Corp., New York*	Vinyl resin
Vistanex	Advance Solvents & Chem. Co., New York	Hydrocarbon resin

* Also makes varnish and lacquer resins.

Table 14. Physical Properties of Rubber and Rubberlike Materials

Property	Ameripol, oil-resisting, soft	Ameripol, tire type, soft	Ameripol hard	Chemigum tire cpd.	Koroseal		Neoprene (all forms)	Perbunan	Pliolite No. 40	Resistoflex molded & ext.	Natural Rubber	
					Hard	Soft					Hard	Soft
Specific gravity	0.99–1.6	0.96–1.20	1.1–1.3	1.19	1.3–1.4	1.2–1.3	1.27–1.30	0.96	1.06	1.26	1.17–1.18	0.93–1.17
Tensile strength, lb. per sq. in.	1,000–4,500	1,000–4,000	4,000–10,000	4,000	2,000–9,000	500–2,500	1,000–4,000	500–5,000	4,000–5,000	2,000–5,000	4,000–11,000	1,000–6,000
Hardness, Shore durometer	15–90	50–80	80–100	65	80–100	30–80	15–95	30–90	....	....	70–100	30–80
Maximum temperature for use, °F.	260–300	240–280	260–300	450+	212	190	300	300	160–248	250	220	150–180
Dielectric strength, volts per mm.	....	....	....	....	30,000–50,000	15,000–30,000	....	....	....	6,000–10,000	40,000–150,000	40,000–55,000
Effect of heat	Stiffens slightly		....	Stiffens	Softens	Softens	Stiffens slightly	Stiffens	Softens	Softens	Softens	Softens and deteriorates
Abrasion resistance	Excellent		Good	Excellent	Good	Good	Equal to rubber	Excellent	....	Good	Good	Excellent
Effect of sunlight	Discolors, cracks less than rubber		....	Deteriorates	None	None	None	Slight	None	None	Discolors	Deteriorates
Effect of aging	Highly resistant, stiffens slightly		None	None	None	None	Better than rubber	Highly resistant	None	None	None	Highly resistant
Machining qualities	Can be ground		Excellent	Excellent	Good	Can be ground	Can be ground	Can be ground	....	....	Excellent	Can be ground

Table 14. Physical Properties of Rubber and Rubberlike Materials—*(Continued)*

Representative Makers of Industrial Rubber Products and Rubberlike Materials

Manufacturer	Products
American Hard Rubber Co., New York	Hard and soft rubber, neoprene, and Thiokol linings, pipe, fittings, shapes, pails, pumps, rubber, paint, etc.
Atlas Mineral Products Co. of Pa., Mertztown, Pa.	Rewbon seamless rubber linings and Zerok synthetic resin linings
Boston Woven Hose & Rubber Co., Boston	Conveyor and transmission belts, hose, mechanical rubber goods
Crane Packing Co., Chicago	Rubber and synthetic rubber packings
Custodis Construction Co., New York	Custoplast soft rubber and neoprene rubber tank linings
Dayton Rubber Mfg. Co., Dayton, Ohio	Oil-proof rubber belts, transmission belting, and synthetic rubber products
E. I. du Pont de Nemours & Co., Rubber Chemicals Div., Wilmington, Del	Neoprene polymerized chloroprene rubber
Firestone Tire & Rubber Co., Akron, Ohio	Perbunan synthetic rubber
Garlock Packing Co., Palmyra, N.Y	Rubber packings, belting, and molded goods
Gates Rubber Co., Denver	Transmission and conveyor belting
L. H. Gilmer Co., Tacony, Philadelphia	Transmission belting
B. F. Goodrich Co., Akron Ohio	Acid- and abrasion-resistant linings, hose, conveyor and transmission belting, packings, hard-rubber pipe and molded goods, rubber paints, Koroseal plastic, Anode process, Ameripol tires and goods
Goodyear Tire & Rubber Co., Akron, Ohio	Hose, conveyor and transmission belting, packings, linings, mechanical rubber goods, Pliolite modified rubber plastic, Chemigum
Hewett Rubber Corp., Buffalo	Hose, transmission and conveyor belting, packings
Hydrocarbon Chemical & Rubber Co., Akron, Ohio	Synthetic rubbers (Ameripol)
Jenkins Bros. Rubber Div., Bridgeport, Conn	Mechanical rubber goods, packings, molded and extruded shapes
The Osborn Mfg. Co., Johns Conveyor Div., Cleveland	Johns rubber and synthetic rubber "moving pipeline" conveyors
Maurice A. Knight, Akron, Ohio	Rubber and neoprene-lined drums, Pyroflex resin-base tank lining
Linear Packing & Rubber Co., Philadelphia	Rubber and synthetic rubber packings
Luzerne Rubber Co., Trenton, N.J	Hard-rubber pipe, fittings, valves, shapes, tanks, rayon apparatus and other equipment
Manhattan Rubber Mfg. Div., Passaic, N.J	Transmission and conveyor belting, blocks, hose, piping, rolls, brake lining, bearings
Miller Rubber Co., Akron, Ohio	Hose, molded products, linings, and coverings
Paramount Rubber Service, Inc., Detroit, Mich	Seamless rubber linings, rubber paint, molds, coatings, and insulations
Resistoflex Corp., New York	Rubberlike oil-resisting resin, various shapes
Self-Vulcanizing Rubber Co., Chicago	Liquid and plastic rubber self-vulcanizing coatings and lining materials
Standard Oil Development Co., Elizabeth, N.J	Perbunan synthetic rubber
Jos. Stokes Rubber Co., Trenton, N.J	Molded hard-rubber products of all kinds
Thermoid Rubber Co., Trenton, N.J	Hose, belting, packing, mechanical rubber goods
Thiokol Corp., Yardville, N.J	Thiokol olefine polysulphide synthetic rubber—crude sheet, molding powder, and liquid dispersions
U.S. Rubber Co., New York	Rubber-lined pipe, hard, semihard and soft linings, abrasion-resistant linings, hard-rubber pipe, acid hose, packings, mechanical rubber goods and belting
U.S. Stoneware Co., Akron, Ohio	Resilon and Tygon polymer tank linings, rubber latex linings
Vulcanized Rubber Co., New York	Hard and semihard rubber molded products

Copyright by *Chem. & Met. Eng.*, September, 1940.

Table 15. Vulcanized Fiber[*]

Marketed forms: S = sheet, R = rod, T = tubes	S.R.T.
Thickness, $\frac{1}{1000}$ in.	4 up
Forming properties	Good
Machining qualities	Good
Colors	Red, brown, white, gray, black, olive
Effect of heat	Stable to charring
Effect of water	Swells and softens. Warps on drying. Not recommended
Effect of mineral, animal and vegetable oils	Very slight absorption
Specific gravity	1.2–1.5
Specific volume, cu. in. per lb.	23–18.5
Tensile strength, lb. per sq. in.	8,000–16,000
Breakdown voltage, 60 cycle ($\frac{1}{16}$-in. sample) volts per mil	175–500

[*] From *Chem. & Met. Eng.*, December, 1932.

Table 16. Wood for Chemical Equipment

Physical Properties of Woods

	Cypress	Douglas fir (coast)	L. L. Y. pine	Red-wood	Sugar maple	White oak
Lb. per cu. ft. (12% moisture)	32	34	41	28	44	47
Tensile str.,[*] lb. per sq. in. (12% moisture)	7,200	8,100	9,300	6,900	9,500	7,900
Compressive str.,[*] lb. per sq. in. (12% moisture)	4,740	6,450	6,150	4,560	5,390	4,350
Thermal cond., B.t.u. per sq. ft., hr., (°F./in)	0.83	0.77	0.96	0.76	1.16	1.22
Hardness	Med.	Med.	Hard	Mod. hard	Med.	Hard

[*] At proportional limit, in static bending, and compression parallel to grain, respectively.

Condition of Woods after 31 Days Immersion in Cold Solutions[*]

Examined after 7 days drying

	Fir	Oak	Oregon pine	Yellow pine
Hydrochloric acid, 5%	NAC	NAC	NAC	SS
Hydrochloric acid, 10%	NAC	NAC	NAC	SS
Hydrochloric acid, 50%	SS,SB,SWF	SS,WF	S,WF	S,WF
Sulfuric acid, 1%	NAC	NAC	NAC	SS
Sulfuric acid, 5%	SS	SS	SS	SS
Sulfuric acid, 10%	S,FSD	S,FSD	S,FSD	S,FSD
Sulfuric acid, 25%	SSp,FSD	SSp,FSD	SSp,FSD	SSp,FSD
Caustic soda, 5%	S,NAC	MSh,SWp	SS	SS,FSD
Caustic soda, 10%	S,FSD	MSh,WF,Horny	SS	SS,SB,FSD
Alum, 13%	NAC	NAC	NAC	NAC
Sodium carbonate, 10%	SB GC	NAC	GC	SB,GC
Calcium chloride, 25%	NAC	NAC	NAC	NAC
Common salt, 25%	NAC	NAC	NAC	SS,GC
Water	NAC	NAC	NAC	NAC
Sodium sulfide	SS,SB	MSh,WF	SB	SB

[*] The two tables describing the condition of eight varieties of woods used for tanks and other chemical-resistant uses are based on a report of James K. Stewart, consulting chemist, to the Mountain Copper Co. Martinez, Calif. Tests were conducted on samples 1 × 4 × ¼ in. in size, seasoned and chosen so as to be as nearly as possible in the same physical condition as the woods would be when used for equipment construction. Results of the tests are described by terms explained in the key on p. 2165.

Table 16. Wood for Chemical Equipment—(*Continued*)

Condition of Woods after 31 Days Immersion in Cold Solutions*

Examined after 7 days drying

	Spruce	Redwood	Maple	Cypress
Hydrochloric acid, 5%	SS	SS	NAC	NAC
Hydrochloric acid, 10%	SS	SS	NAC	NAC
Hydrochloric acid, 50%	S,WF	S,WF	S,WF	S,WF
Sulfuric acid, 1%	SS	NAC	NAC	SS,SB
Sulfuric acid, 5%	SS,SB	SS,SB	NAC	SS,SB
Sulfuric acid, 10%	S,FSD	S,FSD	S,FSD	S,FSD
Sulfuric acid, 25%	SSp,FSD	SSp,FSD	SSp,FSD	SSp,FSD
Caustic soda, 5%	SSp,FSD	SSp,FSD	MSh	SSp,FSD
Caustic soda, 10%	SS,SB,FSD	SS,SB,FSD	MSh	S,SB,FSD
Alum, 13%	NAC	NAC	NAC	NAC
Sodium carbonate, 10%	SB,GC	SB,GC	GC	SB,GC
Calcium chloride, 25%	NAC	NAC	NAC	NAC
Common salt, 25%	SS.GC	SB.GC	NAC	NAC
Water	NAC	NAC	NAC	NAC
Sodium sulfide	SB	SB	MSh,FSD	FSD

Condition of Woods after 8 Hr. Boiling in Solutions*

Examined after 7 days drying

	Fir	Oak	Oregon pine	Yellow pine
Hydrochloric acid, 10%	SB,S	FSD	FSD	FSD
Hydrochloric acid, 50%	FD,Ch,B,S,NG	FD,Ch,B,S,NG	FD,Ch,B,S,NG	FD,Ch,B,S,NG
Sulfuric acid, 4%	SB,GC	SB,GC	SB,GC	SB,GC
Sulfuric acid, 5%	SS,GC	SB,GC	SB,GC	SB,GC
Sulfuric acid, 10%	SS,GC	BFD,Wpd,NG	Sp,FD,NG	B,Sp,FD,NG
Caustic soda, 5%	SS	MSh	S	GC
Alum, 13%	SB,GC	NAC	NAC	SB,GC
Sodium carbonate, 10%	SB,GC	GC	GC	GC
Calcium chloride, 25%	SB,GC	SB,SS,GC	NAC	SB,GC
Common salt, 25%	NAC	NAC	NAC	SB,GC
Water	NAC	NAC	NAC	SB,GC

	Spruce	Redwood	Maple	Cypress
Hydrochloric acid, 10%	FSD	FSD	FSD	FSD
Hydrochloric acid, 50%	FD,Ch,B,S,NG	FD,Ch,B,S,NG	FD,Ch,B,S,NG	FD,Ch,B,S,NG
Sulfuric acid, 4%	SB,GC	SB,GC	SB,GC	SB,GC
Sulfuric acid, 5%	SB,FSD	SB,GC	SB,GC	SB,FSD
Sulfuric acid, 10%	B,Sp,FD,NG	SB,FSD	SB,FSD	B,FD
Caustic soda, 5%	S,GC	S,GC	Sh	SSp
Alum, 13%	SB,GC	SB,GC	NAC	SB,GC
Sodium carbonate, 10%	GC	GC	GC	SB,GC
Calcium chloride, 25%	SB,GC	NAC	NAC	SB,GC
Common salt, 25%	NAC	SB,GC	NAC	NAC
Water	NAC	NAC	NAC	NAC

* The two tables describing the condition of eight varieties of woods used for tanks and other chemical-resistant uses are based on a report of James K. Stewart, consulting chemist, to the Mountain Copper Co., Martinez, Calif. Tests were conducted on samples 1 × 4 × ¼ in. in size, seasoned and chosen so as to be as nearly as possible in the same physical condition as the woods would be when used for equipment construction. Results of the tests are described by terms explained in the following key:

Abbreviation Key:
B —Brittle
Ch —Charred
FD —Fiber disintegrated
FSD—Fiber slightly disintegrated
GC —Good condition
MSh—Much shrunk
NAC—No apparent change
NG —No good
S —Softer
SB —Slightly brittle
Sh —Shrunk
Sp —Spongy
SS —Slightly softer
SSp —Slightly spongy
SWF—Slightly weakened fiber
SWp—Slightly warped
WF —Weakened fiber
Wpd—Warped

Representative Makers of Wood Tanks and Pipe for Chemical Applications

Acme Tank Co., New York
Alert Pipe & Supply Co., Bay City, Mich.
Atlantic Tank Corp., North Bergen, N.J.
Axtell Co., Fort Worth, Tex.
Baltimore Cooperage Tank & Tower Co., Baltimore
Black, Sivalls & Bryson, Inc., Oklahoma City, Okla.
C. F. Braun & Co., Alhambra, Calif.
W. E. Caldwell Co., Louisville, Ky.
Caspar Lumber Co., San Francisco
Challenge Co., Batavia, Ill.
A. J. Corcoran, Inc., Jersey City, N.J.
Cypress Tank Co., Shreveport, La.
Dempster Mill Mfg. Co., Beatrice, Neb.
Drane Tank Co., Fort Worth, Tex.
Drummond Mfg. Co., Louisville, Ky.
Dunck Tank Works, Inc., Milwaukee
G. Elias & Bro., Buffalo
Engle Tank Co., Chicago
Federal Pipe & Tank Co., Seattle, Wash.
Fibre Conduit Co., Orangeburg, N.Y.
Fleming Tank Co., Pittsburgh
Fluor Corp., Ltd. Los Angeles
Foster-Wheeler Corp., New York
General Tank Corp., Kearny, N.J.
Amos H. Hall & Sons, Philadelphia
Hammond & Little River Redwood Co., Samoa, Calif.
Harry Cooling & Equipment Co., Doylestown, Pa.
Hauser-Stander Tank Co., Cincinnati
Henderson Bros. Co., Waterbury, Conn.
R. R. Howell & Co., Minneapolis
James Hunter Machine Co., North Adams, Mass.
Johnson & Carlson, Chicago
Kalamazoo Tank & Silo Co., Kalamazoo, Mich.
Lille-Hoffman Cooling Towers, Inc., St. Louis
Lincoln Tank Co., Shreveport, La.
Marley Co., Kansas City, Kan.
Michigan Pipe Co., Bay City. Mich. (Pipe)
National Tank Co., Tulsa, Okla.
National Tank & Pipe Co., Portland, Ore.
New England Tank & Tower Co., Everett, Mass.
Pacific Cooperage Co., Portland, Ore.
Pacific Tank & Pipe Co., Oakland, Calif.
Pacific Wood Tank Corp., San Francisco
Parkersburg Rig & Reel Co., Parkersburg, W. Va.
Fred C. Pfeil, Inc., Buffalo, N.Y.
J. F. Pritchard & Co., Kansas City, Mo.
Redwood Mfrs. Co., San Francisco
Wm. B. Scaife & Sons Co., Oakmont, Pa.
Schubert-Christy Corp., St. Louis
A. T. Stearns Lumber Co., Boston
Treadwell Construction Co., Midland, Pa.
Union Lumber Co., San Francisco
U.S. Wind Engine & Pump Co. Batavia, Ill.
Wendnagel Co., Chicago
C. H. Wheeler Mfg. Co., North Phila., Pa.
G. Woolford Wood Tank Co., Darby, Pa.
A. Wyckoff & Son Co., Elmira, N.Y.

SECTION 19

HIGH-PRESSURE TECHNIQUE

BY

Hood Worthington, M. S., Experimental Station, E. I. duPont de Nemours & Co.; Member, American Chemical Society, American Association for the Advancement of Science, American Institute of Chemical Engineers.

CONTENTS

HIGH-PRESSURE TECHNIQUE

REFERENCES: Norton, "Creep of Steels at High Temperatures," McGraw-Hill, New York, 1929. Ernst, Reed, and Edwards, A Direct Synthetic Ammonia Plant, *Ind. Eng. Chem.*, **17**, 775 (1925). Ernst, Equipment for High Pressure Reactions, *Ind. Eng. Chem.*, **18**, 664 (1926). Dilley and Edwards, A High-pressure Gas-compression System, *U. S. Dept. Agr., Circ.* 61, 1929. Special High-pressure High-temperature Issue, *Chem. Met. Eng.*, September, 1930. Bone, Newitt, and Townend, "Gaseous Combustion at High Pressures," Longmans, London, 1929. Tables of Chemical Compositions, Physical and Mechanical Properties and Corrosion-resistant Properties of Corrosion-resistant and Heat-resistant Alloys, Reprint from *Proc. Am. Soc. Testing Materials*, vol. 30, Pt. I, 1930. Bridgman, "The Physics of High Pressure," Macmillan, New York, 1931. Tongue, "The Design and Construction of High Pressure Chemical Plant," Chapman and Hall, London, 1934. Newitt, "The Design of High Pressure Plant and the Properties of Fluids at High Pressures," Oxford University Press, New York, 1940. "A.P.I.-A.S.M.E. Code for the Design, Construction, Inspection, and Repair of Unfired Pressure Vessels for Petroleum Liquids and Gases," 3rd ed. A.P.I.-A.S.M.E., New York, 1938. "Rules for Construction of Unfired Pressure Vessels," Am. Soc. Mech. Eng., New York.

The application of high pressures to chemical processes has come to the fore as an important tool in the hands of the chemical engineer only since about 1913. The advantages of high pressures lie in the fact that many systems which undergo no change at atmospheric pressure will react to give commercially attractive yields if the pressure is increased to a few hundred atmospheres. Reactions that involve a diminution in volume are favored, reactions in the liquid phase may be speeded up as a result of higher temperatures possible at elevated pressures, and the size—and sometimes the weight—of equipment is materially reduced.

Since there is great experimental activity in this field, equipment of laboratory scale has been included in the following article in the belief that it will be helpful to those about to start an experimental study. Full-scale plants are relatively few, the equipment in them is specially designed, and information about them, except of the most general sort, is difficult to find in the literature. These conditions have imposed certain limitations upon the summary that follows.

Selection of Materials. With the exception of the long-established use of hydraulic presses for baling, drying, expressing oils, forging, forming, and molding there are few high-pressure processes that do not involve extremes of temperature, either very low as in the liquefaction of gases, or fairly high as in most catalytic processes. The synthesis of ammonia by the Claude process, for example, requires pressures of 800 to 1000 atm. and temperatures of 450° to 600°C., and organic catalytic processes proceed at rather lower temperatures, perhaps 250° to 450°C. Accordingly, the selection of materials for high-pressure work is further complicated by the need to provide for extremes of temperature as well as for special conditions such as oxidation, reduction, corrosion, the diffusion of gases—especially hydrogen—through metals, and the tendency of hydrogen to remove carbon from steel to a dangerous degree, even at moderate temperatures.

Steels are almost the only materials with sufficiently high tensile strength obtainable at sufficiently low cost to be used in quantity, though occasionally a single piece of equipment may be made of other materials such as nickel or nichrome when conditions of temperature or oxidation are severe, or nichrome or aluminum bronze if the material must be non-magnetic. Steel has the disadvantage, however, that its iron content is destructive to organic material at high temperatures, especially to compounds from which water can be split off and to gas mixtures containing carbon monoxide which may be converted to the inert and relatively useless methane. Moreover, carbon monoxide may weaken the walls by the formation of iron carbonyl, which in turn may poison the catalyst or redeposit iron in it, thus causing undesirable side reactions. These conditions require steel equipment to be lined with other metals not in themselves resistant to high pressures. Copper, aluminum, lead, silver, and gold are available for this purpose, and various other metals could undoubtedly be used if the necessity arose. It is safest not to rely on any increase in the strength of a pressure vessel that might result from the presence of a liner. In inserting a liner, care must be taken to keep the vessel dry, in order to prevent rupture due to expansion or vaporization of liquids in the space next to the wall upon subsequent heating. Imperfections in the bond between the lining and the walls necessitate great care in reducing the pressure in lined equipment to prevent gas pockets between the supporting wall and the lining from bursting inward.

In choosing the material for constructing a piece of equipment for high pressures, a list of the materials fulfilling the requirements of special conditions, including workability, should be drawn up. Then a choice from among these materials can be made upon the basis of cost and weight. As most high-pressure equipment is cylindrical, the conclusions drawn from a discussion of the wall thickness of tubing can also be applied to larger apparatus.

The methods ordinarily used for choosing a suitable ratio of the inside to the outside diameter are based on the assumption that a cylinder subjected to an internal pressure so high that it ceases to behave as an elastic body may be considered to have begun to fail. The assumption is not entirely valid, for, when the internal layers are stressed to the yield point, they flow and the stresses are redistributed in the outer layers without failure, other than a departure from elastic behavior. The inner layers are work-hardened by this process, and, after a suitable low-temperature heat treatment, the cylinder may actually be stronger than it was before. This is the process of autofrettage. The assumption is, however, very useful in practice. Five well-recognized methods of arriving at the pressure at which the stress at the inner surface of the cylinder reaches the yield point are available. For design one may use these same methods, using as a working or allowable stress the yield point divided by a suitable factor of safety, rarely less than two. If

S = allowable fiber stress, lb. per sq. in.
P = internal pressure, lb. per sq. in.
R = inner radius/outer radius

then, assuming a uniform stress distribution across the thickness of the cylinder and neglecting the end pull,

$$\frac{P}{S} = \frac{1 - R}{R} \qquad \text{(Thin cylinder)}$$

The maximum principal stress in a cylinder subjected to internal pressure is the tangential stress at the inner wall, which is related to the pressure by Lamé's formula,

$$\frac{P}{S} = \frac{1 - R^2}{1 + R^2} \qquad \text{(Maximum principal stress)}$$

The maximum shear stress is related to the pressure by

$$\frac{P}{S} = \frac{1 - R^2}{2} \qquad \text{(Maximum shear stress)}$$

If we take the coefficient of lateral contraction, or Poisson's ratio, as 0.25 for steel, then the maximum-strain theory gives

$$\frac{P}{S} = \frac{1 - R^2}{1.25 + 0.75R^2} \qquad \text{(Maximum strain)}$$

and the maximum-strain-energy theory gives the relation

$$\frac{P}{S} = \frac{2(1 - R^2)}{\sqrt{10 + 6R^4}} \qquad \text{(Maximum strain energy)}$$

These relations are plotted in Fig. 1.

The thin-cylinder formula is satisfactory for use when R is greater than 0.9. The conflict as to which of the remaining four relations gives the most reliable method of computing the pressure at which the cylinder ceases to behave elastically (and hence by using an appropriate safety factor, the safe pressure) is as yet unresolved. Newitt notes that the maximum-principal-stress theory describes the behavior of brittle materials such as cast iron and adduces data showing that mild steel follows the maximum-strain-energy relation, but that such high-tensile steels as chrome-nickel and chrome-nickel-molybdenum follow the maximum-shear-stress relation. Norris [Combined Stresses in Thick-Walled Cylinders, *Trans. Am. Soc. Mech. Eng., Applied Mechanics*, **51**–6, 61–65 (1929)] found that, for a "high quality gun steel" for which Poisson's ratio was 0.3, the maximum strain was the deciding factor.

At ordinary temperatures a balance must be struck between carbon steels, which with their lower permissible fiber stress require very heavy tubing, and alloy steels, which are far more expensive per pound but permit the use of much lighter walls and are more resistant to corrosion. Since they are less dependent upon their carbon content for their tensile strength, alloy steels are less affected by the removal of carbon by hydrogen at elevated temperatures. Other things being equal, the higher the pressure the greater the superiority of alloy steels.

Alloy steels of very high ultimate strength or limit of proportionality have usually gained these properties as the result of heat treatment in the course of which their ductility has been much reduced. This is shown by a decrease in the values of the elongation and reduction in area. Such steels must be used with caution because, unlike more ductile materials, they cannot yield and permanently relieve concentrated stresses. When subjected to pulsating stresses, they are especially likely to fail, breaking violently and scattering fragments. With all steels used in high-pressure work, but especially with these strong but brittle alloys, it is necessary to round off corners which might otherwise be the site of excessive stresses.

At higher temperatures reliance can no longer be placed exclusively upon simple stress calculations because of the importance of creep, *i.e.*, the tendency of the material to stretch continuously under long-continued stress. As more delicate instruments for measuring strain are developed, the stress which causes an observable continuous change in dimensions becomes lower and lower. Since it is thus improbable that at any given temperature there is a lower limiting creep stress, it is variously recommended that the maximum fiber stress be taken as that causing 10^{-8} strain (change in length per unit length) per hour, or as two-thirds that causing 10^{-7} strain per hour (10^{-8} strain per hour = 0.0088 per cent deformation per year). Data for the former are lacking, but there is a great body of data, including the latter, in

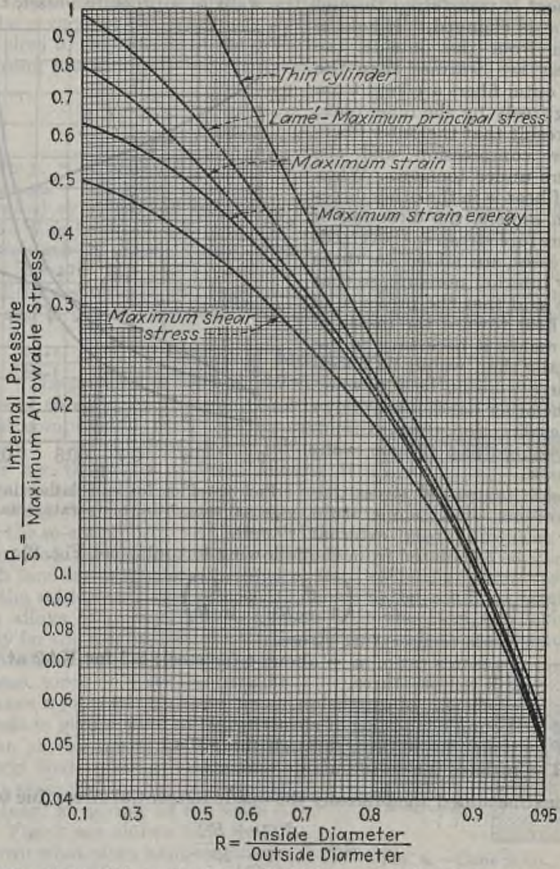


FIG. 1.—Comparison of formulas for wall thickness of cylindrical vessels, based on a value of 0.25 for Poisson's ratio.

"Compilation of Available High-temperature Creep Characteristics of Metals and Alloys," by the Creep Data Section of the Joint Research Committee on Effect of Temperature on the Properties of Metals, A.S.T.M.-A.S.M.E., Philadelphia and New York, 1938.

When creep data are not available, data on the tensile properties of the materials must be used, but with caution. If the material has been heat-treated, it should be used only at temperatures substantially below that at which it was treated, and even then the occurrence of slow changes may cause trouble. Tensile tests should be made on specimens thoroughly aged at the test temperature.

When heat is transferred through the walls of a pressure vessel, the stress distribution is changed. If heat is flowing in from the outside, the outer fibers are warmer than the inner; the outer fibers are kept from expanding freely by the inner ones, with the result that the outer fibers are under compression and the inner fibers under tension. When the flow of heat is in the other direction, as in coolers, the direction of the temperature stresses is reversed. These stresses are added to those caused by internal pressure, so that it may be said that the presence of a flow of heat toward the inside of the tube makes it necessary to use thicker walls than would be necessary if there were no heat flowing; and, on the other hand, if heat flows in the other direction, the maximum stress is reduced and thinner walls may be used.

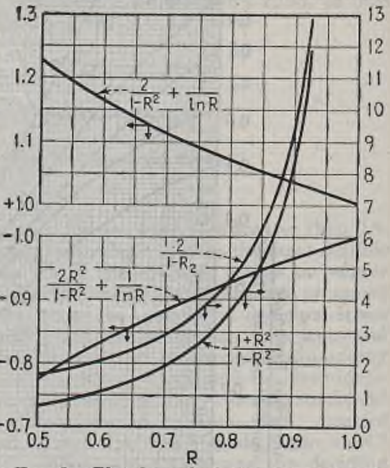


FIG. 2.—Plot for calculating internal and external tangential temperature and pressure stresses.

The inside tangential stress due to the presence of a temperature gradient resulting from external heating may be calculated from the following formula (see Fig. 2):

$$S_{Ti} = \frac{m}{2(m-1)} aE(\Delta T)\left(\frac{2}{1-R^2} + \frac{1}{\ln R}\right)$$

where S_{Ti} = inside temperature stress.

$1/m$ = Poisson's ratio, m = about 4 for steel: 3.3 for KA2 at cracking-still temperatures.

a = coefficient of expansion.

E = Young's modulus of elasticity.

ΔT = temperature drop through the walls.

R = ratio of inside to outside diameter.

To this must be added algebraically the inside tangential stress due to pressure

$$S_{Pi} = \frac{P(1+R^2)}{1-R^2}$$

At the outer surface the tangential temperature stress is given by

$$S_{To} = \frac{m}{2(m-1)} aE(\Delta t)\left(\frac{2R^2}{1-R^2} + \frac{1}{\ln R}\right)$$

and the tangential stress due to pressure is

$$S_{Po} = \frac{2PR^2}{1-R^2}$$

For a more complete discussion of temperature stresses see Luster [Design of Thick-walled Tubes Subjected to Pressure and Heat Input, *Trans. Am. Soc. Mech. Eng., Fuel Steam Power*, **53**-12, 161–172 (1931)].

Thermocouple wells or catalyst containers across which there is a large pressure drop resulting from high velocities or the presence of partial plugs present the reverse of the conditions described above, inasmuch as the pressure is applied to the outside of the cylinder. In long cylinders with $R < 0.9$ the collapsing pressure may be calculated from the following formula:

$$\frac{S_{\text{Y.P.}}}{P_{\text{collapse}}} = \frac{2}{1 - R^2}$$

in which $S_{\text{Y.P.}}$ = stress at the yield point.
P_{collapse} = collapsing pressure.

In thin-walled cylinders the collapsing pressure falls off very rapidly, especially when R is between 0.95 and 1.0. See Jasper and Sullivan [The Collapsing Strength of Steel Tubes, *Trans. Am. Soc. Mech. Eng., Applied Mechanics*, **53**–17*b*, 219–245 (1931)] for further details. Tubes subject to these conditions should be as nearly round as possible, for variations in wall thickness and in out-of-roundness both affect the collapsing pressure.

Design of Apparatus

Joints. The problem of joining two surfaces pressure-tight may be solved by the use of threads, by metallic bonds applied in the molten state, or by mechanical compression with line contact or with gaskets. Very satisfactory cold joints can be made by threading tubing, wetting it thoroughly with solder, and screwing it into a coupling which is already wet with solder, producing the so-called "tinned" joint. When higher temperatures are involved, however, the use of such familiar methods as brazing or hard soldering is restricted by the fact that the soldering metal, unless its choice is the result of careful tests, may form alloys of undesirable brittleness with the steel, producing weak joints. By far the most satisfactory joint of this type is the autogenous weld, made with either the electric arc or the oxyacetylene torch. When a capable welder makes a joint of this type, using welding rods to give a weld of the same composition as the metal he is joining, and properly heat-treating it afterward, there is no reason why the weld should not be at least as strong as the walls it joins. In Fig. 3 are shown four welds. At *a* is shown what often happens unless the faces to be joined are turned or filed to form a V-notch. Such a weld is weak because of lack of metal, lack of penetration, and the likelihood that cracks will be present, forming little regions of high stress intensity which are always potential sources of trouble. The joint at *b* is better, with much improved penetration, but in trying to avoid narrowing the channel as in *d* the welder has left a notch which may cause trouble from excessive stresses. The weld at *c* is free from these objections.

FIG. 3.—Types of weld.

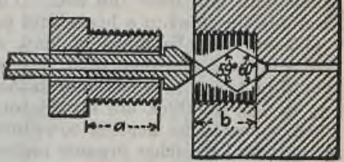


FIG. 4.—Cone joint.

A threaded joint made with one-half the taper of standard pipe threads is capable of giving good results when made by a skillful mechanic. Its use is for the most part limited to pressures of 2000 or 3000 lb. per sq. in.

Small pipe lines, perhaps up to $\frac{1}{2}$ in., can be joined very satisfactorily by means of the cone joint shown in Fig. 4. The angle of the cone is 58 or 59 deg. and of the seat 60 deg., thus providing for line contact as shown in Fig. 5. This joint should be made with a greater than b, and b sufficiently long to permit four or more threads to engage even when the seat is new. Such a joint is good for pressures up to at least 1500 atm. In contrast to this joint, which must be welded to tubing, is that in Fig. 6, in which the cone is turned on the end of the tubing. This has the disadvantage of somewhat weakening the tubing at the end, by decreasing the wall thickness. When the seats of these joints become scored with use, they may be refaced with a small hand reamer. Other workers obtain line contact by using an ovoid instead of a cone, though this shape is more difficult to machine and its superiority is doubtful.

FIG. 5.—Line contact of cone joint.

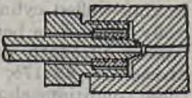


FIG. 6.—Cone turned on tube end.

Line contact between a conical and a spherical surface, the first faced on the end of the tubing, the second ground on a hardened "lens ring," makes the joint shown in Fig. 7 tight. Left-hand threads on the flanges prevent them from moving toward each other as the outer members are turned up. Similar in principle is the wave-ring joint in Fig. 8. This joint must be carefully machined with clearances so close that the parts must be pushed together, but it is self-sealing and with care can be assembled many times.

A wide range of sizes is covered by the joint shown in Fig. 9. To make it, the tubing is first threaded, then three hand reamers are used in succession to face the ends, cut the gasket groove, and cut back the threads at a so that the cupped gasket can slip over the end. This form of gasket makes it possible to assemble the joint in a horizontal position.

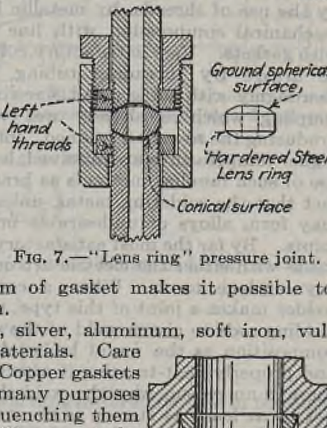


FIG. 7.—"Lens ring" pressure joint.

Gaskets may be made of lead, copper, silver, aluminum, soft iron, vulcanized fiber, or other relatively soft materials. Care should be taken to select smooth sheets. Copper gaskets $\frac{1}{32}$ to $\frac{1}{16}$ in. thick are satisfactory for many purposes when annealed by heating to redness. Quenching them in methanol or other organic materials will prevent the formation of oxides on the surface but is not usually necessary. The width of copper gaskets is regulated by the necessity of producing a pressure of 60,000 lb. per sq. in. in the gasket so that it will flow into the grooves or tool marks which make the joint tighter.

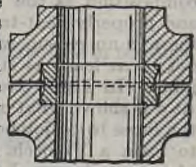


FIG. 8.—"Wave-ring" pressure joint.

The following method of calculating gasket widths for unconfined copper gaskets is due to Ernst and his coworkers. [Ernst, Reed, and Edwards, A Direct Synthetic Ammonia Plant, *Ind. Eng. Chem.* **17**, 775 (1925).]

$$\text{Width} = \left(\frac{2.5Pt}{2F} + \frac{PD}{4}\right) \cdot \frac{1}{60{,}000} \text{ in.}$$

where P = internal pressure, lb. per sq. in.
t = gasket thickness, in.
F = coefficient of friction for sliding, 0.15 to 0.20.
D = inside diameter of gasket, in.

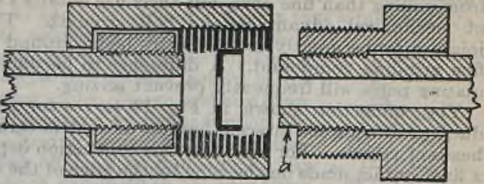


FIG. 9.—Pressure joint.

Let A = gasket area, sq. in. (*i.e.*, area of gasket annulus).
N = number of bolts.

Then 60,000 A/N is the load per bolt, which must not exceed a safe value for the strength of the bolt or for the intensity of shearing stress at the root of the bolt threads.

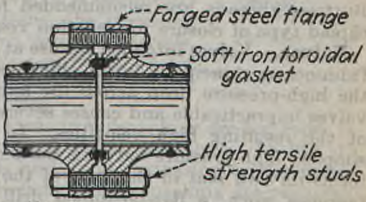


FIG. 10.—Ring joint for large tubes.

Figure 10 shows a flanged and ring-gasketed joint suitable for larger sizes of tubing, for which there is an A.P.I. standard. Other flanged joints are made in which the bearing surface is a gasket between the surfaces of the tubing, much as in Fig. 9, so that no welding is required.

Lines may be joined to apparatus in several ways, two of which are shown in Figs. 11 and 12. The upper end of the connection in Fig. 12 has the disadvantage of requiring space to provide for backing out a considerable distance; this is somewhat offset by the advantage that the gasket is subjected only to normal pressure, with no twisting to tear at the surface as in the lower gasket in Fig. 11. Both figures show how an ample cutback at the gasket seat allows for the free radial flow of the gasket under compression.

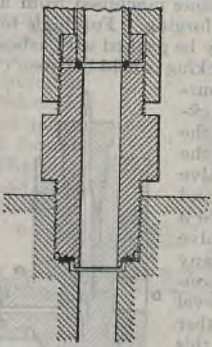


FIG. 11.—Method of attaching line to apparatus.

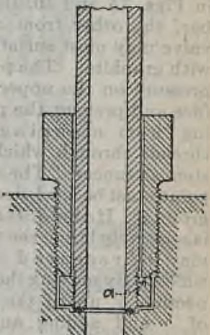


FIG. 12.—Method of attaching line to apparatus.

In making up the joint in Fig. 12, the surface *a* should be liberally lubricated with a paste of powdered graphite and Gredag having approximately the consistency of putty. This same mixture should be applied to all threads and bearing surfaces when tightening pressure vessels, especially those subjected to high temperatures, in order to minimize binding. Long exposure to high temperatures causes the graphite to burn away. To minimize seizing and galling in threads, it is necessary to avoid extremely tight fits and to have the pitch of the male and female threads the same within very close limits. Coarse threads may more easily be made free from galling than fine ones, but their use entails a sacrifice of mechanical advantage and bolt strength. Threaded joints may be greatly improved by being ground in with fine grinding compound. A difference in hardness in the mating parts will frequently prevent seizing.

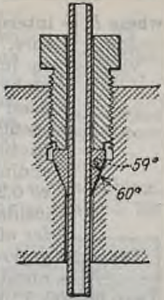
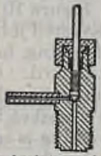


FIG. 13.—Pressure connection for introducing coils.

The connection shown in Fig. 13 serves for the introduction of coils for internal heaters or coolers through the heads of vessels. For tightness this connection depends on a line contact made on the cone by the edge of the seat and on a squeezing of the thin edge of the cone against the tube.

For most purposes it will be found that the U. S. fine standard screw-thread series will give satisfactory results. Buttress threads are recommended for the interrupted-thread type of closure for reaction vessels.

Valves. Valves intended for use at high pressures differ from others principally in the thickness of the walls and in the high-pressure drop across the seat which makes gate valves impracticable and causes severe erosion in regulating valves by reason of the resulting high velocities. In Fig. 14 is shown the simplest high-pressure valve: a hardened conical point which forms its own seat in the steel of the body. When the seat becomes worn, a flat-bottomed drill is used to face it off, allowing the point to reseat itself. Such a valve is useful in the laboratory and for taking small samples or determining liquid levels in the plant.



American Instrument Co.

FIG. 14.—Simple high-pressure valve.

A very useful small valve for plant and laboratory is shown in Figs. 15 and 16, the one machined from a section of steel bar, the other from a forging. For high temperatures this valve may most suitably be packed with asbestos impregnated with graphite. The packing is held between two spacers which ensure normal pressure on the upper surface and prevent the packing from clogging the threads through which the stem advances. The valve point must be hardened and ground. If the distance *a* is relatively large, the valve may be reseated many times. By moving the connection *b′* up to the level of *b* and adding another opening at the bottom, this valve may be changed into a T-valve, useful in constructing manifolds.

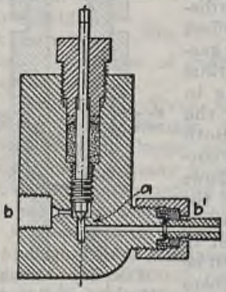


FIG. 15. FIG. 16.

FIG. 15.—Machined valve. FIG. 16.—Valve from forging.

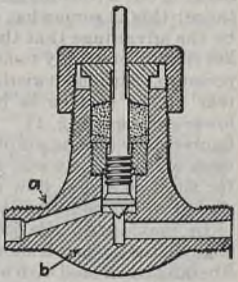


FIG. 17.—Forged valve for repacking under pressure.

The double-cone point on the stem, shown in the forged valve illustrated in Fig. 17, permits it to be repacked under pressure. The upper seat is screwed in on a gasket, using a tool that fits into recesses in the upper surface. By making the angle of the point smaller this may be made into a regulating valve, but, if this is done, the body should be somewhat longer and the seat reamed out to give a larger surface of contact.

The two-piece stem and replaceable seat of Fig. 18 have advantages over the arrangement in Fig. 17 and may be used in a body that is only slightly larger than that of Fig. 17 to accommodate the seat. Both the point *a* and the seat *b* should be made of material of great toughness, to resist the very severe erosive action of the material flowing across the seat under a large pressure drop. A gasket at *c* makes it unnecessary to depend upon the threads for tightness. By allowing proper fillets, the tendency of the button *d* to break off may be diminished. As the valve point moves only back and forth in the seat there is no tendency for it to score, but this same movement makes it impossible to close the valve when there is scale or catalyst dust to lodge in the seat. Though no valve is completely satisfactory under such adverse conditions, the operation may be improved by the use of very hard, strong materials, such as nitralloy or Stellite, for the seating surfaces. A bit of scale that lodges in the valve may then be crushed and blown through, permitting the valve to close.

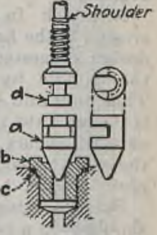


FIG. 18.—Two-piece valve stem and removable seat.

In all the valves so far considered the threads and packing are too close to the fluid stream to be satisfactory at really high temperatures. In Fig. 19 is shown a valve with a three-piece stem and a long extension to the upper part of the valve body, by which the threads and the packing are removed from the region of high temperatures. If required, the extension may be water cooled or provided with fins for air cooling. The principal difficulty is differential expansion between the stem and the bonnet. The insertion of a gasket at *a* permits the valve to be repacked in the open position. The rounded end of the stem gives opportunity for the slight play required by the point in finding the seat. With slight changes in the design of seat and point, this valve may be transformed from a globe into a regulating valve. The complete removability of the seat and stem permits the valve body to be welded into place without much danger of trouble from warping.

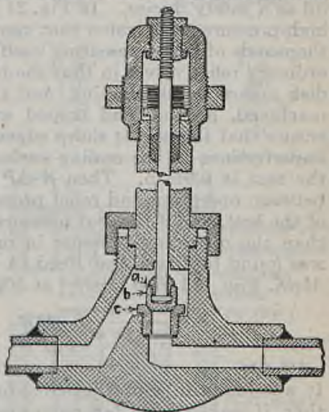


FIG. 19.—Valve for high temperatures.

One of the principal sources of trouble with high-pressure valves is the tearing away of packing by slightly rusted stems—a trouble readily avoided by the use of stainless steel. When circumstances require, the stems, seats, and point may be made of monel metal or other special alloys. Many workers provide a vent for gas leaking past the upper seat of repackable valves, thus preventing a possible accident when the bonnet is removed and the packing is no longer held in place.

By a slight sacrifice of symmetry the weight of many high-pressure valves may be materially reduced, the fabrication made somewhat easier, and some confusion avoided. Looking at Fig. 17 it is at once apparent that if the left-hand opening were on a level with the seat, useless metal at *b* would be eliminated, the wall at *a* could readily be made thicker, the slanting inlet would be discarded, and no one could be in doubt as to which is the inlet side of the valve. The same is true of Fig. 19.

Although in these valves the principal weapon against erosion is the hardness and toughness of the steel used, a recent design suggested by the Mathieson Alkali Works, Inc., avoids the trouble by using fresh surfaces as the wear progresses. This stem and seat are shown in section in Fig. 20. The stem is lapped into the seat with a clearance of but 0.0001 in., and as wear occurs the stem is advanced farther and farther into the seat. A conical portion at the upper end provides for complete closure.


Mathieson Alkali Works

FIG. 20.—Mathieson Alkali pressure valve.

Safety requires that the valve stem be provided with a shoulder in a position such that if the threads by which it is advanced should break, the stem would not be ejected with great force from the valve. Valves should be anchored securely in position so that in opening or closing them the leads will not be subjected to unusual stresses.

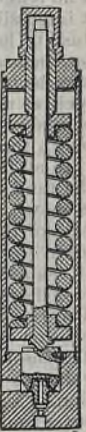

Ernst, Reed

FIG. 21.—High-pressure relief valve.

There is some difficulty in making high-pressure relief valves reseat themselves after relief. Such valves are a source of danger, for the tendency is to screw down on the spring until the valve does reseat, whereupon the valve ceases to be useful as a safety device. In Fig. 21 is shown a cross section of a high-pressure relief valve that can be made to relieve pressure thousands of times, reseating itself each time. It differs from ordinary relief valves in that the ball contact at the top of the disk makes it self-aligning, and that the edge of the seat is machined, ground, and lapped with the greatest of care to ensure that the seat is sharp edged and free from the smallest imperfections in the sealing surface. A convenient width of the seat is 0.01 in. Then if ΔP is the allowable difference between operating and relief pressures, and D is the diameter of the seat, and if the seat pressure must be 20 per cent higher than the operating pressure in order to prevent leakage, as was found by Ernst and Reed [A High-pressure Relief Valve, *Mech. Eng.*, **48**, 595 (1926)] at 5000 lb. per sq. in.,

$$D = \frac{0.048P}{(\Delta P)}$$

It was found in actual practice by Ernst and Reed that, if leakage of a few cubic feet per day can be tolerated, a valve with a disk calculated in this way can be set to relieve with only one-half as great a pressure difference. The seat is made in a gasket of copper or other soft material.

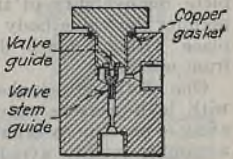


FIG. 22.—High-pressure check valve. (*Courtesy of American Instrument Co.*)

Figure 22 shows a check valve for use at high pressures.

In Fig. 23 is shown a cross section through a magnetic control valve by means of which small streams of material may be controlled. The operation of the valve is controlled by means of a relay actuated by a contact on a Bourdon gage. The force pushing up against the needle valve is approximately balanced by the valve spring, so that a relatively slight pull on the armature serves to open the valve. Suitable modifications of the ordinary motor-operated valve will give excellent service in high-pressure lines requiring automatic or distant control of flow.

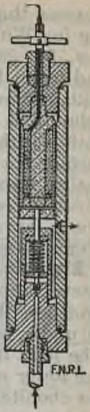


Fig. 23.—Magnetic control valve.

Sight Glasses and Windows. Cylindrical sight glasses of the usual sort, though readily constructed and capable of being used at high pressures, are actually little used because it is difficult to secure perfect alignment and because unequal thermal expansion causes the glass to break. Where only an indication of level is desired, other methods are available, some of which are discussed below. Where circumstances require the use of cylindrical glasses the table below, from Onnes and Braak [quoted by Bone, (Bone, Newitt, and Townend, "Gaseous Combustion at High Pressures," Longmans, London, 1929)], may be used as a guide.

In Fig. 24 is shown a transverse section through a sight glass manufactured by the Diamond Power Specialty Company for high-pressure work. The really high stress intensities which make the apparatus tight are not transmitted to the glass at all but are found in the edges of the three sheets between the glass and the body. Being subject to the considerably lower pressures of the asbestos gasket on one side and the gasket plus the operating pressure on the other, the glass is sufficiently free to move so that it adjusts itself to changes in pressure and temperature. The manufacturers state that it can be used at as high as 2200 lb. per sq. in.

Bursting Tests on Glass Tubes

Outside diameter, mm.	Inside diameter, mm.	Ratio	Bursting pressure, atm.
5.8	0.46	12.60	1200
6.7	0.24	27.90	1100
9.2	3.00	3.06	380
10.4	4.00	2.60	240
3.8	2.42	1.57	283
6.4	4.78	1.34	221

Bone (*loc. cit.*) describes a quartz window which is illustrated in Fig. 25. The truncated quartz or glass cone is ground into the steel shell and held in place with an asbestos-graphite washer and a steel collar. The window is said to be tight against explosions producing pressures of 400 atm.

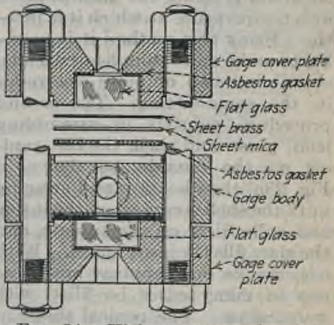


Fig. 24.—High-pressure sight glass. (*Courtesy of Diamond Power Specialty Co.*)

Conical windows, however, are gradually forced so far into the steel shell that when the pressure is reduced the elastic recovery of the shell subjects them to such high radial compressive stresses that they break. Poulter [A Glass Window Mounting for Withstanding Pressures of 30,000 Atmospheres, *Phys. Rev.*, **35**, 297 (1930)] has described a window which, in modified form, Bridgman ["The Physics of High Pressure," Macmillan, New York] has illustrated as shown in Fig. 26. The cylindrical window is merely fastened over the central hole with a little adhesive to keep it in place. The steel-to-glass surface is carefully finished, but it is not necessary to ensure optical flatness of either the glass cylinder or the steel plate on which it rests. Windows of this type, with various modifications, have been used at pressures of several thousand atmospheres.

FIG. 25.—High-pressure quartz window. (*By permission of Longmans, Green & Co.*)

Electric Leads. It is sometimes necessary to introduce electric apparatus into pressure vessels. In Fig. 27 is shown a lead described by Bone (*loc. cit.*) for use at low temperatures, or under conditions where heat lasts for very short periods. The joint is made tight by warming the lower end of the lead to soften the ebonite, then tightening the nut at the top, repeating the process two or three times if necessary, and finishing with the application of zinc oxychloride cement.

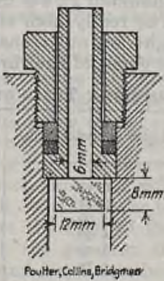


FIG. 26.—High-pressure cylindrical window.

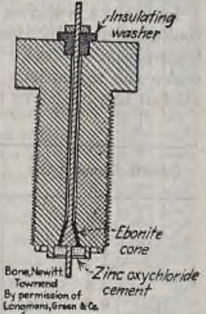


FIG. 27.—Electric lead for high pressure.

In Fig. 28*b* is shown a simple but very effective lead of a type manufactured by the General Electric Company and capable of being used at 450°C. and 300 atm. (Mackay, "Seal for Electric Devices" U. S. Patent 1,456,110). Two methods of manufacture are possible: one consists in reaming out the steel shell and pressing the glass in place at a temperature at which it is plastic. Using this method it is possible to make more complicated leads, containing two or more electrodes in the same fitting. The second procedure consists in assembling lead, glass, and shell, the last conical on the outside, as shown in Fig. 28*a*, then heating and swaging until the shell is conical on the inside and cylindrical on the outside, and the glass fills the inner space. With use, cracks may appear, but these can in many cases be filled with water glass. The conical shape of the glass core effectively prevents it from being blown out.

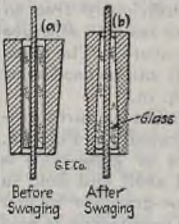


FIG. 28.—Electric lead for high pressure.

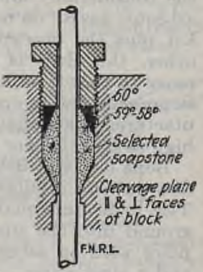


FIG. 29.—Electric lead for high pressure.

Figure 29 shows still another type of lead which is suitable for both high pressures and high temperatures and has been found very satisfactory by the Fixed Nitrogen Research Laboratory.

It is also worthy of mention that some workers have found automobile spark plugs of value as high-pressure leads.

Reaction Vessels. Reaction vessels are usually forged or welded cylinders, of which many are covered by the A.S.M.E. Code for Unfired Pressure Vessels and the A.P.I.-A.S.M.E. Code for Unfired Pressure Vessels for Petroleum Liquids and Gases (see list of references, p. 2168). Even where these codes do not apply, or apply only through paragraphs calling for experimental proof of the safety of the design, they are very useful guides. The sections of the A.P.I.-A.S.M.E. Code dealing with tests and inspection will be found especially helpful. After fabrication, pressure vessels should be tested under hydraulic pressures 50 per cent higher than the operating pressure for a sufficiently long time to permit the detection of leaks. On welded vessels the pressure should then be reduced to 25 per cent more than the working pressure and the joints hammered with hammers whose heads have weights in pounds equal to ten times the thickness of the metal in inches, but no more than 10 lb. The pressure should then be raised to 50 per cent more than the working pressure, and leaks again should be sought for. If the equipment is to be used at an elevated temperature as well as at high pressures and cannot be tested under operating conditions, it should be tested at a correspondingly higher pressure at ordinary temperatures. A word of caution is necessary here: there is a general belief that when a vessel is tested under hydraulic pressure and fails, the metal splits and, as a result of the low compressibility of water, a small amount of water oozes out quietly. Though this is true in general, yet at high pressures the potential energy stored up as elastic deformation of the walls of the vessel and compression of the liquid is by no means small, and pieces weighing several hundred pounds have been hurled 10 to 50 ft. with great violence when a vessel was tested to destruction.

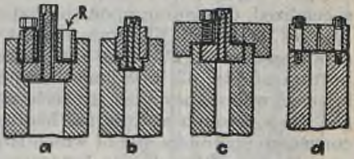


FIG. 30.—Methods of closing cylinders.

In Fig. 30 are shown several methods of closing cylinders. The ring *R* of type *a* is provided with interrupted buttress threads which permit it to be dropped in place and rotated a fraction of a turn until the threads engage. The closing head is then tightened down on the gasket by turning the bolts in the ring. Smaller cylinders may be closed by the method shown at *b*. At *c* is shown a method similar to *a* except that the threads are outside the cylinder. Though it excels in simplicity and in the ease with which it is machined, the closure shown at *d* has the disadvantage of somewhat weakening the cylinder walls. However, the method of manufacture often permits the head of a pressure vessel to be enlarged to provide space for a bolt circle, as in Fig. 31, so that weakness of the walls need not follow the use of method *d*. In Fig. 31 a type of closure is shown in which the threads that hold the head in place do not also take up the stresses required to compress the gasket.

Ammonia Casale S.A.
Brit. Pat. 294,756

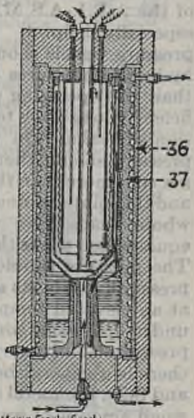
FIG. 31.—Closure.

It is worth mentioning that enlargement of the head of a vessel permits an increase in the space for inlet and outlet tubes, thermocouple wells, and so on, that is of great value in making the vessel easy to use. The top of the closing head should project above the bolt heads as shown at Fig. 30*a*, whenever the setting permits the bolts to be tightened with such an arrangement, as tightening the connections is thereby made much easier. A comparison of Fig. 30*a* with Fig. 33 will illustrate this point.

Usually only one end of a reaction vessel is closed with a plug, the other end being integral with the walls. If the integral end is a hemisphere concave outward, its thickness may be that of the walls, but if it is flat, the thickness must be materially increased to obtain the stiffness required to resist buckling (*cf.* Fig. 33). For the same reason plugs or combinations of plug and nut as in Fig. 30*a*, *b*, and *c* are thicker than the vessel walls. Sharp corners should be avoided so as to prevent stress concentration.

The design of the inside of reaction vessels is highly specialized, depending upon the nature of the reaction. Figures 32 and 33 illustrate two of the many possible arrangements. Following the gases entering at the bottom of Fig. 32, one finds them passing through an injector where they pick up recirculating unconverted gas, then through an internal heat exchanger made of concentric cylinders one of which is provided with corrugations or other means for increasing the effectiveness of heat transfer, next up around an internal heater, down through the catalyst, through the heat exchanger, and then along a cooler which condenses the product. The residual gases pass through ports into the injector, and the liquid product passes out at the bottom. If any inert gases are present, some means must of course be provided for removing them continuously.

The cooling system shown in Fig. 32 is made up of two shells. Shell 37 is provided with a helical cut turned in a lathe. The surfaces between shells 36 and 37 are carefully finished, but the inside of the pressure-resisting vessel need not be so carefully prepared. Now when the vessel is made up and pressure is applied, the pressure on the inside of shell 37 expands it and shell 36 against the walls of the main vessel, making a tight joint between shell 36 and shell 37, and thus providing a channel for the cooling medium.



Maria Casale-Sacchi
Brit. Pat. 328,139

FIG. 32.—Section through reaction vessel.

The reaction vessel shown in Fig. 33 receives the material at the bottom, whence it passes up between the outside walls and the catalyst chamber, thus helping in maintaining the walls of the vessel at a low temperature. At the top of the chamber, the gas passes down around the outside of three sets of heat-exchanger coils in parallel, then around the internal heater. At the bottom of the heater it enters the first catalyst zone, where a high linear velocity somewhat distributes the heat output due to the reaction. The second zone, of larger area, receives the partially reacted gas, giving it more time to approach equilibrium; this process is carried still further in the still larger third zone. Such an arrangement serves to level out the temperature gradients in the reaction vessel. In vessels of simpler design the same end is attained by diluting the catalyst that meets the incoming gas. From the catalyst bed of Fig. 33 the gases pass through the coils of the three heat exchangers out to a condenser. Electric leads for the heater are provided both inside and outside the vessel. Thermocouple wells are provided in the first and third catalyst zones.

Though an injector is shown in Fig. 32 as a means of recirculating the gases, it should be noted that the use of a booster or recirculating pump will usually give flexibility and satisfactory service. The manufacturers of high-pressure compressors can also provide pumps of this sort.

The decision whether to use a short vessel of large diameter or a long, narrow one must rest on a careful consideration of a number of conflicting factors. If a long, narrow vessel is used, the actual thickness of the walls will be less and the area of the walls greater than in a short, wide vessel. These conditions favor the transfer of heat, but the greater area of the walls may increase the yield of undesirable by-products which are the result of the catalytic action of the walls. The high linear velocity in a long, narrow vessel favors heat transfer to a still greater extent than the thinner walls, but it has the disadvantage of causing a large pressure drop. Not only does such a pressure drop represent a loss of power, but if the direction of flow is downward through the catalyst bed, the pressure drop added to the force of gravity may crush the catalyst. If the flow is upward through the catalyst, on the other hand, the catalyst may be picked up bodily and churned to dust.

Pumps and Compressors. Little has been published about the highly specialized art of designing compressors for very high pressures. Standard gas compressors for 4000 to 6000 lb. per sq. in. are readily available and are very satisfactory, but compressors for 6000 to 60,000 lb. per sq. in. are still to be considered special equipment.

Certain generalizations may be made, however. For pressures of 1000 to 1200 lb. per sq. in., three stages are usual; four stages may be used up to 5000 lb. per sq. in.; five to seven stages, up to 15,000 lb. per sq. in. It should be remembered that the power required for compression rises only logarithmically with the pressure (*cf.* Fig. 34), so that usually this is a relatively small part of the cost of an increase in pressure. Though many stages make the compression more nearly isothermal and hence reduce the power required, the advantages of increasing the number of stages as shown in Fig. 34 are offset by the increasing complexity of the apparatus and the rising cost as the separate cooling systems between the stages are added.

Experience shows that the parts of gas compressors that give most trouble are valves, piston packings, and stuffing boxes. Accordingly, good design requires that these parts should be as accessible as possible. It is advantageous to have valves made of stainless steel to resist corrosion, and to make them as large in area as construction permits—even putting two or more in parallel if necessary, as the high gas densities make erosion important. Figure 35 shows a section through a plate valve that is very light and can be used in a

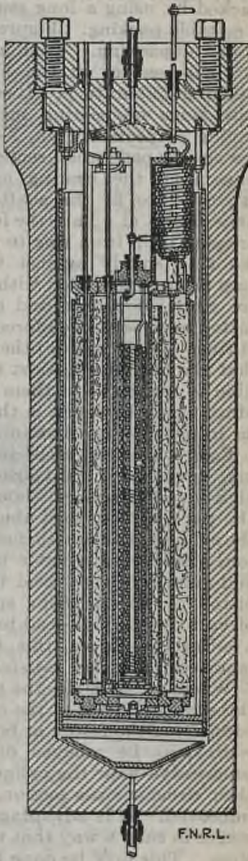
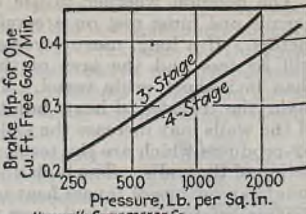


FIG. 33.—Section through reaction vessel.

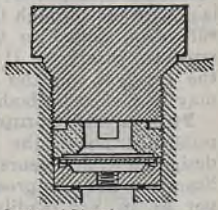
recirculating pump or in the lower stages of a compressor, where the pressure drop across the valve is not too high for so thin a disk to withstand. Higher pressure drops in the later stages require more sturdy construction.



Norwalk Compressor Co.

FIG. 34.—Power requirements of three- and four-stage compressors.

Single-acting pistons are fairly readily packed by using a long stuffing box with a suitable packing. Figure 36 shows two kinds of packing. That at *a* has given excellent service at the Fixed Nitrogen Research Laboratory. It consists of alternate rings of bearing metal and steel, having respectively an X-shaped and a lozenge-shaped cross section. The steel rings keep the bearing metal slightly expanded to fit the shaft. The S.E.A. rings shown at *b* have the feature that the pressure drop from ring to ring keeps the thin edge of the ring snugly against the piston. The space *c* may be partially filled with a lubricant. These rings are made of rubber and graphite composition with cotton or asbestos reinforcement. By using two sets of rings opposing each other, which are so constructed that the flap rubs against the cylinder wall, it is possible to make use of them for double-acting pistons. More usual, however, is the use of piston rings similar in principle to those on automobile pistons.



Ernst, Reed, Edwards

FIG. 35.—Plate valve.

Figure 37 shows a special piston-ring type of packing, devised by the Garlock Packing Company for high-pressure service, though these rings, like the packing described above, fit in a recess in the cylinder wall and do not travel with the piston. Each ring is made up of three tongued-and-grooved pieces, held together against the piston by an encircling spring. When made of cast iron or high-lead bronze and used with a nitralloy piston, this type of packing is reported to be very satisfactory.

(*a*) *Fixed Nitrogen Research Laboratory*
(**b**) *S.E.A. rings* Ronald Trist & Co. Ltd.

FIG. 36.—High-pressure packing.

Whenever possible, the piston should extend through both ends of the cylinder, for then by supporting the piston at both ends the cylinder walls may be relieved of its weight. With progressive wear the cylinders will then retain their circular cross section, and their life and tightness against leaks will be improved. It is advantageous to arrange the cylinders of the high-pressure stages in such a way that they may be replaced by spare cylinders in a short time. This may be done by placing the last two stages on either side of the wrist pin, or on either side of a reciprocating traverse which is supported by a shoe and connected around one of the cylinders to the wrist pin by means of two long heavy bars.

Compressors are lubricated by means of small hydraulic pumps which feed oil into the cylinders and stuffing boxes. The use of too much oil is to be avoided, since it causes erosion of the pistons and packing. In compressing oxygen, oil cannot be used because of the excessive danger of explosions; water is the usual lubricant for this purpose. In services in which liquids may condense in the compressor, the condensate itself is sometimes used as the lubricant.

With a feed of gas at 4500 lb. per sq. in. or higher it is possible to perform the final compression to 15,000 lb. per sq. in. hydraulically, using either oil or water. This method is of great usefulness in small-scale testing, but it is by no means confined to such use, as compressors with fifth- and sixth-stage pistons operated by hydraulic intensifiers have been built to handle as much as 160,000 cu. ft. of free gas per hour at 1000 atm. In hydraulic work, the compressibility of the fluid cannot be neglected at high pressures. At 1000 atm., the volume of water is reduced approximately 3 per cent, so that to obtain the greatest delivery from a high-pressure pump it is necessary to keep the clearances as small as possible.

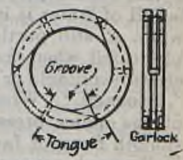


FIG. 37.—High-pressure piston-ring packing.

For the injection of large amounts of liquids, hydraulic pumps of proved design have long been available. The small pumps used as oilers on compressors will be found very useful for injecting small amounts of material. A frequent arrangement of valves and piston in hydraulic pumps of all sizes is shown in Fig. 38.

A type of packing for pistons that do not move too rapidly was developed by Bridgman for use at pressures far higher than those met in present-day commercial work. This packing (Fig. 39) may be used at lower pressures, not only for pistons, but, with slight modification, for joints in large apparatus. It consists of a piston a with a separate end b. In an annular space between the two is a much softer packing c, which may conveniently be made of rubber provided the temperature and stress permit. For very high pressures and very slight motion, copper, aluminum, or even soft iron may be used in place of rubber. The upward force on the end of b is balanced by an equal downward force on the upper face. As this force, however, is exerted only over the ring-shaped lower surface of the packing, the force per unit area or the pressure on the upper face of b must be greater than that on the lower face. Now the material of c is chosen so that it is plastic enough to transmit this pressure hydraulically to the sides of the cylinder at d. The hydraulic pressure at d is therefore always greater than that in the body of the cylinder, which accordingly cannot leak. At the same time, the clearances between the cylinder and the parts a and b are so small that the packing c cannot actually flow out of position. Figure 40 shows this principle applied to a joint. The small shoulder at e permits the joint to be made tight with no pressure in the cylinder; but when pressure is applied, reliance is placed only on the higher pressure in c.

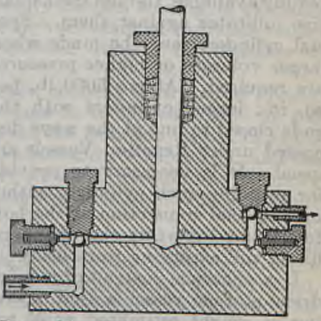


FIG. 38.—Valves and piston of hydraulic pump.

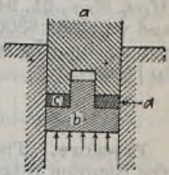


FIG. 39.—Bridgman type of packing.

In Fig. 41 is shown an arrangement of hydraulic pump, valves, and storage cylinders that has been found especially useful for testing purposes. One of the two cylinders, *A*, is filled with water which is then pushed out by gas at 300 atm. When the cylinder is full of gas, the waste valve is closed and water is pumped in until the pressure reaches 1000 atm. During this time, *B* has been supplying the needs of the system. When *B* is full of water it is cut off and refilled with 300 atm. gas, while *A*, still partly full, maintains the flow. It is possible to tell when the cylinder is full of water by the use of electric contacts or level indicators (see below).

FIG. 40.—Pressure joint.

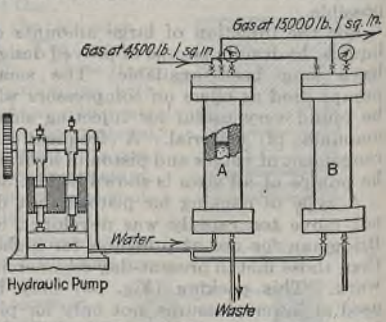


FIG. 41.—Arrangement of hydraulic pump, valves, and storage cylinders.

Storage.* When it is necessary to provide for storage of gas under pressure or to provide a surge chamber to smooth flows or facilitate control, steel cylinders are generally used, connected in banks to a manifold. For pressures up to 2000 lb. per sq. in. the familiar cylinders used for shipping compressed gases are readily available, though their small size militates against them. Special cylinders must be made when larger volumes or higher pressures are required. Above 5000 lb. per sq. in., forged cylinders with the ends closed in one of the ways discussed under Reaction Vessels are usual. It is necessary to provide for draining such cylinders, either by a special connection at the bottom or by inverting the cylinders, filling and emptying from below.

If a spherical instead of a cylindrical storage vessel is used, a saving in weight estimated at 25 per cent results [Furrer, Jasper, and Needham, "Special Containers for Transporting Gas under High Pressure," *Mech. Eng.*, **51**, 941 (1929)]. A single sphere, roughly 72 in. inside diameter and weighing 8300 lb. contains 15,400 cu. ft. free gas at a pressure of 2000 lb. per sq. in. To equal this, 77 ordinary 200 cu. ft. gas cylinders would have to be used.

Measurement and Control†

Pressure. The most usual method of measuring pressure is by means of the Bourdon tube pressure gage, which consists of a flattened bronze or steel tube bent into an arc. As pressure is applied the tube tends to straighten, and this movement is transmitted to a dial through a suitable magnification train. Bourdon tubes for high pressures are made of steel. Since so much depends upon them, only tubes made according to the most exacting standards and carefully aged by the manufacturer should be used. It is customary to use the gages at one-half the maximum pressure provided on the scale on fluctuating pressures and two-thirds of that pressure when the pressure is steady. If a Bourdon tube is overranged and subjected to a pressure higher

---

* See Sec. 20, p. 2276.

† Revised with the aid of E. D. Haigler.

than that at which it was stressed in the aging process, permanent distortion may occur, causing recalibration to become necessary.

Gages that are in continuous use, and especially those subjected to constant rapid fluctuations of pressure, should be checked at frequent intervals. A convenient way of doing this is to have an accurate master gage that can be attached for comparison somewhere in the line to the regular gage. At regular intervals the master gage should be checked against a dead-weight gage.

The dead-weight gage consists essentially of a precisely fitted piston and cylinder. Pressure is developed by direct (or indirect) dead-weight loading of the piston while it is being rotated or oscillated to avoid static friction. The oil or other viscous liquid with which the cylinder is filled serves both to transmit the pressure to the instrument under test and to lubricate the piston. With the effective piston area and the piston loading both precisely known, great accuracy is possible.

A pressure-measuring element operating on the dead-weight principle, using a combination of dead-weight and mercury-column loading, is available from the Bailey Meter Company. In it the high accuracy of the dead-weight principle is combined with automatic operation, remote transmission of reading, and extreme suppression of range.

Also of interest is the piston-type pressure element with helical spring, made by Esterline-Angus. This element, made in ranges to 20,000 lb. per sq. in., may be considered intermediate between the Bourdon spring, in which actuating area and resisting load or spring are one and the same unit, and the dead-weight loaded piston. It is particularly suited for rapidly and violently fluctuating high pressures such as are commonly encountered in hydraulic testing of pipe.

Pressure control may be effected pneumatically, electrically, or hydraulically. Most commonly used are the highly developed flexible pneumatic systems which lend themselves equally well to open-and-shut, proportional, or proportional-reset control action. Now less commonly used, except for special applications, are the electric systems. While types are now available which avoid compromise of pen or pointer accuracy to secure adequate contact actuation, the extra cost imposed by explosion-proof construction, when required, and the very considerable limitation of control accuracy or rangeability and response imposed by floating-type, motor-operated valves, or by open-and-shut action valves in a by-pass, restrict the use of electric control. Similar control limitations as well as relatively high cost have restricted the application of hydraulic controls.

For opening and closing relatively small streams a solenoid-operated valve may be used (see Fig. 23).

Temperature. The temperature in high-pressure apparatus is usually measured by means of thermocouples in wells which are let in either through the heads of the larger vessels or through special fittings in the connecting lines. Since cylinders are not stable to external pressure, it often happens that thermocouple wells collapse under conditions which do no damage to other parts of the apparatus. Unless, as sometimes happens, the tubing breaks, the collapse in itself does no harm. It may of course be prevented by having the thermocouple fill the well clear to the bottom, or, when this is impossible, by filling the empty portion with a rod which may be attached to the end of the couple. The heat capacity and condition of heavy walls required to resist the higher pressures make it impossible to get the exact temperature at any one moment or at any one spot in the surroundings of the well. Where such exact measurement is required, the thermocouple or resistance thermometer leads may be brought in through connections of the sort described in the paragraphs on Electric Leads, and the bare junction or coil placed directly in the catalyst bed. Errors are introduced by this method as a result of the effect of pressure on resistance and thermoelectric force, but these errors are relatively small. A very practical method involves the use of a thermocouple consisting of an insulated constantan wire running through a copper or iron tube to the closed end of which it is fastened in such a way as to form a proper thermoelectric junction. Several such commercial thermocouples or several dozen special ones can be welded into a fitting similar to that in Fig. 11 and the whole screwed into the head of the reaction vessel, taking no more space than a single well. Wires from the constantan and the tube lead to the pyrometer switch or pyrometer.

Thermocouples in wells are often satisfactory guides, because in practice it is usual not to decide in advance upon the temperature at which a certain point shall be kept, but to find by trial a temperature that gives satisfactory results. Moreover, this fact makes it possible, when reaction vessels are externally heated, to control the reaction temperature not directly by a thermocouple well in the reaction zone but indirectly by a couple imbedded in the wall of the vessel, thus giving an assurance that the wall at no time exceeds a safe temperature. A thermocouple in the reaction zone then merely serves as a guide to the necessary wall temperature.

Temperature control itself presents no features different from that at low pressures, except that the huge forces unleashed when a failure occurs in a fired vessel or in a vessel in a confined space may upon occasion make it advisable to have dual control, so that if one instrument fails another is at hand to take up the duty automatically. The same considerations make it advisable to check thermocouples frequently and to make sure that they are not subjected to temperatures and chemically reactive atmospheres for which they are not suited. Just as in the case of pressure control, temperature control installations in high-pressure synthesis plants are now largely pneumatic. Especially rapid has been the extension of pneumatic control to potentiometer-pyrometer measurements.

Flow. Two types of flow meters are useful at high pressures: one the familiar orifice meter, the other the Thomas electric meter. The latter depends upon measuring either the temperature rise from a fixed heat input or the energy input to maintain a fixed temperature rise. In the laboratory this may conveniently be accomplished by winding two equal resistances on the tubing on either side of the heating element, making these resistances the arms of a Wheatstone bridge and measuring only the difference between them. This method has the advantages of inherent compensation for temperature and pressure changes in gaseous mediums but the disadvantages of high cost and complication and extremely sluggish response in plant-scale, high-pressure equipment.

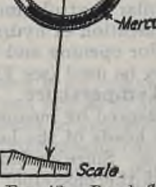


FIG. 42.—Bosch flow meter.

In an orifice meter the problem is that of measuring a small pressure drop at a high absolute pressure. In the Bosch flow meter (Fig. 42) the two ends of a semicircular mercury-in-steel pivoted manometer are connected to the orifice in the main line of flow, using very fine tubing wound into coils which interfere as little as possible with the free motion of the tube. As the flow increases, the mercury rises on the low-pressure side of the semicircle, which swings about the fulcrum to keep the center of gravity directly under the point of support. The motion of the semicircle is measured by a pointer and a scale.

Domestic industrial flow meters suitable for the high static and differential pressures usually encountered are generally of the stationary U-tube and float type. In the Foxboro meter, made in ratings of 5000 lb. per sq. in., the motion of the float is carried directly to the pointer through a precision seal bearing, which is lapped to negligible friction and leakage, when tested dry and then completly sealed by a film of lubricant. Similar pressure-tight bearing constructions are used in the Westcott (American Meter Company) meter and the Brown mechanical meter, both made in ratings to 2500 lb. per sq. in. In the Tagliabue meter, rated at 2500 lb. per sq. in., a pair of magnets mounted concentrically within and without a pressure-sustaining non-magnetic tube serve in place of the lapped bearing of Foxboro. In the Brown electric flow meter, made in ratings to 5000 lb. per sq. in., the float raises and lowers a magnetic core in a pressure-sustaining non-magnetic shell, outside which is a split inductance coil. A similar arrangement of coil and core in an indicator is at balance only when the positions of the two cores in the coils correspond. In the Republic meter, the rise of the mercury is measured by means of resistances which are successively short-circuited as the mercury rises. The mercury is covered by oil and must be kept very clean. By arranging the resistances so that the short-circuiting points rise in a parabola above the mercury, the current is made to increase stepwise as the square root of the mercury rise and so directly as the flow. A voltage-compensated ammeter —really an ohmmeter—serves as the indicator, and a voltage-compensated, ampere-hour (or watt-hour) meter as an integrator.

The advantages and limitations of the various types influencing their selection for specific applications are difficult to generalize upon. The mechanical meters are not inherently remote-indicating types like the electric meters, but they can be equipped with pneumatic or electric transmission when required. The mechanical meters are usually limited by the unsuitability of available bearing lubricants for specific process conditions. The non-magnetic tube meters are limited by the rapid decline of the transmitted force or torque with the increasing tube thickness required for higher pressures. The contact meters are limited by their stepwise action, especially in the lower part of the scale, and possibly by problems of keeping contacts clean without oil or of avoiding foaming of seal oil on major pressure changes.

The control of flow involves, of course, the same instrument mechanism that is used for pneumatic or electric control of other variables, but the translation of control action to flow changes is a real problem at high pressures. Valve designs are entirely special and very costly, and the variation of compressor or circulating-pump through-puts by bleeding back to the suction is wasteful of power and costly in valve maintenance. Similar results with economy are achieved at constant-drive speed by variable-clearance compressors. Since increased clearance means that the piston travels back farther before the pressure becomes low enough to open the intake valves, this means that a smaller portion of the stroke is actually used to bring in fresh gas, or that less fresh gas is brought in. Accordingly, the output of the compressor is decreased. The control of flow is also readily accomplished by the use of variable-speed motors, which have been brought to a high degree of perfection for alternating as well as for direct current.

Level. The measurement of the position of the interface between gas and liquid at high pressures may be accomplished by balancing the head of liquid against a column of mercury. Liquid level gages identical with the types of mercury-manometer flow meters described above are used to indicate the level of the meniscus. As the operation of these level gages in no wise differs from those used at low pressures, there is no need for describing them at length. A second very useful method consists in mounting on scales either the main receptacle or a smaller vessel connected with the main body by flexible tubing. A third method involves mounting one or more suitable conductivity cells in the vessel. These operate signal lights or electrical indicating instruments when wet with process liquid.

The level may be controlled by pneumatically or electrically operated valves actuated by a gage. This kind of control makes it possible to separate a gas and a liquid under pressure, drawing off one and recirculating the other. There is often a definite advantage in this: it saves recompression costs, the removal of a reaction product often permits the reaction to be carried further in another vessel so that the concentration of the desired product is built up, and there is less handling loss due to recirculation of unconverted material later in the process. Moreover, if 90 per cent of the gas can be removed at a total pressure in comparison with which the vapor pressure of the liquid is negligible, the vapor carried off with the remaining 10 per cent when the pressure is released will be a relatively small amount.

Pressure Drop. Pressure drops in a high-pressure synthesis system may be very significant, and simply placing two record pens on the same chart may not be sufficiently accurate or direct-reading. For moderate pressure drops the usual flow meter serves as a direct-reading pressure-drop instrument, but at higher drops range tubes 2 in. high per pound differential become unwieldy. In such cases readings transmitted from scattered static-pressure instruments can be shown on a single direct-reading pressure-drop recorder.

Safety

When reasonable precautions are taken, high-pressure plants can be made as safe as any. There are a few hazards peculiar to work at high pressures that it seems worth while to enumerate.

1. At least one fatality and several accidents are known to have resulted from split Bourdon tubes. The fatality is believed to have been the result of a rush of compressed inflammable gas into a Bourdon tube containing air. The minor explosion which occurred split the tube and a particle flying from the gage case was responsible for the death of a man. To permit the escape of gases, the cases of Bourdon gages should be provided with large vent openings covered with paper or foil for dust protection. Whenever possible the gage glass should be removed and the face left open or, for dust protection, the glass replaced with thick transparent sheeting or with safety glass so that the danger from flying particles will be avoided. Bourdon gages should be placed above eye level. In some laboratories it is required that Bourdon gages be placed behind a barrier and read only by their reflection in mirrors so that the observer is at all times out of the direct line of danger. The usual procedures should be followed, such as the use of oil seals to prevent corrosion, throttling down to prevent fatigue, and the use of built-in checks to prevent the pressure from rising or falling too suddenly.

2. With inflammable gases at high pressures, the Armstrong effect is a possible source of trouble: when a gas containing finely divided liquid or solid particles passes at high velocity over an insulated metallic object, that object

becomes electrically charged. The charge may become sufficiently great to cause a spark which in turn ignites the gas.

3. When metallic containers rupture, frictional effects may cause very high local temperatures and the ignition of inflammable gases.

4. It is rarely possible to cool a direct-fired vessel and its setting in a hurry. Such apparatus should be followed by a by-pass leading to the atmosphere so that if a failure or a plug occurs farther along in the process the by-pass can be opened and material kept running through the direct-fired apparatus while it cools.

5. Where it is not possible to use a self-seating safety valve, it is sometimes possible to have two safety valves: one set slightly above the operating pressure with a shut-off valve between it and the body to be protected, and the other set somewhat higher and without the shut-off valve. When the lower safety valve releases, it can be reseated with the shut-off valve closed, while close observation and the higher safety valve provide the necessary security.

6. A variety of pressure relief that has met with favor because of its positive action and because it cannot readily be prevented from blowing off when the pressure exceeds a certain value is the rupture disk, which consists of a plate held in place over an opening by means of flanges. When the pressure rises too high, the disk ruptures. The disadvantages of the method are that the entire contents of the pressure system are discharged, and that corrosion may so weaken the disk that unless frequently inspected and renewed it may burst at too low a pressure; on the other hand, since the working pressure produces stresses close to the yielding stress of the material of the disks, they tend to become strain hardened with continued use and may then fail to relieve the pressure at the desired point.

7. Carbon monoxide rapidly attacks pure nickel, forming nickel carbonyl, and even attacks iron, though much more slowly. When carbon monoxide is to be used, the proper selection of alloys is important. It is also necessary to warn against the use of mercury in any apparatus in which it can come in contact with copper, brass, or other metals with which it can amalgamate. The general belief that mercury does not wet steel or form alloys with it is incorrect: Bridgman found that mercury can wet surfaces made by breaking steel in mercury. This is of great importance where there is any possibility of a crack in a mercury container opening slightly under pressure, allowing the mercury to wet the steel and start to undermine the walls.

8. Some catalysts, notably very active nickel or iron powders, are highly pyrophoric. Where there might be a danger of their leaking through valves or stuffing boxes, drying out and glowing, it is desirable to provide a constant stream of water to wash them away so that any inflammable vapors or gases that accompany them may not catch fire.

9. The larger and hotter pieces of high-pressure equipment may be placed to advantage behind heavy barricades, for in case of failure, heavy walls and light roofs give protection to the surroundings. At the same time it should be urged that anything permitting the formation of gas pockets is to be avoided: the buildings in which high-pressure apparatus is housed should permit the free passage of air through all parts, especially up under the roof. Where poisonous or inflammable gases are used, additional protection for the operators should be provided by the installation of forced ventilation. This system is, however, worse than useless if the draft is too strong or its temperature too high or too low.

10. Since leaks are a serious fire or health hazard, they should be repaired as promptly as possible, especially as erosion quickly makes the leak worse and

may cut into the metal so severely as to require replacement of a portion of the apparatus. Leaks should not be tightened under pressure.

11. Welded apparatus for very low temperatures must be exceptionally well annealed. Several split welds in high-pressure equipment cooled with liquid air have shown how serious a cause of accident this might be.

12. Oxygen cannot be compressed with safety in the presence of oil [Hersey, A Study of the Oxygen-Oil Explosion Hazard, *J. Am. Soc. Naval Eng.*, **36**, 231 (1924)], so that water must be used as a lubricant. It has been found that when oxygen is admitted rapidly at high pressures into a space containing a bit of oily material explosions may ensue. The explosion in the Bourdon tube mentioned above may have been of this nature. Every reason indicates that pressure should always be built up slowly.

13. A word of caution is needed because of a common factory usage according to which *oxygen* is called *air*. Since compressed air may be used with oil at pressures much higher than are safe with oxygen, the substitution of oxygen when compressed air was desired has been the cause of accidents.

14. The periodic inspection of high-pressure equipment is a very real necessity and is a matter of routine in all high-pressure plants. Apparatus should be constructed to permit inspection of the inside surfaces and measurement of the outer dimensions. Sections of piping should be removed and inspected for changes in dimensions and for the growth of longitudinal cracks formed in drawing.

SECTION 20

MOVEMENT AND STORAGE OF MATERIALS*

BY

G. L. Montgomery, M. E., Licensed professional engineer, New York; member, American Society of Mechanical Engineers; formerly of the editorial staff of *Chemical and Metallurgical Engineering,* Associate Editor, *Food Industries.*

George A. Prochazka, Jr., E. M., Consulting Engineer; formerly Chemical Economist, E. I. du Pont de Nemours & Co.; formerly Manager, Central Dyestuff & Chemical Company; Member, American Institute of Chemical Engineers, American Institute of Mining and Metallurgical Engineers; Author, "Accounting and Cost Finding for the Chemical Industries."

R. W. Lahey, B. S. American Cyanamid Company; Chairman, Carboy Committee of the Manufacturing Chemists Association; Member, Steel Barrels and Drums Committee of Manufacturing Chemists Association, Poisonous Articles and Miscellaneous Packages Committee of the Manufacturing Chemists Association, Committee D-6 of the American Society for Testing Materials.

CONTENTS

PAGE

MOVEMENT AND STORAGE OF MATERIALS

Movement of Solid Materials and Containers

PAGE

Movement of Liquids

---

* Figures 32 to 50, except Figs. 34 and 49, from Badger and McCabe, "Elements of Chemical Engineering."

PAGE

Movement of Gases and Vapors

Storage of Materials

FEEDERS AND FEEDING MECHANISMS

Feeding Gases:

Reagent Feeders:

PAGE

PACKAGING AND FILLING EQUIPMENT

General:

Dry Materials:

Liquids:

MOVEMENT AND STORAGE OF MATERIALS

BY G. L. MONTGOMERY

REFERENCES: "Material Handling Cyclopedia," Simmons-Boardman Co., New York, 1921. Tournier, "Materials Handling Equipment," McGraw-Hill, New York, 1930. Koshkin, "Modern Materials Handling," Wiley, New York, 1932. Merriman, "Treatise on Hydraulics," 10th ed., Wiley, New York, 1916. Lea, "Hydraulics," 5th ed., Longmans, Green, New York, 1930. "Pump Handbook," Worthington Pump & Machinery Corporation, Harrison, N. J. Swindin, "Modern Theory and Practice of Pumping," Van Nostrand, New York, 1924. Carrier (ed. by Madison), "Fan Engineering," 4th ed., rev. and enlarged, Buffalo Forge Co., Buffalo, 1938. Lucke, "Engineering Thermodynamics," McGraw-Hill, New York, 1912. Badger and McCabe, "Elements of Chemical Engineering," McGraw-Hill, New York, 1931. Peele, "Mining Engineers' Handbook," 2d ed., Wiley, New York, 1927. Marks, "Mechanical Engineers' Handbook," 4th ed., McGraw-Hill, New York, 1941.

General. The chemical engineering industries, as soon as they emerged from small-scale batch processes, were forced to adopt whatever mechanical handling equipment was available. The high proportion of liquids among the materials handled and the nature of many of the materials made this course necessary. These same factors caused a rapid development of handling methods in the chemical engineering field. More recently, progress in other industries has developed many new handling methods, and these have been applied in chemical engineering practice as soon as available. In this way, conveying and lifting equipment has been adapted from the metal-working industries and pumping equipment from the food field and from power-generating practice.

Movement and Storage of Materials. This unit process of chemical engineering comprises the handling and storing of all materials, solids, liquids, and gases, during the course of manufacturing. It takes the materials into, through, and out of the plant.

It has been estimated that from 20 per cent to as high as 90 per cent of all manufacturing costs in the chemical engineering industries is made up of the cost of moving and storing materials, depending upon the particular industry scrutinized. Economy in the conduct of this unit process is therefore of great importance.

In general, it may be stated that economy can be obtained by the substitution of machine operation for hand labor. Where a machine is already used, the substitution of a more economical machine will lessen costs. This is in large part due to the rapid increase in labor costs in recent years. However, in making such substitutions, care must be taken to select the equipment most suitable to the operation to be performed.

Economy in manufacture is in part based on holding at a minimum inventories of materials in storage and in process. This means that materials should make the most rapid progress possible through the plant. Fully mechanized handling of materials and storage operations are the best means of attaining this end.

In order to realize the maximum advantage from the use of mechanical equipment for moving materials, it is necessary to adjust all the work, the operation of every step in the process, to the most advantageous operating rate of this equipment. When this is done, the count of material units passing any point is a measure of performance and makes possible accurate manufacturing control.

MOVEMENT OF SOLID MATERIALS AND CONTAINERS

Solid materials moved in the chemical engineering industries include raw materials, materials in process, and finished products.

Raw solid materials include quarry products such as limestone, phosphate, dolomite, shale, clay, gypsum, bauxite, and sand; mined products such as salt, sulfur, coal, pyrites, and metallic ores, and minerals such as feldspar, pigments, asphalts, waxes, and pitches; vegetable products such as wood, cotton, cotton seed, flax seed, palm kernels, soya beans, copra, raw sugar, and sugar beets; cellulose-bearing materials such as bagasse, straw, and pulp; crude rubber; and animal products such as hides, fats, and bone.

Solid materials entering a process often become **fluid,** while materials entering a process as fluids often become solid. **Solid materials in process** are powdered, granular, or bulky in form, and their nature will depend upon the processing which they have undergone. This must be determined by a study of the particular industry to which the materials-handling equipment is to be applied.

Solid finished products of the chemical engineering industries include such materials as coke, charcoal, sugar, salt, soap, pottery, porcelain, brick, explosives, fertilizer, glass, gelatin, caustic, leather, lime, cement, grease, pigments, paper, pulp, paraffin, rubber products, dyes, and drugs.

Containers are used for holding *solid*, *liquid*, and *gaseous materials*, whether these are raw materials, materials in process, or finished products. Common types of containers, all of which must be handled as though they were solids, include bales, bags, boxes, kegs, barrels, drums, bottles, carboys, tanks, packages, saggers, and special skid platforms.

Four general classes of **equipment** are available **for handling solid materials** and materials in containers. These are:

1. Conveyors.
2. Overhead equipment, as hoists or cranes.
3. Trucks, tractors, and trailers.
4. Industrial railways.

1. Conveyors

Conveyors are usually permanent structures and are frequently integral parts of the manufacturing equipment. Portable conveyors of various types are also available for certain classes of work.

Types of conveyors, classified in accordance with the direction of movement for which these are primarily suited, are:

Horizontal Movement.
- Belt conveyors.
- Chain or cable conveyors.
- Chain trolley conveyors.
- Platform, slat, apron, and pan conveyors.
- Gravity roller conveyors.
- Wheel conveyors.
- Live, or driven, roller conveyors.
- Bucket carriers.
- Scraper, flight, pusher-bar, and drag conveyors.

- Redler conveyors.
- Screw conveyors.
- Johns conveyors.
- Drag-line scraper conveyors.
- Pneumatic conveyors.
- Solids pumps.
- Vibrating conveyors.

Vertical Movement.

- Bucket elevators.
- Skip hoists.
- Chain elevators.
- Barrel, package, bag, and tray elevators.
- Bucket carriers.
- Belt elevators.
- Spiral chutes.
- Roller spirals.
- Chutes or spouts.
- Redler conveyors.
- Pneumatic conveyors.
- Johns conveyors.
- Solids pumps.

Combined Vertical and Horizontal Movement.

- Inclined belt conveyors.
- Inclined chain conveyors.
- Inclined chain trolley conveyors.
- Inclined scraper flight, drag, and pusher-bar conveyors.
- Inclined platform apron slat and pan conveyors.
- Inclined live roller conveyors.
- Inclined gravity roller conveyors.
- Inclined-wheel conveyors.
- Inclined screw conveyors.
- Redler conveyors.
- Bucket carriers.
- Chutes.
- Pneumatic conveyors.
- Johns conveyors.
- Solids pumps.
- Vibrating conveyors.

These types of equipment are defined as follows:

Belt conveyors (Fig. 1) consist of endless belts carried on a driving pulley at one end of their run and an idler pulley at the other end, and supported at appropriate intervals at both the top and bottom parts of the run by rollers. Material is carried on the upper surface of the belt in either or both parts of the run but usually on the upper part. Driving equipment, take-ups to keep the belt at the proper tension, and loading and unloading devices such as belt trippers and feeders* are the important accessory devices.

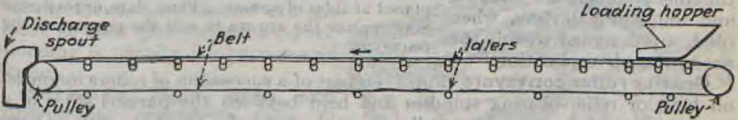


FIG. 1.—Simple type of horizontal belt conveyor loaded at one end by hopper and discharging into spout; belt may be troughed or flat.

---

* See p. 2286.

Chain conveyors consist of endless runs of chain or cable carried on a driving sprocket, idler sprockets, and, by means of rollers, on guides and equipped with hooks, pushers, or other devices for engaging the material to be moved. Loading and unloading devices are not standard but, if used, are designed for the material to be handled. Take-ups and driving equipment are the chief accessories.

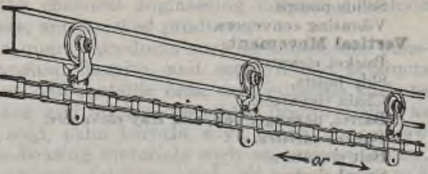


Fig. 2.—Chain links, roller, and guide as used in simple form of chain conveyor of trolley type for overhead service.

Chain trolley conveyors (Fig. 2) are a variant of the chain conveyor, adapted for overhead service by the use of trolleys which serve to support the conveyor and allow it to move along a suspended track of I-beam, channel, or inverted T-beam type. Loads are suspended from the chain, generally at points directly beneath the trolley attachments, by means of hooks or carriers. In general practice with these conveyors, the trolley serves to transmit the load to the track, while the chain serves as a means of moving the conveyor.

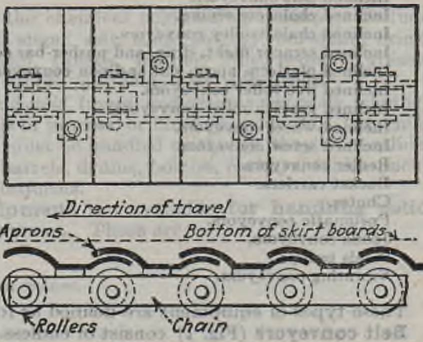


Fig. 3.—Sketch of part of simple type of apron conveyor. Two chains are used in larger sizes, placed at sides of aprons. Pans, slats, or platforms may replace the aprons to suit the purpose of the conveyor.

Platform, slat, apron, and pan conveyors (Fig. 3) consist of one or two parallel, endless runs of chain or cable, or of a belt, upon the outer surface of which are affixed platforms, aprons, slats, pans, or other receptacles. These conveyors are provided with driving and idler sprockets or pulleys at the ends of the run and suitable guides or supports for the top and bottom parts of the run. Driving equipment and take-ups are the chief accessories. Loading and unloading devices, when used, are designed to suit the needs of each application.

Gravity roller conveyors (Fig. 4) consist of a succession of rollers mounted on ball- or roller-bearing spindles and held between the parallel sides of a structural steel frame. Live roller conveyors or chain conveyors, such as the pusher-bar type, are frequently used in connection with the gravity roller conveyor to accomplish vertical lifts of the material.

Wheel conveyors are a special form of roller conveyor in which the rollers are replaced by wheels equipped with antifriction bearings. The usual form of this conveyor is made up of two side members in which the spindles of the wheels are so fixed that the rims of the wheels project above, providing a live surface over which objects may be pushed or may move by gravity. Structural steel supports are provided at appropriate intervals. Some wheel conveyors are provided with a third line of wheels in the center, to

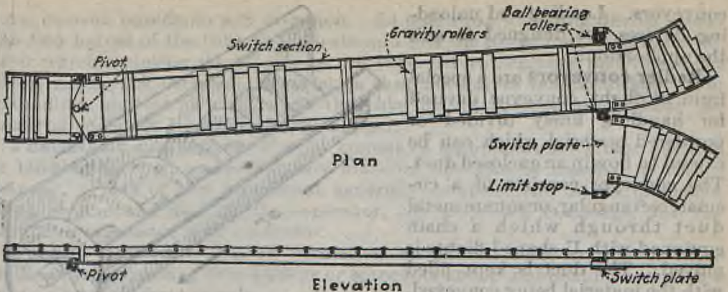


FIG. 4.—Gravity roller conveyor, showing switch section, indicating flexibility of such equipment.

give added support to the objects being conveyed. Wheel conveyors are generally used for handling bundles, flat-bottom containers, or other similar materials.

Live, or driven, roller conveyors are similar to gravity roller conveyors, with a means of positively rotating the rolls, such as bevel gearing, added. Accessories consist of the driving equipment.

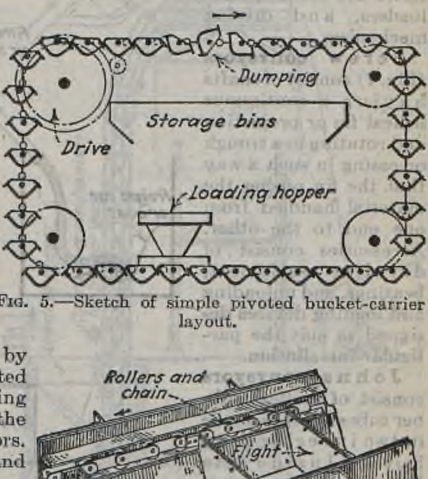


FIG. 5.—Sketch of simple pivoted bucket-carrier layout.

Bucket carriers (Fig. 5), whether of the V bucket, gravity discharge, or of the pivoted bucket type, consist of one or two endless chains employed as in chain conveyors but carrying a series of attached buckets which serve as receptacles for the material being handled. Buckets discharge by gravity or, in the case of the pivoted type, by means of a tripping arrangement. Accessories are the same as used with chain conveyors. Loading is usually by hopper and spout.

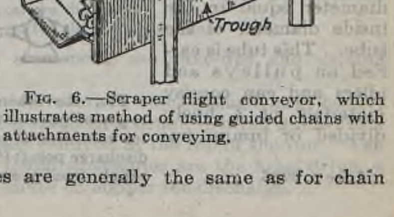


FIG. 6.—Scraper flight conveyor, which illustrates method of using guided chains with attachments for conveying.

Scraper, flight, drag, and pusher-bar conveyors (Fig. 6) consist of one or more chains, in general, similar to those making up chain conveyors but passing through a trough or set of guides, so that the motion of the chain, either alone or assisted by attachments such as flights, moves the material that is placed in the trough or between the guides. Accessories are generally the same as for chain conveyors. Loading and unloading devices are designed to suit the application.

Redler conveyors are a special form of flight conveyor devised for handling finely divided or powdered material which can be caused to flow in an enclosed duct. The conveyor consists of a circular, rectangular, or square metal duct through which a chain equipped with U-shaped flights is moved. The duct is kept filled with the material being conveyed, and the movement of the flights causes this column of material to move along the duct. Movement can be in any direction. Accessories are feeders,* unloaders, and driving mechanism.

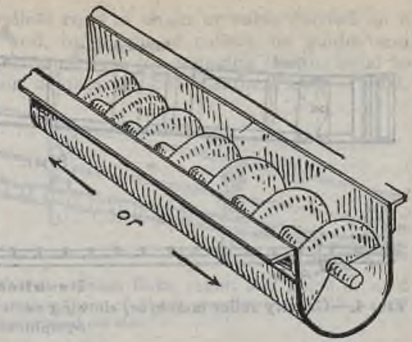


Fig. 7.—Sketch of part of screw conveyor in open trough.

Screw conveyors (Fig. 7) consist of shafts having a continuous helical fin or projection and rotating in a trough or casing in such a way that the fin pushes the material handled from one end to the other. Accessories consist of driving equipment, bearings, and unloading and loading devices designed to suit the particular installation.

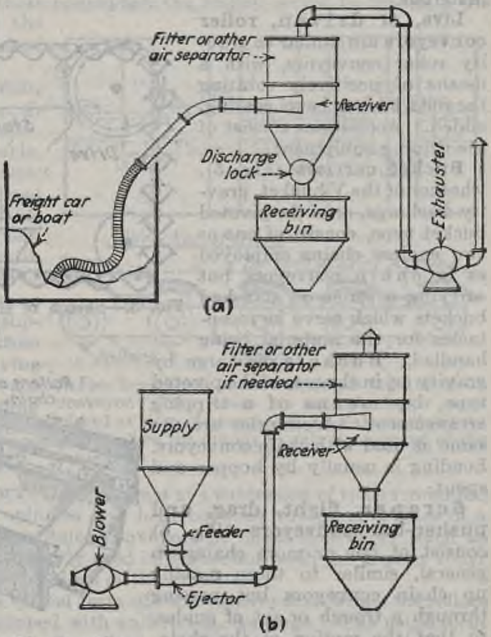


Fig. 8.—Schematic diagrams of two most common types of pneumatic conveying systems. (a) Suction type for unloading or taking a supply from several points to one discharge point; (b) pressure type used principally in taking supply from one point to several discharge points.

Johns conveyors consist of endless rubber tubes or pipes, made in two interlocking halves. Inside one half, at regular intervals, are fixed circular rubber partitions of a diameter equal to the inside diameter of the tube. This tube is carried on pulleys and idlers and can convey materials of a finely divided or lumpy na-

---

* See p. 2286.

ture, or even liquids, in any direction. In operation a loading device causes the two halves of the tube to separate and fills the lower half with material, after which a closing device brings the tube halves together and interlocks them, giving the tube a slight twist so that the halves remain locked. For unloading, another device causes the tube halves to separate so that the material spills out into a discharge hopper.

Pneumatic conveyors (Fig. 8) consist of equipment for either sucking or blowing through a pipe line any material that can be handled in such a manner. Parts of this equipment generally comprise the intake or feeder, the pipe line and fittings, the separator, the air-locked discharge, and the blower or exhauster with its motor.

Bucket elevators (Fig. 9) consist of an endless belt, or one or more endless chains, passing over pulleys or sprockets at top and bottom and with a succession of buckets attached for holding the material that is conveyed. There is usually a specially designed loading hopper, called a "boot," a casing, and a discharge hopper or chute. Usual accessories are take-ups and driving mechanism.

FIG. 9.

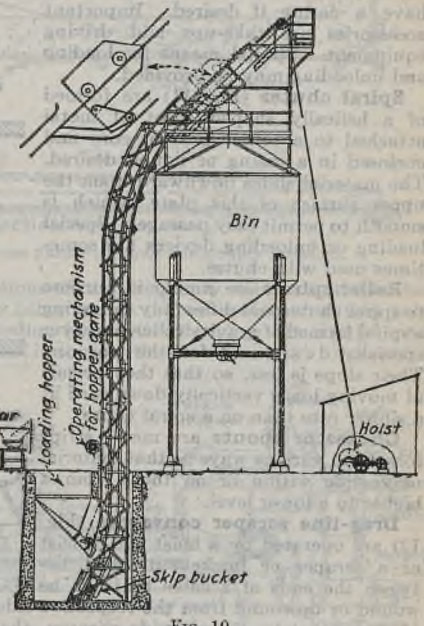


FIG. 10.

FIG. 9.—Typical bucket elevator with spaced buckets and centrifugal discharge. Other types have continuous buckets or positive discharge.

FIG. 10.—Skip hoist, used for elevating materials to storage bins.

Skip hoists (Fig. 10) are automatic elevators in which a single large bucket or "skip" is raised by the winding of a cable on a drum, it being guided in a structural steel frame and automatically dumped at the top of the run. The hoist is designed for quick return. Usual accessories are the hoist drive, a feeder for loading, and some type of chute or hopper for discharge.

Chain elevators, barrel, package, and tray elevators, are all variations of the same mechanism, which consists of one or more endless chains running over sprockets at top and bottom, with or without a casing, as desired, and equipped with some type of engaging device for supporting barrels, packages, trays, or other material. Usual accessories are driving mechanism and take-ups. Loading and unloading devices are generally special.

Belt elevators consist of an endless belt, running over pulleys at top and bottom and having cleats or some other means of moving material affixed to the outer surface. These elevators may have a casing if desired. Important accessories are take-ups and driving equipment. Special means for loading and unloading may be provided.

Spiral chutes (Fig. 11) are formed of a helically shaped sheet of metal attached to a center post or core and enclosed in a casing or not as desired. The material slides downward upon the upper surface of this plate, which is smooth to permit easy passage. Special loading or unloading devices are sometimes used with chutes.

Roller spirals are similar in purpose to spiral chutes and differ only in having a spiral formed of gravity roller conveyor specially designed for the purpose. Their slope is less, so that they permit of moving loads vertically downward at a slower rate than on a spiral chute.

Chutes or spouts are metal strips formed in various ways so that material may slide within or on them from a higher to a lower level.

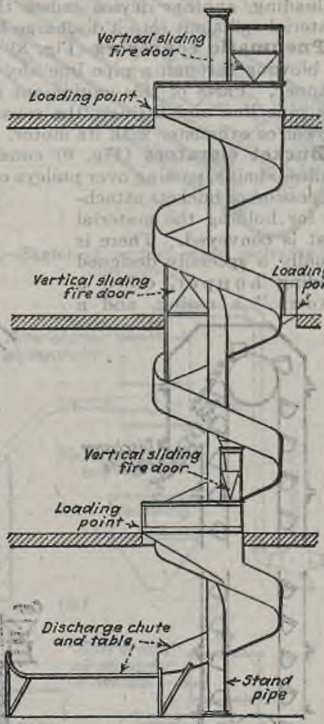


Fig. 11.—Open-type single-blade spiral chute. Double-blade and enclosed types are available.

Drag-line scraper conveyors (Fig. 12) are operated by a hoist and consist of a scraper or bucket attached between the ends of a cable that can be wound or unwound from the respective sides of a double drum. By means of appropriately placed guide sheaves, the scraper is caused to drag over and pick up the material to be moved. This conveyor can be used for storing material or reclaiming it from storage, depending upon the point of loading and unloading. It is generally used in conjunction with other conveyors.

Solids pumps (Fig. 13) may be considered as combinations of screw conveyors and pneumatic conveyors but are actually neither. Material of a finely divided nature, or made up of reasonably small pieces, is fed by a screw through an air chamber where numerous jets of air are introduced from a ring around the pump casing. This aerates the material, expands it, and causes it to "flow," thus increasing the ability of the screw to force it through a pipe line to the point of delivery. Usual accessories are a drive for the screw, special valves for the line, and a compressor for air supply.

Vibrating conveyors consist of open or closed troughs, or tubes, in which material is conveyed horizontally, or up or down inclines. Movement of the material is caused by vibrations of high frequency applied to the bottom of the conveyor by electrical or other means. Feed can occur at any point along the conveyor, through feeding hoppers. Discharge can also be made through bottom gates at any point in the length of the conveyor. Accessories are feed and discharge controls and mechanism for applying vibration to the conveyor bottom. These conveyors are suitable for handling coarsely or finely divided materials in bulk, under closely controlled conditions.

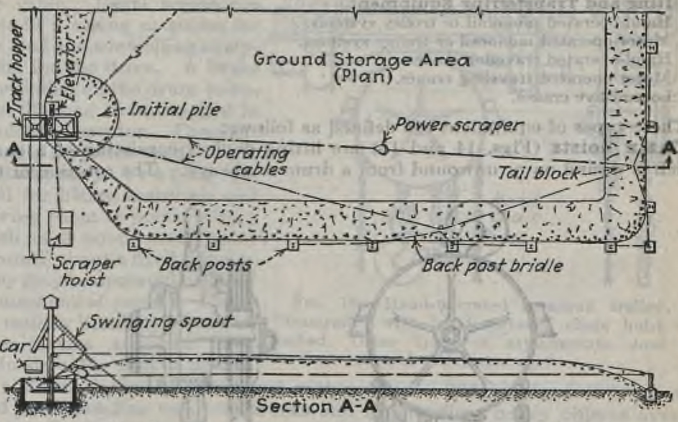


FIG. 12.—Layout of storage yard with drag-line scraper installed.

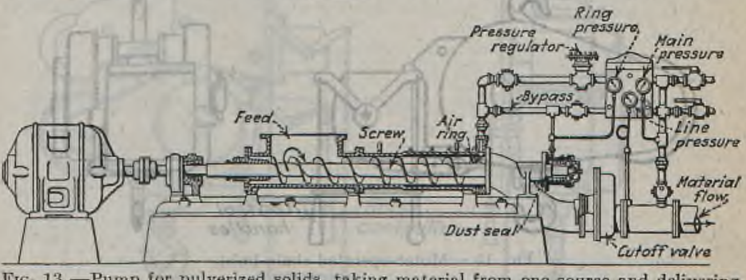


FIG. 13.—Pump for pulverized solids, taking material from one source and delivering to several points by pipe.

2. Overhead Equipment, as Hoists and Cranes

Overhead equipment is of two classes, that which only lifts materials, and that which transports as well as lifts.

Lifting Equipment.
- Hand chain hoists.
- Motor chain hoists.
- Drum hoists and platform elevators.
- Stationary cranes.

Lifting and Transferring Equipment.
- Hand-operated monorail or trolley systems.
- Motor-operated monorail or trolley systems.
- Hand-operated traveling cranes.
- Motor-operated traveling cranes.
- Locomotive cranes.

These types of equipment are defined as follows:

Chain hoists (Figs. 14 and 15) are lifting devices consisting of a chain which is wound on or unwound from a drum or sheave. The rotation of the drum is caused by a hand chain that passes over a sprocket; or by an electric motor, acting through a gear train upon the drum. The whole hoist is supported from above, either from a ceiling beam or from a structure built for the purpose. The hoisting chain is provided with a hook or other means of engaging the load.

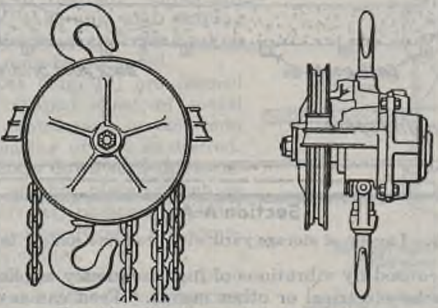


Fig. 14.—Hand-operated chain hoist.

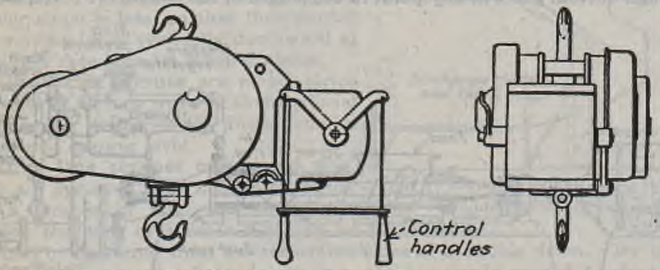


Fig. 15.—Motor-operated chain hoist.

Drum hoists and platform elevators operate on the same general principle as chain hoists, except that a wire rope or cable is usually substituted for the chain. The drum is frequently located at the lower floor level or ground level, with the hoisting cable passing through a sheave or guide pulley at the upper level. Steam engines, internal-combustion engines, or electric motors are used for driving these hoists or elevators, generally through a gear train, although hand power is sometimes used. A cage, platform, hook, or other means of lifting material is attached to the end of the cable. Usual accessories include a casing or guides for the load and controlling equipment for the drive. A brake is provided on the drum to enable the load to be stopped in the desired position. Counterweights are often used.

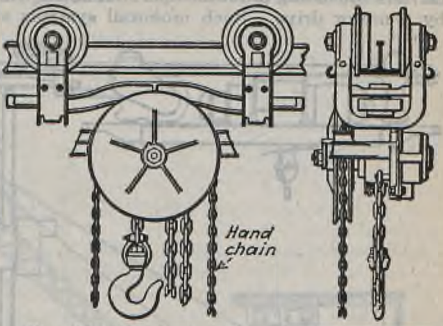


FIG. 16.—Hand-operated monorail trolley, or "tramrail," with hand-operated chain hoist attached. Other types of attachments, such as carriers, are used with monorails.

Stationary cranes are used for lifting materials and moving them to points within reach of a boom constructed to rotate about a fixed column. They may be operated by hand or mechanical power. Loads are usually lifted by a hook or other device attached to a chain or cable that is wound on a drum similar to that used with a hoist. This equipment is mostly used for unloading cars and trucks and for handling heavy objects over a restricted area.

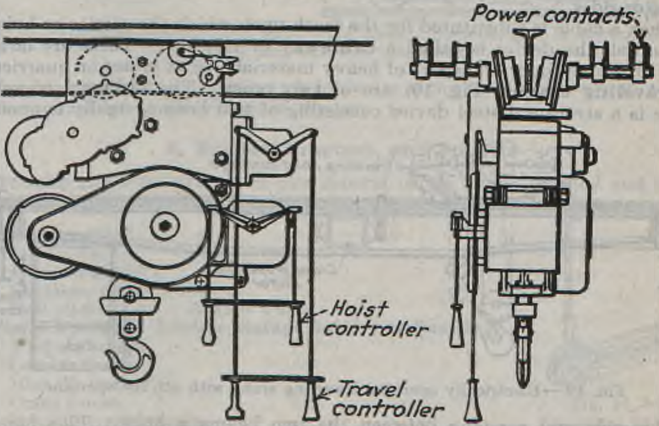


FIG. 17.—Electrically operated monorail trolley with motor-driven chain hoist, arranged for operation from floor.

Monorail or trolley systems (Figs. 16, 17, and 18) are means of conveying materials along an overhead trackway consisting of a single rail or track of various suitable designs. The simplest form of such equipment consists of a light track suspended from a ceiling or suitable supporting structure, upon which operate various designs of wheeled carriers propelled by hand. One of the most used forms has for a carrier a hand or motor-operated chain hoist, propelled by hand. The more complicated forms of monorail systems have carriers consisting of motor-operated hoists that are propelled along the track by a motor drive. Such monorail systems are operated from the floor in the smaller sizes, while the larger types are provided with a cab in which the operator rides.

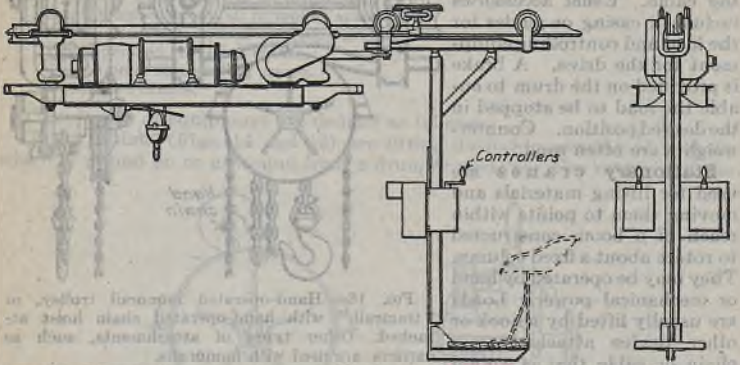


Fig. 18.—Electrically operated monorail trolley with motor-driven hoist and seat for operator.

When a cable is substituted for the track upon which the carrier or hoist is supported, the device is called a **cableway** or **telpher**. These are largely used for the outdoor transport of heavy materials, as at mines or quarries.

Traveling cranes (Fig. 19) are of two types. The ordinary traveling crane is a structural steel device consisting of two beams, rigidly connected at the ends and carrying between the two beams a hoist. This hoist is arranged to move along the structure on tracks. The whole device is mounted at the ends on rails in such a manner that it can be moved at right angles to the direction of travel of the hoist. Such cranes are operated either by hand or by motor drive, or are part hand and part motor driven. **Gantry** or **bridge** cranes are modifications of traveling cranes in which the whole crane is elevated above the tracks upon which it travels by means of structural

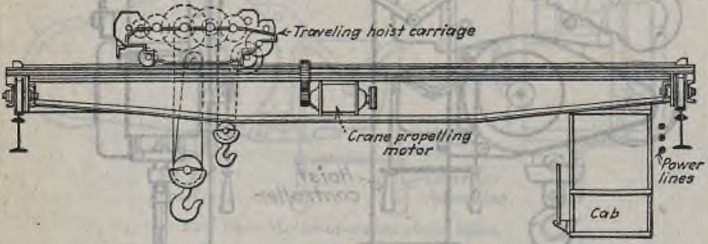


Fig. 19.—Electrically operated traveling crane with cab for operator.

supports. These latter cranes are adapted to service in outdoor storage yards where no overhead supporting structure is conveniently available.

Locomotive cranes (Fig. 20) are rail cars with a crane of the stationary, revolving-boom type mounted upon them. The driving mechanism of the crane is arranged to propel the whole device along the track through a gear-train connection to the wheels of the car.

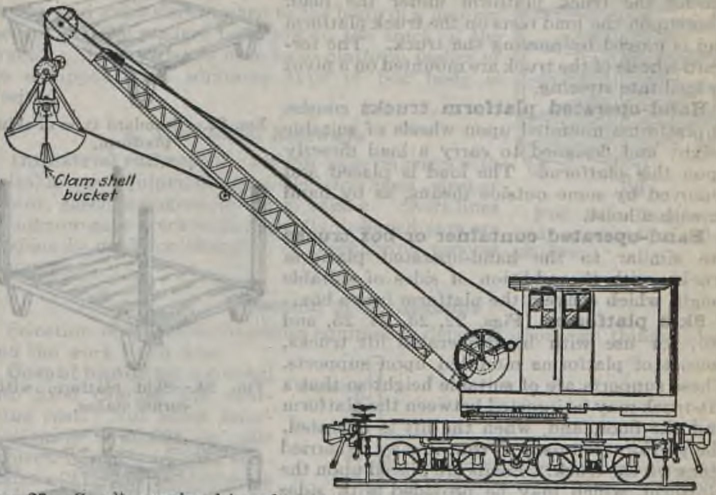


Fig. 20.—Gasoline-engine-driven locomotive crane. Steam-engine or electric drive is also used with this equipment.

3. Trucks, Tractors, and Trailers

Trucks and tractors are of two general types, hand operated and power operated.

Hand Operated.
- Lift trucks.
- Platform trucks.
- Container or box trucks.
- Skid platforms used with lift trucks.

Power Operated (Electric Storage Battery or Gasoline).
- Platform trucks.
- Low-lift trucks.
- High-lift trucks.
- Crane trucks.
- Special-purpose trucks.
- Skid platforms for lift trucks.
- Tractors.
- Trailers for tractors.

Fig. 21.—Hand-operated lift truck in which lift is accomplished by movement of truck handle or tongue.

These types of equipment may be defined as follows:

Hand-operated lift trucks (Fig. 21) are low platforms mounted on wheels and propelled by the operator, who pulls on the handle provided for that purpose. These trucks are provided with a hand-operated lifting mechanism, by means of which the platform can be elevated a few inches. The purpose of this is that the truck can be pushed beneath a load standing on low legs and these legs then raised from the floor by raising the truck platform under the load. Thereupon the load rests on the truck platform and is moved by moving the truck. The forward wheels of the truck are mounted on a pivot to facilitate steering.

FIG. 22.—Standard type of skid platform.

Hand-operated platform trucks consist of platforms mounted upon wheels of suitable height and designed to carry a load directly upon this platform. The load is placed and removed by some outside means, as by hand or with a hoist.

Hand-operated container or box trucks are similar to the hand-operated platform trucks, with the addition of sides of suitable height which convert the platform into a box.

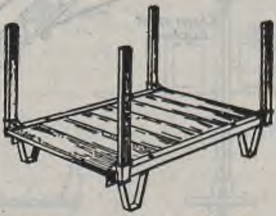


FIG. 23.—Skid platform with corner stakes.

Skid platforms (Figs. 22, 23, 24, 25, and 26), for use with hand-operated lift trucks, consist of platforms mounted upon supports. These supports are of suitable height so that a lift-truck may be inserted between the platform and the floor, and, when the lift is operated, the platform is raised so that it may be carried along on the truck. Loads are placed upon the platform, which may be provided with sides as needed. If these platforms are supplied with wheels in place of skid supports, they may be moved by hand and are more suitable to installations where flexibility of storage is needed. It is usual to store loads upon skid platforms in plants where the trucking system is employed, thus avoiding rehandling. In some cases, loads are shipped on skids by rail or boat, from one plant to another, to facilitate loading and unloading and reduce handling.

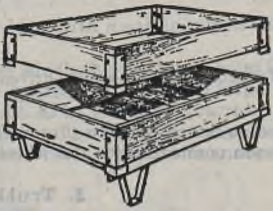


FIG. 24.—Sectional box or bin body on skid platform.

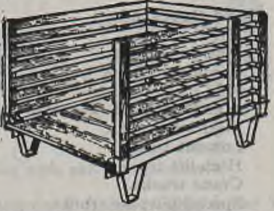


FIG. 25.—Slat-type body mounted on skid platform.

Power-operated trucks (Figs. 27, 28, and 29), similar in purpose to hand-operated trucks, are operated by electric motors, supplied from storage batteries, or by gasoline motors. These trucks are available in a **platform truck**, a **low-lift truck** for handling skid platforms, a **high-lift truck** for raising loads to storage or working level, **crane trucks** equipped with a crane for raising loads and somewhat similar in purpose to a locomotive crane, and many types of **special-purpose trucks,** designed to suit the needs of specific jobs, such as charging furnaces, lifting large newsprint rolls, or casting metal.

Skid platforms for power-lift trucks are similar to those described for hand-lift trucks, except as to dimensions which are sometimes different.

Tractors (Fig. 30) are storage-battery electric or gasoline motor-driven trucks, designed for the purpose of pulling one or more loaded **trailers** (Fig. 31). These trailers are similar to hand-operated platform trucks but are arranged to be connected in trains, and axles are attached to the body in a manner permitting the trailers to follow in the track of the tractor. These trailers may have platform bodies or be equipped with whatever type of box body is desired.

4. Industrial Railways

Industrial railways are narrow-gage railways with suitable car equipment, propelled by hand, electric motor, gasoline engine, or steam engine. Short lines of narrow-gage track with specially designed cars are frequently used for charging or discharging process equipment.

Fig. 26.—Rack-type body mounted on skid platform.

Selection of Equipment

Selection of materials-handling equipment depends upon two things, cost and the work to be done.

Cost of handling is divided into fixed charges and operating cost. Not much information is available on this subject. Cost and cost estimates will be discussed at some length at the conclusion of the more specific discussion of type selection.

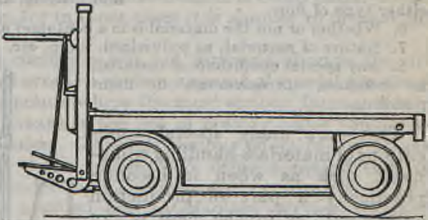


Fig. 27.—Electric storage-battery industrial truck of platform type.

Preliminary estimates for materials-handling equipment should be based on a selection made in accordance with the work to be done. Before purchase of equipment is made, this choice should be reviewed by a competent consulting engineer or by the engineer of the company that is to supply the equipment and final estimates and selection made in the light of his recommendations.

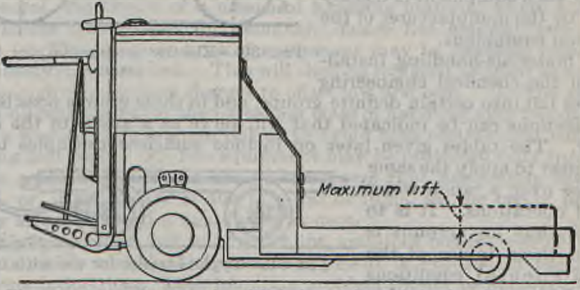


Fig. 28.—Electric storage-battery low-lift industrial truck for handling skid platforms.

The first step in selection of materials-handling equipment, therefore, is to study the work to be done. From this study the following things are determined:

1. The movement to be made, whether horizontal, vertical, or a combination of the two; and the distance covered.
2. The quantity to be moved per hour, per day, or for any other convenient unit of time, in weight, number of pieces (if this factor applies), or volume (where that is of importance).
3. How the material is to be delivered to the handling equipment.
4. How the material is to be discharged by the handling equipment.
5. Whether continuous or intermittent flow is possible, including whether the use of storage is available as an aid to obtaining either type of flow.
6. Whether or not the material is in a container and, if so, description of the container.
7. Nature of material, as pulverized, lump, etc.
8. Any special conditions of material, as moisture, corrosiveness, or flammability.

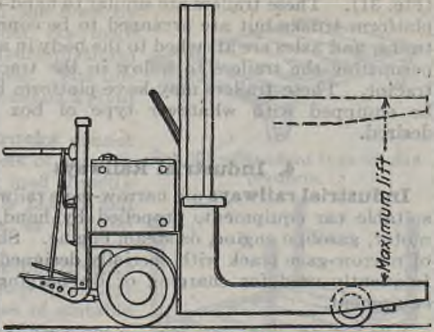


FIG. 29.—Electric storage-battery high-lift industrial truck for handling skid platforms, unloading and loading, and storing containers.

There are many special applications of materials-handling equipment such as when it becomes practically a part of production equipment, as in the case of an annealing lehr with a conveyor hearth. These cannot be discussed, as to selection, in any general consideration of the subject. In any case, such equipment is usually selected by the manufacturer of the production equipment.

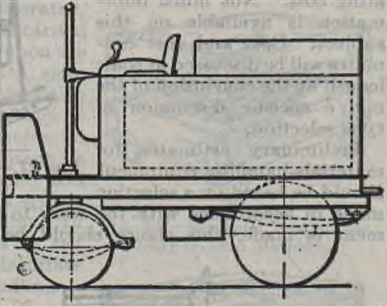


FIG. 30.—Electric storage-battery industrial tractor.

Most materials-handling installations in the chemical engineering industries fall into certain definite groups, and in these groups possible equipment selections can be indicated that will serve as a guide to the chemical engineer. The tables given later on include sufficient examples to enable the engineer to apply the same principles over a much wider range of operations. It is to be noted that no account is taken in these tables of the effect of special conditions such as high moisture content, corrosiveness, etc. Where these are present it is advisable to have the material in containers when possible. Where this is not possible resort must be had in general to specially designed equipment. Analysis of the handling problem will readily show if this is

FIG. 31.—Typical trailer for use with tractors.

necessary, and recourse should then be had to the engineering department of a manufacturing concern making equipment of the type needed.

Special Requirements of the Chemical Engineering Field

Generally speaking, the handling problems encountered in the chemical engineering industries are no different from those encountered elsewhere. Consequently, the general line of materials-handling equipment as manufactured for, and used by, industry, in general, can be applied in this field.

Five conditions, however, that are found in some handling operations in this field, influence the choice of handling equipment. These are:

1. Corrosion hazard from chemicals or excess moisture.
2. Heat damage hazard from hot materials.
3. Fire or explosion hazards.
4. Poison hazards.
5. Sanitary requirements.

1. Corrosion from chemicals or excess moisture is the most frequently encountered special condition. It is a hazard most often met where materials are handled loose. Such corrosion makes the use of conveyors something to be carefully considered, at least when they are constructed of iron or steel and are unprotected from attack. Under such conditions it is often better to use truck equipment for all handling, for in most cases it is simpler to protect the trucks from damage.

In many cases, however, the corrosion will not seriously shorten the life or affect the usefulness of the conveyor if a certain amount of special construction is undertaken to protect the points where the most serious damage would result. An example of what is meant is the use of gravity roller conveyors in a plant making bleaching powder. The only serious effect of the attack was experienced in the bearings of the rollers. By using a special type of bearing, filled with a lubricant under high pressure, the penetration of any corrosive into the bearings was avoided and satisfactory service obtained from the conveyor.

Another example of special construction to avoid the effects of corrosion is that in which freshly quenched material is handled on an apron conveyor equipped with cast-iron aprons which withstand the effects of the moisture better than steel plate.

In general, the choice of a means of handling material where a corrosion hazard exists means that the economic choice lies among three possible courses: (*a*) The cheapest type of equipment may be selected, regardless of how seriously it is attacked. This will then be replaced as often as necessary, incurring high replacement charges to offset low first cost. (*b*) The usual type of handling equipment for the particular class of handling may be constructed in such a way or of such materials as to avoid high replacement costs, while increasing first costs. (*c*) The equipment may be protected fully from attack by some such means as the use of containers that adequately seal in the material, or, if the attack is due to a gas or fume, by the use of protecting auxiliary structures and a ventilating system.

This discussion does not fully cover the manifold possibilities of corrosion attack to which materials-handling equipment is exposed in the chemical engineering industries. Each handling problem should be analyzed to ascertain if any corrosion hazard is present, and, if one is discovered, its effect on the choice of equipment should be considered along the lines that have been indicated.

2. Heat damage hazard from hot materials is usually encountered in the handling of furnace or oven products such as coke, clinker, etc. The usual expedient for avoiding such damage is to armor the parts coming into contact with the heat by the use of cast-iron plates, liners, containers, or other parts. Where the heat is too high for safe use of cast iron, a special material should be obtained suitable to the condition.

For handling molten materials, such practice is to use buckets and spouts (chutes) lined with refractory for high temperatures, or of cast iron where the temperatures are moderate.

3. Fire and explosion hazards are encountered in the handling of materials in certain of the chemical engineering industries. In cases of fire hazard alone, it is usually sufficient to provide for the protection of the material handled from exposure to flame or sparks. As electric equipment or static is the usual source of such hazards, the procedure usually followed is to select equipment which can be operated by motors placed in a sealed space or in an adjacent room and to provide ground connections that will prevent the accumulation of static charges.

Another type of fire and explosion hazard encountered occurs where flammable dusts are produced. To avoid trouble in such cases, four courses are open: (1) Spaces in which handling is carried out may be ventilated to reduce dust concentrations below the danger point; (2) these spaces may be thoroughly screened from any possible spark or flame; (3) the material may be handled in an atmosphere too low in oxygen to support combustion; (4) the material may be handled in containers that eliminate the scattering of dust.

Handling of explosives such as gunpowders, dynamite, etc., is a hazard restricted to a very few industries. These have made a study of the problem for years and have developed methods that avoid the exposure of the material to jars, jolts, and other shocks. In a problem as special as this type of handling, nothing should be carried out except under the guidance of an expert of long experience in the industry.

4. Poison hazards encountered in materials-handling work are hazards to the operator, not to the equipment. In selecting a handling method where a poison hazard exists, therefore, the course to follow is to select equipment that can be operated from a distance or automatically; or so to enclose the material in containers, pipe lines, or by other means that the hazard is avoided.

5. Sanitary requirements frequently have an effect on the selection of materials-handling equipment. In handling food products, or materials such as fats, oils, animal or vegetable matter, there is a possibility of the accumulation of material in or around the equipment that will ferment or spoil, causing a condition that may damage material in process or form a health hazard.

An example of such conditions is found in the dairy industry. Should milk be slopped on the materials-handling equipment and allowed to stand in crevices of the equipment, a condition is soon developed that is detrimental to the product and disagreeable to the workers. Therefore, all equipment is thoroughly washed at frequent intervals. This necessitates the use of moisture-proof bearings. This particular feature has become so important with gravity roller conveyors that a standard line of moisture-proof designs is now available on the market.

Another example of the effect of sanitary requirements is found in food-packing plants, where overhead equipment or trolley conveyors are preferred, in order that the floor may be clear and thoroughly accessible for cleaning.

Table 1. Large Quantity Handled in Continuous Flow

Over 100 tons per hour

Material handled	Horizontal movement	Vertical upward movement	Vertical downward movement	Combined horizontal and vertical movement
Bulk material of lumpy, broken or crushed nature	Belt conveyors* Apron or pan conveyors Pneumatic conveyors Solids pumps	Bucket elevators* V-bucket carriers Pneumatic conveyors	Chutes* V-bucket carriers	Belt conveyors* Chutes Pneumatic conveyors Solids pumps Pivoted bucket carriers V-bucket carriers
Bulk material of finely divided or pulverized nature	Pneumatic conveyors* Belt conveyors Solids pumps	Pneumatic conveyors* Bucket elevators V-bucket carriers	Chutes* Pneumatic conveyors	Pneumatic conveyors* Solids pumps Chutes
Containers such as bags or sacks	Gravity roller conveyors* Platform conveyors Slat conveyors Belt conveyors	Chain elevators* Belt elevators Tray elevators	Chutes* Spiral chutes Roller spirals	Gravity roller conveyors* Live roller conveyors Belt conveyors Platform conveyors Slat conveyors Chutes
Containers such as barrels or drums	Slat conveyors* Chain conveyors Platform conveyors Gravity roller conveyors	Barrel elevators* Chain elevators	Barrel elevators* Chain elevators Roller spirals Chutes	Slat conveyors* Chain conveyors Platform conveyors Chutes
Containers such as boxes or packages	Chain conveyors* Gravity roller conveyors* Wheel conveyors Belt conveyors Platform conveyors Slat conveyors Push-bar conveyors	Package elevators* Chain elevators Belt elevators	Spiral chutes* Package elevators Chutes Roller spirals	Chain conveyors* Gravity roller conveyors* Belt conveyors Push-bar conveyors Chutes Live roller conveyors
Pieces such as logs, tires, pulp laps, large lumps	Chain-trolley conveyors* Chain or cable conveyors Apron or pan conveyors Gravity roller conveyors Belt conveyors	Chain elevators* Belt elevators Tray elevators	Spiral chutes* Chutes	Chain trolley conveyors* Chain or cable conveyors Apron or pan conveyors Gravity roller conveyors Chutes

* Usual choice.

Choice of Materials-handling Equipment

To classify the solutions of handling problems, Tables 1 to 12 are presented, showing three general divisions into which all handling problems fall: large quantity; medium quantity; small quantity. With these factors, such other factors as type of flow, direction of flow, and condition of material are considered. The tables present only such choice as might be made to satisfy ordinary conditions, wherein no special difficulties are encountered. Note that the length of the movement which is to be made has an effect on selection and that it is frequently best practice (if long distances are to be covered) to use truck equipment or industrial railways.

Following Tables 1 to 12 are other tables giving data as to capacity, horsepower, and first cost of frequently used types of materials-handling equipment. The type most usually selected for a given handling job is indicated in Tables 1 to 12 by an asterisk (*).

Table 2. Large Quantity Handled in Intermittent Flow

Over 100 tons per hour

Material handled	Horizontal movement	Vertical upward movement	Vertical downward movement	Combined horizontal and vertical movement
Bulk material of lumpy, broken or crushed nature	Drag-line scrapers* Industrial railways* Tractors and trailers Traveling cranes Monorail trolleys (motor driven)	Skip hoists* Platform elevators	Chutes*	Traveling cranes* (power) Monorail trolleys (motor driven) Locomotive cranes
Bulk material of finely divided or pulverized nature	Industrial railways* Tractors and trailers* Monorail trolleys (motor driven)	Skip hoists* Platform elevators	Chutes*	Traveling cranes* (power) Monorail trolleys (motor driven)
Containers such as bags or sacks	Tractors and trailers* Industrial railways Traveling cranes Monorail trolleys (motor driven)	Drum hoists* Platform elevators Stationary cranes	Chutes* Platform elevators Drum hoists Stationary cranes	Traveling cranes* (power) Monorail trolleys (motor driven)
Containers such as barrels or drums	Tractors and trailers* Industrial railways	Platform elevators* Stationary cranes Drum hoists	Platform elevators* Drum hoists Stationary cranes	Monorail trolleys* (motor driven) Traveling cranes (power)
Containers such as boxes or packages	Tractors and trailers* Industrial railways Traveling cranes Monorail trolleys (motor driven)	Platform elevators* Stationary cranes Drum hoists	Chutes* Platform elevators Drum hoists Stationary cranes	Monorail trolleys* (motor driven) Traveling cranes (power)
Pieces such as logs, tires, pulp laps, large lumps	Monorail trolleys* (motor driven) Tractors and trailers* Industrial railways Traveling cranes	Platform elevators* Stationary cranes Drum hoists	Chutes* Platform elevators	Monorail trolley* (motor driven) Traveling cranes (power) Locomotive cranes

* Usual choice.

Equipment Data and Costs

Cost data given in the tables and text of this section were first assembled and checked in 1934. From that time until the present revision (1940) price changes have been small and not enough to require revision when used only for preliminary estimates. Naturally, before any construction work is contracted for or carried out, the engineer will procure a firm figure of cost from a conveyor manufacturer, to serve as basis for final cost estimates and actual appropriations.

Belt Conveyors (see pp. 2197 and 2202). Table 13 gives general information that will assist in the selection of belt conveyors for preliminary estimating purposes. The table does not fully cover the subject, but suggests the ranges of belt sizes, speeds, capacities, and size of material handled.

Table 3. Large Quantity Handled; Intermittent Demand with Storage to Give Continuous Flow Effect

Over 100 tons per hour

Material handled	Horizontal movement	Vertical upward movement	Vertical downward movement	Combined horizontal and vertical movement
Bulk material of lumpy, broken or crushed nature	Belt conveyors* Pneumatic conveyors Solids pumps Apron or pan conveyors Scraper or drag conveyors	Bucket elevators* V-bucket carriers Pneumatic conveyors	Chutes* V-bucket carriers	Belt conveyors* Pneumatic conveyors Solids pumps Chutes V-bucket carriers Pivoted bucket carriers
Bulk material of finely divided or pulverized nature	Pneumatic conveyors* Belt conveyors Flight conveyors Solids pumps	Pneumatic conveyors* Bucket elevators V-bucket carriers	Chutes* V-bucket carriers	Pneumatic conveyors* Solids pumps Chutes
Containers such as bags or sacks	Gravity roller conveyors* Belt conveyors Platform conveyors Slat conveyors	Chain elevators* Belt elevators Tray elevators	Chutes* Spiral chutes Roller spirals	Gravity roller conveyors* Live roller conveyors Belt conveyors Platform conveyors Slat conveyors Chutes
Containers such as barrels or drums	Slat conveyors* Chain conveyors Platform conveyors Gravity roller conveyors	Barrel elevators* Chain elevators	Barrel elevators* Chain elevators Roller spirals Chutes	Slat conveyors* Chain conveyors Platform conveyors Chutes
Containers such as boxes or packages	Chain conveyors* Gravity roller conveyors* Belt conveyors Platform conveyors Slat conveyors Push-bar conveyors Wheel conveyors	Package elevators* Chain elevators Belt elevators	Spiral chutes* Chutes Roller spirals Package elevators	Chain conveyors* Gravity roller conveyors* Belt conveyors Push-bar conveyors Chutes
Pieces such as logs, tires, pulp laps, large lumps	Chain trolleys* Chain or cable conveyors* Apron or pan conveyors Gravity roller conveyors	Chain elevators* Belt elevators Tray elevators	Spiral chutes* Chutes	Chain trolleys* Chain or cable conveyors* Apron or pan conveyors Gravity roller conveyors Chutes

* Usual choice.

Capacities given assume uniform loading with feeder. Extra width should be provided for peak loads. Speeds should be governed by size of material, method of loading, incline of conveyor, if any, and other affecting factors.

Horsepower required to drive the belt, taking into consideration the weight of the belt, is figured from the following formula (Link-Belt Company):

$$\text{Horsepower} = \frac{KL}{1{,}000}\left(T + \frac{SB}{16}\right) + \frac{TH}{990} + P$$

where K is a power constant from Table 14; L is length of conveyor (between pulley centers) in feet; T is tons per hour of material; S is speed of belt in feet per minute; B is weight of belt in pounds per foot, from Table 15; H is rise of belt in feet (if belt is inclined); P is the horsepower for tripper (if used) from Table 14.

Table 4. Large Quantity Handled; Continuous Demand with Storage to Give Intermittent Flow Effect

Over 100 tons per hour

Material handled	Horizontal movement	Vertical upward movement	Vertical downward movement	Combined horizontal and vertical movement
Bulk material of lumpy, broken or crushed nature	Drag-line scrapers* Industrial railways* Tractors and trailers Traveling cranes Monorail trolleys (motor driven)	Skip hoists* Platform elevators	Chutes*	Traveling cranes* (power) Monorail trolleys (motor driven) Locomotive cranes
Bulk material of finely divided or pulverized nature	Tractors and trailers* Industrial railways* Monorail trolleys (motor driven)	Skip hoists* Platform elevators	Chutes*	Traveling cranes* (power) Monorail trolleys (motor driven)
Containers such as bags or sacks	Tractors and trailers* Industrial railways Traveling cranes Monorail trolleys (motor driven)	Platform elevators* Stationary cranes Drum hoists	Chutes* Platform elevators Drum hoists Stationary cranes	Traveling cranes* (power) Monorail trolleys (motor driven)
Containers such as barrels or drums	Tractors and trailers* Industrial railways	Platform elevators* Stationary cranes Drum hoists	Platform elevators* Drum hoists Stationary cranes	Monorail trolleys* (motor driven) Traveling cranes (power)
Containers such as boxes or packages	Tractors and trailers* Industrial railways Traveling cranes (power) Monorail trolleys (motor driven)	Platform elevators* Stationary cranes Drum hoists	Chutes* Platform elevators Drum hoists Stationary cranes	Monorail trolleys* (motor driven) Traveling cranes (power)
Pieces such as logs, tires, pulp laps, large lumps	Monorail trolleys* (motor driven) Tractors and trailers* Industrial railways Traveling cranes	Platform elevators* Stationary cranes Drum hoists	Chutes* Platform elevators	Monorail trolleys* (motor driven) Traveling cranes (power) Locomotive cranes

* Usual choice.

To the preceding calculation add:

1. For horsepower consumed by the terminals (head and foot shafts only), 20 per cent for conveyors under 50-ft. centers; 10 per cent for conveyors under 100-ft. centers; 5 per cent for conveyors under 150-ft. centers.

2. For horsepower consumed by countershaft drives, 10 per cent of total for each speed reduction through cast tooth gears, or 5 per cent of total for each speed reduction through cut gears.

Table 5. Medium Quantity Handled in Continuous Flow

10 to 100 tons per hour

Material handled	Horizontal movement	Vertical upward movement	Vertical downward movement	Combined horizontal and vertical movement
Bulk materials of lumpy, broken or crushed nature	Belt conveyors* Apron or pan conveyors Drag chain conveyors Redler conveyors Johns conveyors Vibrating conveyors	Bucket elevators*	Chutes*	Belt conveyors* Chutes Flight conveyors Apron or pan conveyors V-bucket carriers Pivoted bucket carriers Redler conveyors Johns conveyors Vibrating conveyors
Bulk material of finely divided or pulverized nature	Solids pumps* Pneumatic conveyors* Belt conveyors Scraper conveyors Flight conveyors Screw conveyors Redler conveyors Johns conveyors Vibrating conveyors	Pneumatic conveyors* Bucket elevators	Chutes* Pneumatic conveyors	Solids pumps* Pneumatic conveyors* Chutes Screw conveyors Flight conveyors Redler conveyors Johns conveyors Vibrating conveyors
Containers such as bags or sacks	Belt conveyors* Gravity roller conveyors* Chain trolleys	Chain elevators* Belt elevators Tray elevators	Chutes* Spiral chutes Roller spirals	Belt conveyors* Gravity roller conveyors* Live roller conveyors Chutes
Containers such as barrels or drums	Chain conveyors* Gravity roller conveyors	Barrel elevators* Chain elevators	Barrel elevators* Chain elevators Roller spirals Chutes	Chain conveyors* Chutes
Containers such as boxes or packages	Chain conveyors* Gravity roller conveyors* Chain trolleys Belt conveyors Wheel conveyors	Package elevators* Chain elevators Belt elevators	Spiral chutes* Package elevators Chutes Roller spirals	Gravity roller conveyors* Chain conveyors* Belt conveyors Chain trolleys Push-bar conveyors Chutes
Pieces such as logs, tires, pulp laps, large lumps	Chain trolleys* Chain or cable conveyors* Belt conveyors Gravity roller conveyors	Chain elevators* Belt elevators Tray elevators	Spiral chutes* Chutes	Chain trolleys* Chain or cable conveyors* Pan conveyors Gravity roller conveyors Chutes

* Usual choice.

List prices and weights for standard, three-roll, antifriction troughing idlers in unit stands, with return for conveyor belt, are given in Table 16. There are idlers available at higher and lower prices than given in the table, depending on design and purpose. Those given will serve as a guide in estimating, when used with Table 17, giving spacing of idlers.

Table 6. Medium Quantity Handled in Intermittent Flow

10 to 100 tons per hour

Material handled	Horizontal movement	Vertical upward movement	Vertical downward movement	Combined horizontal and vertical movement
Bulk material of lumpy, broken or crushed nature	Monorail trolleys* (motor driven) Tractors and trailers Power trucks with skids	Skip hoists* Platform elevators Drum hoists	Platform elevators* Drum hoists	Monorail trolleys* (motor driven) Traveling cranes (power)
Bulk material of finely divided or pulverized nature	Power trucks with skids* Tractors and trailers Monorail trolleys (motor driven)	Platform elevators* Skip hoists Drum hoists	Platform elevators* Drum hoists	Monorail trolleys* (motor driven) Traveling cranes (power)
Containers such as bags or sacks	Power trucks with skids* Tractors and trailers* Platform trucks (power)	Drum hoists* Platform elevators Stationary cranes Chain hoists (power)	Drum hoists* Chain hoists (power) Platform elevators Stationary cranes	Monorail trolleys* (motor driven) Traveling cranes (power)
Containers such as barrels or drums	Power trucks with skids* Tractors and trailers* Platform trucks (power)	Platform elevators* Drum hoists Stationary cranes Chain hoists (power)	Platform elevators* Drum hoists Stationary cranes Chain hoists (power)	Monorail trolleys* (motor driven) Traveling cranes (power)
Containers such as boxes or packages	Power trucks with skids* Tractors and trailers* Platform trucks (power)	Platform elevators* Drum hoists Stationary cranes Chain hoists (power)	Platform elevators* Drum hoists Stationary cranes Chain hoists (power)	Monorail trolleys* (motor driven) Traveling cranes (power)
Pieces such as logs, tires, pulp laps, large lumps	Power trucks with skids* Tractors and trailers* Platform trucks (power) Monorail trolleys (motor driven)	Chain hoists* (power) Platform elevators Drum hoists Stationary cranes	Chain hoists* (power) Platform elevators Drum hoists Stationary cranes	Monorail trolleys* (motor driven) Traveling cranes (power)

* Usual choice.

In estimating belt conveyor costs, allowance should also be made for head and foot pulleys, take-up, and drive. Where trippers, unloaders, and feeders are used, these also should be included.

Bucket Elevators. In Table 18 are given the speed, horsepower, and capacity of centrifugal-discharge bucket elevators with malleable-iron buckets

Table 7. Medium Quantity Handled; Intermittent Demand with Storage to Give Continuous Flow Effect

10 to 100 tons per hour

Material handled	Horizontal movement	Vertical upward movement	Vertical downward movement	Combined horizontal and vertical movement
Bulk material of lumpy, broken or crushed nature	Belt conveyors* Apron or pan conveyors Drag chain conveyors Redler conveyors Johns conveyors Vibrating conveyors	Bucket elevators*	Chutes*	Belt Conveyors* Chutes Flight conveyors Apron or pan conveyors V-bucket carriers Pivoted bucket carriers Redler conveyors Johns conveyors Vibrating conveyors
Bulk material of finely divided or pulverized nature	Pneumatic conveyors* Solids pumps* Belt conveyors Scraper conveyors Flight conveyors Screw conveyors Redler conveyors Johns conveyors Vibrating conveyors	Pneumatic conveyors* Bucket elevators	Chutes* Pneumatic conveyors	Solids pumps* Pneumatic conveyors* Chutes Screw conveyors Flight conveyors Redler conveyors Johns conveyors Vibrating conveyors
Containers such as bags or sacks	Belt conveyors* Gravity roller conveyors* Chain trolleys	Chain elevators* Belt elevators Tray elevators	Chutes* Spiral chutes Roller spirals	Belt conveyors* Gravity roller conveyors* Live roller conveyors Chutes
Containers such as barrels or drums	Chain conveyors* Gravity roller conveyors	Barrel elevators* Chain elevators	Barrel elevators* Chain elevators Roller spirals Chutes	Chain conveyors* Chutes
Containers such as boxes or packages	Chain conveyors* Gravity roller conveyors* Belt conveyors Chain trolleys Wheel conveyors	Package elevators* Belt elevators Chain elevators	Spiral chutes* Package elevators Roller spirals Chutes	Chain conveyors* Gravity roller conveyors* Belt conveyors Push-bar conveyors Chutes Chain trolleys
Pieces such as logs, tires, pulp laps, large lumps	Chain trolleys* Chain or cable conveyors* Belt conveyors Gravity roller conveyors	Chain elevators* Belt elevators Tray elevators	Spiral chutes* Chutes	Chain trolleys* Chain or cable conveyors* Pan conveyors Gravity roller conveyors Chutes

* Usual choice.

spaced on single-strand chain and enclosed in steel casings. These are a commonly used type of elevator.

Horsepower of bucket elevators may be figured from the equations

$$\text{Horsepower} = \frac{HT}{500}$$

for spaced buckets and digging boots, and

$$\text{Horsepower} = \frac{HT}{550}$$

for continuous buckets with loading leg. Here H is the center-to-center distance in feet and T is in tons per hour.

Cast-iron elevator boots vary in list price from $60 to $180 for the range of sizes given in Table 18, while steel boots vary from $110 to $230. Buckets of standard weight range from $0.35 to $1.35 for these sizes.

Table 8. Medium Quantity Handled; Continuous Demand with Storage to Give Intermittent Flow Effect

10 to 100 tons per hour

Material handled	Horizontal movement	Vertical upward movement	Vertical downward movement	Combined horizontal and vertical movement
Bulk material of lumpy, broken, or crushed nature	Monorail trolleys* (power) Tractors and trailers Power trucks with skids	Skip hoists* Platform elevators Drum hoists	Platform elevators* Drum hoists	Monorail trolleys* (power) Traveling cranes (power)
Bulk material of finely divided or pulverized nature	Power trucks with skids* Tractors and trailers Monorail trolleys (power)	Platform elevators* Skip hoists Drum hoists	Platform elevators* Drum hoists	Monorail trolleys* (power) Traveling cranes (power)
Containers such as bags or sacks	Power trucks with skids* Tractors and trailers* Platform trucks (power)	Drum hoists* Platform elevators Stationary cranes Chain hoists (power)	Drum hoists* Platform elevators Stationary cranes Chain hoists (power)	Monorail trolleys* (power) Traveling cranes (power)
Containers such as barrels or drums	Power trucks with skids* Tractors and trailers* Platform trucks (power)	Platform elevators* Drum hoists Stationary cranes Chain hoists (power)	Platform elevators* Drum hoists Stationary cranes Chain hoists (power)	Monorail trolleys* (power) Traveling cranes (power)
Containers such as boxes or packages	Power trucks with skids* Tractors and trailers* Platform trucks (power)	Platform elevators* Drum hoists Stationary cranes Chain hoists (power)	Platform elevators* Drum hoists Stationary cranes Chain hoists (power)	Monorail trolleys* (power) Traveling cranes (power)
Pieces such as logs, tires, pulp laps, large lumps	Power trucks with skids* Tractors and trailers* Platform trucks (power) Monorail trolleys (power)	Chain hoists* (power) Platform elevators Drum hoists Stationary cranes	Chain hoists* (power) Platform elevators Drum hoists Stationary cranes	Monorail trolleys* (power) Traveling cranes (power)

* Usual choice.

An elevator with 6- by 4-in. buckets, 30 ft. from center to center, complete with boot and head, including drive gears and pulleys, will list at $350, with $4 added for each extra foot in length when no casing is included and buckets are mounted on rubber belting in place of chain. The same type of elevator, but with 16- by 8-in. buckets, will list at $1,100. These figures will indicate the range in first cost for such equipment.

Cost of driving equipment can be estimated from horsepower requirements.

Table 9. Small Quantity Handled in Continuous Flow
Up to 10 tons per hour

Material handled	Horizontal movement	Vertical upward movement	Vertical downward movement	Combined horizontal and vertical movement
Bulk material of lumpy, broken or crushed nature	Belt conveyors* Drag chain conveyors Scraper conveyors Redler conveyors Johns conveyors Vibrating conveyors	Bucket elevators*	Chutes*	Belt conveyors* Chutes Redler conveyors Johns conveyors Vibrating conveyors
Bulk material of finely divided or pulverized nature	Belt conveyors* Pneumatic conveyors Solids pumps Redler conveyors Johns conveyors Vibrating conveyors	Bucket elevators* Pneumatic conveyors	Chutes* Pneumatic conveyors	Belt conveyors* Chutes Pneumatic conveyors Solids pumps Redler conveyors Johns conveyors Vibrating conveyors
Containers such as bags or sacks	Belt conveyors* Gravity roller conveyors*	Chain elevators* Belt elevators	Chutes* Spiral chutes	Belt conveyors* Gravity roller conveyors* Chutes
Containers such as barrels or drums	Chain conveyors* Gravity roller conveyors	Chain elevators*	Chutes*	Chain conveyors* Chutes Gravity roller conveyors
Containers such as boxes or packages	Gravity roller conveyors* Belt conveyors Chain trolleys Wheel conveyors	Chain elevators* Belt elevators	Spiral chutes* Chutes	Gravity roller conveyors* Belt conveyors Chain conveyors Chain trolleys Chutes
Pieces such as logs, tires, pulp laps, large lumps	Chain or cable conveyors* Chain trolleys* Gravity roller conveyors Belt conveyors	Chain elevators* Belt elevators	Spiral chutes* Chutes	Chain or cable conveyors* Chain trolleys* Belt conveyors Chutes Gravity roller conveyors

* Usual choice.

Chain Conveyors. Whether of the type that runs under the load, such as log conveyors, or of the type that runs in guides with attachments to carry the load, chain conveyors consist of an appropriate type of chain, with sprockets, attachments, guides, and supports. Table 19 gives data for a style of chain used for roller-chain conveyors where the load is supported on attachments. These attachments vary in price from $3 to $6 per ft., assembled in the chain on every link.

Data for chain used for directly supporting the load do not differ much from the above, except that such chain usually has a higher working strength.

Table 10. Small Quantity Handled in Intermittent Flow

Up to 10 tons per hour

Material handled	Horizontal movement	Vertical upward movement	Vertical downward movement	Combined horizontal and vertical movement
Bulk material of lumpy, broken or crushed nature	Hand trucks with skids* Power trucks with skids* Platform trucks (hand or power) Monorail trolleys (hand)	Platform elevators* (portable) Drum hoists	Platform elevators* (portable) Drum hoists	Monorail trolleys* (hand) Traveling cranes (hand)
Bulk material of finely divided or pulverized nature	Power trucks with skids* Hand trucks with skids* Platform trucks (hand or power) Monorail trolleys (hand)	Platform elevators* (portable) Drum hoists	Platform elevators* (portable) Drum hoists	Monorail trolleys* (hand) Traveling cranes (hand)
Containers such as bags or sacks	Power trucks with skids* Hand trucks with skids* Platform trucks (hand or power) Monorail trolleys (hand)	Chain hoists* (power or hand) Drum hoists Platform elevators (portable)	Chain hoists* (power or hand) Drum hoists Platform elevators (portable)	Monorail trolleys* (hand) Traveling cranes (hand)
Containers such as barrels or drums	Power trucks with skids* Hand trucks with skids* Platform trucks (hand or power) Monorail trolleys (hand)	Chain hoists* (power or hand) Drum hoists Platform elevators (portable)	Chain hoists* (power or hand) Drum hoists Platform elevators (portable)	Monorail trolleys* (hand) Traveling cranes (hand)
Containers such as boxes or packages	Power trucks with skids* Hand trucks with skids* Platform trucks (hand or power) Monorail trolleys (hand)	Platform elevators* (portable) Drum hoists Chain hoists (power or hand)	Platform elevators* (portable) Drum hoists Chain hoists (power or hand)	Monorail trolleys* (hand) Traveling cranes (hand)
Pieces such as logs, tires, pulp laps, large lumps	Power trucks with skids* Hand trucks with skids* Platform trucks (hand or power) Monorail trolleys (hand)	Chain hoists* (power or hand) Drum hoists Platform elevators (portable)	Chain hoists* (power or hand) Drum hoists Platform elevators (portable)	Monorail trolleys* (hand) Traveling cranes (hand)

* Usual choice.

List prices of cast-iron sprocket wheels for these chains vary from about $7 for a 6-in. pitch diameter to about $85 for a 58-in. pitch diameter. Drive and supports must also be included in any estimate.

Chain Trolley Conveyors. Chain trolley data are similar to those for chain conveyors. The lighter, less expensive chains used are balanced in cost by the greater expense of attachments, including the trolley hangers and rollers. Table 20 gives size and loading data on these trolleys.

Table 11. Small Quantity Handled; Intermittent Demand with Storage to Give Continuous Flow Effect

Up to 10 tons per hour

Material handled	Horizontal movement	Vertical upward movement	Vertical downward movement	Combined horizontal and vertical movement
Bulk material of lumpy, broken or crushed nature	Belt conveyors* Drag chain conveyors Scraper conveyors Redler conveyors Johns conveyors Vibrating conveyors	Bucket elevators*	Chutes*	Belt conveyors* Chutes Redler conveyors Johns conveyors Vibrating conveyors
Bulk material of finely divided or pulverized nature	Belt conveyors* Pneumatic conveyors Solids pumps Redler conveyors Johns conveyors Vibrating conveyors	Bucket elevators* Pneumatic conveyors	Chutes* Pneumatic conveyors	Belt conveyors* Chutes Pneumatic conveyors Solids pumps Redler conveyors Johns conveyors Vibrating conveyors
Containers such as bags or sacks	Belt conveyors* Gravity roller conveyors*	Chain elevators* Belt elevators	Chutes* Spiral chutes	Belt conveyors* Gravity roller conveyors* Chutes
Containers such as barrels or drums	Chain conveyors* Gravity roller conveyors	Chain conveyors*	Chutes*	Chain conveyors* Chutes Gravity roller conveyors
Containers such as boxes or packages	Gravity roller conveyors* Belt conveyors Chain trolleys Wheel conveyors	Chain elevators* Belt elevators	Spiral chutes* Chutes	Gravity roller conveyors* Chain trolleys Belt conveyors Chain conveyors Chutes
Pieces such as logs, tires, pulp laps, large lumps	Chain or cable conveyors* Chain trolleys* Belt conveyors Gravity roller conveyors	Chain elevators* Belt elevators	Spiral chutes* Chutes	Chain or cable conveyors* Chain trolleys* Belt conveyors Chutes Gravity roller conveyors

* Usual choice.

Apron, Pan, Slat, and Platform Conveyors. List prices of these conveyors are determined as are those of any other chain conveyor, as these conveyors are single- or double-strand chain conveyors with the pans, aprons, slats, or platforms serving as attachments.

The horsepower required to drive such a conveyor is given approximately by the following formula, which does not take into account losses due to friction and other factors that result in a higher horsepower requirement.

Table 12. Small Quantity Handled; Continuous Demand with Storage to Give Intermittent Flow Effect

Up to 10 tons per hour

Material handled	Horizontal movement	Vertical upward movement	Vertical downward movement	Combined horizontal and vertical movement
Bulk material of lumpy, broken, or crushed nature	Power trucks with skids* Hand trucks with skids* Platform trucks (hand or power) Monorail trolleys (hand)	Platform elevators* (portable) Drum hoists	Platform elevators* (portable) Drum hoists	Monorail trolleys* (hand) Traveling cranes (hand)
Bulk material of finely divided or pulverized nature	Power trucks with skids* Hand trucks with skids* Platform trucks (hand or power) Monorail trolleys (hand)	Platform elevators* (portable) Drum hoist	Platform elevators* (portable) Drum hoists	Monorail trolleys* (hand) Traveling cranes (hand)
Containers such as bags or sacks	Power trucks with skids* Hand trucks with skids* Platform trucks (hand or power) Monorail trolleys (hand)	Chain hoists* (power or hand) Drum hoists Platform elevators (portable)	Chain hoists* (power or hand) Drum hoists Platform elevators (portable)	Monorail trolleys* (hand) Traveling cranes (hand)
Containers such as barrels or drums	Power trucks with skids* Hand trucks with skids* Platform trucks (hand or power) Monorail trolleys (hand)	Chain hoists* (power or hand) Drum hoists Platform elevators (portable)	Chain hoists* (power or hand) Drum hoists Platform elevators (portable)	Monorail trolleys* (hand) Traveling cranes (hand)
Containers such as boxes or packages	Power trucks with skids* Hand trucks with skids* Platform trucks (hand or power) Monorail trolleys (hand)	Platform elevators* (portable) Drum hoists Chain hoists (power or hand)	Platform elevators* (portable) Drum hoists Chain hoists (power or hand)	Monorail trolleys* (hand) Traveling cranes (hand)
Pieces such as logs, tires, pulp laps, large lumps	Power trucks with skids* Hand trucks with skids* Platform trucks (hand or power) Monorail trolleys (hand)	Chain hoists* (power or hand) Drum hoists Platform elevators (portable)	Chain hoists* (power or hand) Drum hoists Platform elevators (portable)	Monorail trolleys* (hand) Traveling cranes (hand)

* Usual choice.

Note, in figuring loads, that the formula is for horsepower of the head shaft, so that weight of conveyor as well as of material should be taken. Also note that inclined conveyors of this type will require more horsepower for a given load than horizontal conveyors.

$$\text{Horsepower} = \frac{\text{load in pounds} \times \text{speed in feet per minute}}{33{,}000}$$

Capacities for various widths of apron or pan conveyors are given in Table 21. This table is based on the assumption that the conveyor is uniformly loaded to 75 per cent of the maximum cross section (net width of carrying

Table 13. Belt Conveyors; Capacities, Speeds, and Sizes of Pieces
Link-Belt Company

Width of belt, in.	Belt ply		Cross section of load, sq. ft.	Cu. ft. per hr. at 100 ft. per min.	Capacities in tons per hr. at 100 ft. per min., at weight per cu. ft. of material of				Maximum advisable belt speed, ft. per min.	Medium size of material, in.	
	Min.	Max.			50 lb.	75 lb.	100 lb.	150 lb.		80% under	20% not over
12	3	4	0.084	504	12.6	18.9	25.2	37.8	300	1½	2
14	3	5	0.114	686	17.1	25.6	34.3	51.3	300	2	3
16	3	5	0.149	896	22.4	33.6	44.8	67.2	300	2½	4
18	4	6	0.189	1,134	28.3	42.5	56.7	84.9	350	3	5
20	4	6	0.233	1,400	35.0	52.5	70.0	105.0	350	3½	6
24	4	6	0.336	2,016	50.4	75.6	100.8	151.2	400	4½	8
30	5	7	0.525	3,150	78.7	118.1	157.5	236.2	500	7	12
36	6	8	0.751	4,536	113.4	170.1	226.8	340.2	600	8	16
42	6	9	1.029	6,174	154.3	231.5	308.7	463.0	600	10	20
48	6	10	1.333	8,064	201.6	302.4	402.2	604.8	600	12	24
54	8	11	1.701	10,206	255.1	382.6	510.2	765.3	600	13	26
60	8	12	2.100	12,600	315.0	472.5	630.0	945.0	600	14	28

Table 14. Tripper Horsepower and Constants for Belt-conveyor Horsepower
Link-Belt Company

Width of belt, in.	Value of K		H.p. for tripper P	
	Plain brgs.	Anti-frict. brgs.	Plain brgs.	Anti-frict. brgs.
12	0.149	0.070	0.75	0.50
14	0.144	0.070	1.00	0.75
16	0.140	0.070	1.00	0.75
18	0.135	0.069	1.50	1.25
20	0.132	0.066	1.50	1.25
24	0.125	0.062	1.75	1.25
30	0.117	0.057	2.50	1.75
36	0.109	0.054	3.00	2.50
42	0.102	0.051	4.00	3.00
48	0.097	0.048	5.00	3.25
54	0.092	0.047	6.00	5.00
60	0.090	0.045	7.00	6.00

surface, in inches, multiplied by height of sides, in inches), and traveling at a speed of 10 ft. per min. For other widths of carrying surface, heights of sides, and speeds, than those listed, the capacities should be figured in direct proportion to the varying dimensions; for instance, if the speed is 50 ft. per min., multiply the listed capacity by 5.

Table 15. Approximate Weights of 28-oz. Duck Rubber Conveyor Belt

Link-Belt Company
Pounds per foot

Ply of belt	Thickness of top cover, In.	Width of belt, In.							
		12	14	16	18	20	24	30	60
3	1/32	1.27	1.48	1.69					
	1/16	1.51	1.76	2.01					
	1/8	1.99	2.32	2.65					
	3/16	2.46	2.88	3.28					
4	1/32	1.55	1.78	2.05	2.33	2.56	3.08		
	1/16	1.78	2.08	2.37	2.67	2.96	3.56		
	1/8	2.27	2.64	3.01	3.39	3.77	4.53		
	3/16	2.75	3.19	3.64	4.09	4.54	5.46		
5	1/32	....	....	2.41	2.71	2.99	3.61	4.49	
	1/16	....	....	2.71	3.05	3.41	4.07	5.09	
	1/8	....	....	3.36	3.78	4.20	5.04	6.30	
	3/16	....	....	4.00	4.50	5.00	6.00	7.50	

		Width of belt, In.							
		20	24	30	36	42	48	54	60
6	1/32	3.44	4.13	5.06	6.19	7.22	8.42		
	1/16	3.84	4.61	5.76	6.92	8.06	9.4		
	1/8	4.64	5.57	6.96	8.35	9.75	11.4		
	3/16	5.44	6.54	8.16	9.80	11.42	13.35		
7	1/32	....	....	5.85	7.02	8.20	9.36	10.52	
	1/16	....	....	6.46	7.75	9.04	10.32	11.60	
	1/8	....	....	7.66	9.18	10.80	12.23	13.76	
	3/16	....	....	8.86	10.62	12.38	14.15	15.92	
8	1/32	....	....	....	7.82	9.14	10.40	11.70	13.00
	1/16	....	....	....	8.55	9.96	11.38	12.78	14.22
	1/8	....	....	....	9.97	11.63	13.30	14.95	16.62
	3/16	....	....	....	11.40	13.31	15.21	17.12	19.00
9	1/32	....	....	....	.....	10.08	11.52	12.96	14.40
	1/16	....	....	....	.....	10.92	12.48	14.03	15.63
	1/8	....	....	....	.....	12.60	14.40	16.20	18.00
	3/16	....	....	....	.....	14.28	16.32	18.36	20.40
10	1/32	....	....	....	.....	.....	12.65	14.15	15.80
	1/16	....	....	....	.....	.....	13.60	15.30	17.16
	1/8	....	....	....	.....	.....	15.56	17.50	19.40
	3/16	....	....	....	.....	.....	17.50	19.65	22.00

Note. The weight of 32-oz. duck, rubber-covered conveyor belt is one-seventh heavier than above weights.

Capacities for slat or platform conveyors are given in Table 22. In this table several package weights are given for each conveyor width. Stronger chain is used as these package weights increase.

In estimating the cost of apron, pan, slat, or platform conveyors, it should be borne in mind that, in addition to the cost of the chain and attachments, the cost of the sprockets, driving equipment, and supporting structure must be included.

Table 16. List Prices and Average Weights in Pounds for Belt Conveyor Idlers

Link-Belt Company

Width of belt, in.	Unit troughing idler		Unit troughing idler			
			With 5-in. diam. return idler		With 6-in. diam. return idler	
	Price per unit	Weight per unit, lb.	Price	Weight, lb.	Price	Weight, lb.
14	$34.60	85	$45.80	118	$46.40	122
16	35.00	89	46.60	125	47.20	128
18	35.40	93	47.40	131	48.20	135
20	35.80	99	48.20	137	49.20	152
24	36.60	107	49.80	149	51.20	154
30	39.00	127	53.60	179	55.40	186
36	42.00	138	58.00	195	60.20	203
42	45.00	159	62.40	215	65.00	223
48	48.00	164	66.80	237	69.80	245
54	51.00	169	71.20	259	74.60	267
60	54.00	174	75.60	261	79.40	289

Note. Above weights and prices are based on using standard 9-in. pedestal. When 15 in. is used, add 8 lb. to weight and $2.00 to list price.

Table 17. Spacing in Feet of Carrying Idlers for Belt Conveyors

Link-Belt Company

Width of belt, in.	Weight of material in lb. per cu. ft.			
	35 and under	40–70	75–100	125–150
12 & 14	5½	5	5	4½
16	5½	5	5	4½
18	5	4½	4½	4
20	5	4½	4½	4
24	5	4½	4	4
30	5	4½	4	3½
36	4½	4	3½	3
42	4½	4	3½	3
48	4	3½	3½	3½
54	4	3½	3½	3
60	4	3½	3½	3

Spacing of return idlers

12- to 30-in. belt	10-ft. centers
36- to 48-in. belt	9-ft. centers

Chain and Tray Elevators. Barrel, package, bag, or tray elevators are double-strand chain conveyors with various types of attachment for holding such containers, in either ascent or descent. Data for estimate will be similar to those used for other types of chain conveyors, except that the special attachments for holding the load, coming in sets of two, one for each chain strand, will range in cost from $12 to $30, depending on their purpose.

More elaborate conveyors of this type, with automatic unloading, loading, and control attachments are available. If these are to be used, preliminary data should be obtained from the manufacturers.

Table 18. Specifications and Dimensions for Bucket Elevators up to 80-ft. Centers

Palmer-Bee Company

Centrifugal-discharge, malleable-iron buckets, single-strand chain, steel casing

Size of buckets, in.	Spacing of buckets, in.	Diameter of head sprocket, in.	Head shaft, revolutions per minute	Speed of buckets, ft. per min.	H.p. at countershaft	Capacity, tons per hr.
6 × 4	13	16	48	207	1.2	6.7
7 × 4½	13	16	48	207	2.0	10.1
8 × 5	16	20	42	220	2.8	12.3
10 × 6	16	20	42	220	3.1	21.9
11 × 6	16	20	42	220	3.4	23.9
12 × 6	18	24	40	250	4.9	26.5
12 × 7	18	24	40	250	5.5	35.9
14 × 7	18	24	40	250	6.5	42.2
14 × 8	18	24	40	250	8.0	55.0

Table 19. Data for Steel Roller Chain

Jeffrey Manufacturing Company

Pitch, in.	Approximate weight per ft., lb.	Working strength in lb. at 150 ft. per min.	Maximum speed, ft. per min.	Average ultimate strength, lb.	List price per ft.
1.50	3.10	2,100	800	14,000	$2.65
1.50	3.30	1,400	700	10,000	2.00
1.63	3.25	1,400	800	13,000	2.00
1.65	3.00	1,400	700	10,000	1.85
2.00	4.25	1,900	700	13,000	2.50
4.00	12.38	3,700	500	30,700	3.40
6.00	9.25	3,750	450	30,700	2.60
6.00	13.50	5,200	400	50,000	3.00
6.00	13.50	9,250	400	76,300	
9.00	7.70	3,900	350	23,400	2.20
9.00	13.00	4,500	350	44,000	3.00
9.00	14.50	5,200	300	47,000	3.40
9.00	16.00	8,000	315	60,000	3.60
9.00	16.00	8,000	315	60,000	3.55
12.00	12.20	5,200	300	47,000	2.80
12.00	11.92	5,200	300	47,000	2.70
12.00	11.25	5,200	300	47,000	2.40
12.00	14.20	6,500	200	60,000	3.25
12.00	14.00	6,500	200	60,000	3.10
12.00	14.00	6,500	200	60,000	2.80
12.00	16.75	9,700	200	78,500	4.00
12.00	23.65	9,700	200	78,500	4.60

Bucket Carriers. Pivoted bucket and V-bucket carriers are special designs of chain conveyors, similar in the main to bucket elevators, but having upper and lower horizontal runs in addition to the upward and downward runs of the bucket elevators.

Capacities of pivoted bucket carriers are given in Table 23. This table does not include horsepower requirements. Horsepower is given in Table 24, for V-bucket carriers and will not vary greatly for pivoted bucket carriers of similar length and capacity. Prices for estimate purposes may be obtained from the data given in Table 24, for V-bucket carriers. Note that these list prices do not include the driving mechanism.

Flight or Scraper Conveyors. These conveyors are also of the two-strand chain type. In estimating cost, data given for chain conveyors should be followed. Costs for scraper flights and for steel-plate troughs are given on p. 2231.

Capacities given in Table 25 are figured on the basis of dry coal weighing 50 lb. per cu. ft. Capacities differ for other materials of other densities or angles of repose. The capacities given are about 80 per cent of those that would be obtained with perfect loading.

Horsepower requirement for each 100-ft. centers and sufficient for inclines varies from about 2.5 for the first flight size to about 10 for the largest for

Table 20. Data for Chain Trolleys

Chain Belt Company

Chain				Trolley		
Average pitch, in.	Average ultimate strength, lb.	Maximum working load, lb.	Weight plain, per ft., lb.	Wheel diam., in.	Maximum capacity, lb.*	Weight, lb.
2.609	14,000	2,000	3.5	2¼	100	5½
2.609	14,000	2,000	3.5	3⅛	500	7
3.075	24,000	3,200	6.7	3⅛	500	8
4.031	30,000	3,000	3.0	3⅛	500	7
6.031	60,000	6,000	6.2	3⅛	500	7

* Based on a maximum trolley speed of 50 ft. per min.

Table 21. Capacities of Apron and Pan Conveyors in Cubic Feet per Hour

Link-Belt Company

Actual width of carrying surface in.	Height of sides, in.														
	2	3	4	5	6	7	8	10	12	14	16	18	20	24	30
18	112	168	225	281	337	393	450	562	675	787	900	1012	1125	1350	1688
24	150	225	300	375	450	525	600	750	900	1050	1200	1350	1500	1800	2250
30	187	281	375	468	562	656	750	937	1125	1312	1500	1687	1875	2250	2813
36	225	337	450	562	675	787	900	1125	1350	1575	1800	2025	2250	2700	3375
42	262	393	525	656	787	918	1050	1312	1575	1837	2100	2362	2625	3150	3937
48	300	450	600	750	900	1050	1200	1500	1800	2100	2400	2700	3000	3600	4500
54	337	506	675	843	1012	1181	1350	1688	2025	2362	2700	3037	3375	4050	5063
60	375	562	750	937	1125	1312	1500	1875	2250	2625	3000	3375	3750	4500	5625

Table 22. Capacities of Slat or Platform Conveyors

Chain Belt Company
Different chain strengths for each conveyor width

Conveyor width, in.*	Capacity tons per hr.†	Max. weight package, lb.	Min. package spacing, ft.	Conveyor load per ft., lb.
18	45	150	6	25
	67½	150	4	37
	90	150	3	50
	90	200	4	50
24	60	150	4.6	33
	90	150	3	50
	120	150	2.3	65
	135	200	2.6	75
	135	600	8	100
30	75	200	5	40
	112½	200	3.2	62
	150	200	2.4	83
	180	250	2.5	100
	180	600	6	100
36	90	200	4	50
	135	200	2.6	75
	180	200	2	100
	225	250	2	125
	225	600	4.8	125
	252	600	4.3	140
	270	900	6	150
48	270	500	3.3	150
	324	500	2.8	180
	360	700	3.5	200

* Conveyor width equals length of conveyor slats.
† Capacity per hour based on a speed of 60 ft. per min.

Table 23. Data for Pivoted Bucket Carriers

Stephens-Adamson Manufacturing Company
Using standard long-pitch steel-bushed roller chain

Size of bucket, in.	Pitch, in.	Capacity, in tons per hr.		Speed ft. per min.	† Weight of chain and bucket, per ft., lb.
		Coal, 50 lb. per cu. ft.	* Stone, 100 lb. per cu. ft.		
18 × 15	18	15– 20	30– 40	30–40	97
18 × 21	18	22– 30	44– 60	30–40	103
24 × 18	24	40– 50	80–100	40–50	96
24 × 24	24	50– 62	100–124	40–50	106
24 × 30	24	70– 87	140–174	40–50	126
24 × 36	24	90–112	180–224	40–50	136
30 × 24	30	78– 96	156–192	45–55	166
30 × 30	30	100–125	200–250	45–55	176
30 × 36	30	125–150	250–300	45–55	211
36 × 36	36	175–210	350–420	50–60	249

* Based on material 1½ in. and under in size.
† Heavier chain also available for each size.

horizontal conveyors. Inclined conveyors will require more if the quantity handled is not cut down somewhat. Note that these figures are for coal at 50 lb. per cu. ft.

Scraper flights vary in list price from $0.45 for a 12-gage, 4- by 10-in. flight to $2.45 for a 10- by 24-in. flight, $\frac{5}{16}$ in. thick.

Troughs vary in list prices as follows: For 10-gage shallow troughs, from $13 per 10-ft. length to $21.20 per 10-ft. length, over the same range of flight

Table 24. Data and List Prices for Standard V-bucket Carriers

Jeffrey Manufacturing Company

Data					List prices, dollars								
					Terminals			Per foot centers					
Average size material, in.	Maximum size pieces, in.	Capacity tons per hr. (coal)	Bucket size, in. (length by width by depth)	Hp. for maximum centers	Complete *	Less, if drive frame omitted	Less, if head housing omitted	Vertical run complete †	Less, if casing and guide angles omitted	Horizontal run complete ‡	Less, if steel conveyor frame omitted	Valve with standard bifurcated chute, each
3½	7	28	18 × 14 × 7	7	1300	230	296	29.85	17.40	24.45	7.50	120
4	8	42	20 × 16 × 8	9	1380	233	298	31.65	18.00	26.40	7.70	120
4	8	31	20 × 16 × 8	8½	1405	230	306	29.15	18.00	23.35	7.15	120
4	8	28	20 × 16 × 8	6½	1490	230	318	34.20	18.00	28.55	7.30	120
4½	9	47	24 × 18 × 9	11	1780	246	336	35.85	20.70	28.85	8.25	158
4½	9	41	24 × 18 × 9	9	1880	244	348	38.50	20.70	31.15	7.90	158
5	10	63	26 × 20 × 10	13½	1825	249	329	37.40	21.25	30.40	8.25	158
5	10	56	26 × 20 × 10	11½	1885	249	329	41.65	21.25	34.10	8.25	158
6	12	92	30 × 24 × 12	17½	2480	261	359	48.50	24.30	39.51	8.65	167

* Terminal complete includes boot, upper corner section with its housing, curved trough corner section and drive frame, shafts, bearings, sprockets, gears, etc., to and including second countershaft.

† Vertical run complete includes chain and buckets and guide angle with steel casing.

‡ Horizontal run complete includes chain, buckets, trough, and steel conveyor frame for spans not greater than 12 ft. Supports for conveyor frames are extra.

Table 25. Data for Scraper or Flight Conveyors

Chain Belt Company

Flight size, width by length, in.	Capacities in tons of coal per hr., speed 100 ft. per min.						
	Horizontal conveyor				Inclined conveyor flights spaced 24 in.		
	Flight spacing, in.			Lb. moved per flight	10 deg.	20 deg.	30 deg.
	16	18	24				
4 × 10	33.75	30	22.5	15	18.0	14.5	10.5
4 × 12	42.75	38	28.5	19	24.0	18.0	13.5
5 × 12	51.75	46	34.5	23	28.5	22.5	16.5
5 × 15	69.75	62	46.5	31	40.5	31.5	22.5
6 × 18	.....	30	60.0	40	49.5	40.5	31.5
8 × 18	.....	120	90.0	60	72.0	57.0	48.0
8 × 20	.....	...	105.0	70	84.0	66.5	56.0
8 × 24	.....	...	135.0	90	120.0	96.0	72.0
10 × 24	.....	...	172.5	115	150.0	120.0	90.0

sizes as in Table 25. For deep troughs, ¼ in. thick, cost varies from $19.50 per 10-ft. length to $37.80 per 10-ft. length, for this same range of sizes.

Drag-chain Conveyors. Drag-chain conveyors are chains running in a trough, generally of wood, and propelling material that lies upon them along the trough. No flights are used, contact with the chain being sufficient to

Table 26. Data for Drag-chain Conveyors

Chain Belt Company

Capacity in cu. ft. per hr.* speed, ft. per min.		Pitch diameter 8-tooth head sprocket, in.	R.p.m. of head shaft	
30	40		30 ft. per min.	40 ft. per min.
400	533	15.68	7.5	10.0
600	800	15.68	7.5	10.0
800	1,067	20.90	5.63	7.5

* Depth of material conveyed depends on whether material is fine or lumpy. Above capacities are based on a 4-in. depth, which is conservative for fine material. Speeds for abrasive material should not exceed 30 ft. per min. For speeds other than those stated, capacities will be in direct proportion.

Table 27. Sizes, Capacities, and Speeds of Screw Conveyors

Chain Belt Company

Diameter, in.			Standard length, ft.*	Bearing length, in.	Size material in.†		Light, non-abrasive material as grain			Heavy, non-abrasive material as coal			Heavy, abrasive material, as sand and ashes		
Conveyor	Pipe, inside	Coupling			Maximum uniform size	Maximum unsized	Thickness flights, gage or in.	Max. speed, r.p.m.	Capacity‡ per hr. cu. ft.	Thickness flights, gage or in.	Max. speed, r.p.m.	Capacity‡ per hr. cu. ft.	Thickness, flights, in.	Max. speed, r.p.m.	Capacity‡ per hr., cu. ft.
4	1	1	8	1½	⅜	1	18	200	105	10	110	55	3/16	85	45
6	1½	1½	10	2	⅝	1½	16	180	315	10	100	175	3/16	75	130
9	1½	1½	10	2	¾	2¼	14	175	1,100	10	95	600	3/16	70	440
9	2	2	10	2	¾	2¼	14	175	1,100	10	95	600	¼	70	440
10	1½	1½	10	2	⅞	2½	14	160	1,400						
10	2	2	10	2	⅞	2½	14	160	1,400						
12	2	2	12	2	1	3	12	150	2,260	3/16	85	1,280	¼	65	980
12	2½	2 7/16	12	3	1	3	12	150	2,260						
12	3	3	12	3	1	3	12	150	2,260	3/16	85	1,280	⅜	65	980
14	2½	2 7/16	12	3	1⅛	3½	12	150	3,600	3/16	80	1,930	¼	60	1,450
14	3	3	12	3	1⅛	3½	10	150	3,600	3/16	80	1,930	⅜	60	1,450
16	3	3	12	3	1⅜	4	10	150	5,400	¼	75	2,700	⅜	55	2,000
18	3	3	12	3	1½	4½	10	130	6,600	¼	65	3,300	⅜	50	2,550
20	3½	3	12	3	1¾	5	3/16	115	7,900	¼	60	4,100			
20	4	3 7/16	12	4	1¾	5	..	...	.....	⅜	60	4,100	½	45	3,100
24	4	3 7/16	12	4	2	6	¼	100	12,100	⅜	50	6,050	½	40	4,800

* Standard length of conveyor section, measured from center to center of bearings; actual length of flights equals standard length minus bearing length.

† Approximately 90 per cent of material to be of "Max. uniform size" stated; not more than 10 per cent of unsized material to be of "Max. unsized" stated.

‡ Capacities stated are at maximum r.p.m. with uniform and continuous flow of material, based on cross-sectional area of material in box equal to about one-third of the area of the screw. Capacities at other speeds are in direct proportion to those listed.

Size of the material handled should never exceed one-fourth of the diameter of the screw.

The above data on speeds and capacities represent good practice, but may be departed from, depending on conditions. It is recommended that the maximum speeds shown be not exceeded.

Table 28. List Prices of Standard Screw-conveyor Flights*

Chain Belt Company

Diameter, in.			List price in dollars per ft. of standard black conveyor					
			On heavy pipe core					On standard pipe core, standard weight
			Thickness steel flights, gage or in.					
Conveyor	Pipe, inside	Couplings,	No. 10	3/16	1/4	3/8	1/2	
4	1	1	$3.00	$ 3.25	$ 3.75	......	......	$ 2.75
6	1½	1½	3.25	4.00	5.00	$ 8.00	......	3.00
9	1½	1½	4.25	5.00	5.75	......	......	4.00
9	2	2	5.00	5.75	6.50	8.50	......	4.50
12	2	2	.....	6.25	7.00	......	......	4.50
12	3	3	.....	7.50	8.25	11.00	......	5.00
12	3½	3	.....	......	......	......	$16.00	5.00
14	2½	2 7/16	.....	8.00	9.00	......	......	6.00
14	3	3	.....	9.00	10.00	13.00	......	7.00
14	4	3	.....	......	......	......	18.00	7.50
16	3	3	.....	9.50	10.50	14.00	......	8.50
16	4	3	.....	......	......	......	20.00	9.00
18	3	3	.....	12.00	14.00	19.00	......	11.00
18	4	3	.....	......	......	......	25.00	
20	3½	3	.....	17.00	19.00			
20	4	3 7/16	.....	......	......	25.00	30.00	
24	4	3 7/16	.....	23.00	25.00	35.00	45.00	

---

* Standard lengths same as Table 27.

Table 29. List Prices of Screw-conveyor Accessories

	Price Range
Cast-iron box ends for wooden boxes	$ 2.00–$ 17.00
Cast-iron box ends for steel boxes	2.00– 17.00
Discharge box ends	1.75– 18.00
Countershaft box ends, wooden boxes	21.00– 126.00
Countershaft box ends, steel boxes	21.00– 126.00
Right-angle drives	38.00– 205.00
Hanger bearings	2.00– 20.00
Steel linings for wooden boxes, per foot	0.25– 7.00
Steel troughs with covers, per ft	3.50– 30.00
Flanges	1.75– 13.00
Saddles	1.25– 8.00
Gates and spouts (depending on design)	1.50– 110.00

Table 29A. Sizes, Speeds, and Capacities of Johns Conveyors

Osborn Manufacturing Co.

Conveyor size, in.	Capacity, tons per hr.									
	Materials at 100 lb. per cu. ft. Conveyor speeds, ft. per min.					Materials at 50 lb. per cu. ft. Conveyor speeds, ft. per min.				
	100	200	300	400	500	100	200	300	400	500
2	6	12	18	24	30	3	6	9	12	15
3	14	28	42	56	70	7	14	21	28	35
4	24	48	72	96	100	12	24	36	48	60

move the class of materials, such as ashes, on which drag chains are used. In general, these conveyors should be estimated just as other chain conveyors are. Data as to capacities and speeds are given in Table 26. Horsepower required for these conveyors can be estimated from the data given and the weight of the material, due allowance being made for friction. Cost can be estimated by estimating the cost of the structure, sprockets, and drive, and to that adding list prices of $1.80, $2.50, or $3.40 per ft. for the chain in the three cases given in the table.

Table 30. Handling Capacities of Drag-line Scrapers

Sauerman Bros., Inc.

Estimated hourly capacities in cubic yards for different sizes of scrapers operating on various lengths of haul

Length of haul, ft.	Size of scraper, cu. yd.										
	⅓	½	¾	1	1½	2	2½	3	4	5	6
100	22	34	51	80	120	160	200	240	320	400	480
200	12	18	27	42	63	84	105	127	168	210	254
300	8	12	19	28	42	56	70	84	112	140	168
400	6	10	14	21	32	43	53	63	86	106	126
500	5	7	11	17	26	34	43	52	68	86	104

The above handling capacities are based upon line speeds attained in average material. Actual capacities will vary from this table by amounts depending entirely upon local conditions.

Screw Conveyors. Sizes, capacities, and speeds usual with screw conveyors are listed in Table 27. List prices are in Table 28. The horsepower required to drive a screw conveyor can be determined by applying the formula

$$\text{Horsepower} = \frac{W \times L \times C}{33{,}000}$$

where W is weight of material moved in pounds per minute; L is length of conveyor in feet; $C = 1.3$ for grain and similar material; 2.5 for coal, cement, etc.; and 4.0 for ashes, sand, and gravel. Horsepower obtained by this formula will be conservative with due allowances for ordinary misalignment and other variable factors.

For this same range of screw-conveyor sizes, the list-price ranges given in Table 29 may be used for purposes of estimating.

Gravity Roller Conveyors. Gravity roller conveyors come in two general types and several sizes and bearing designs. The two types are two rail and three rail. Two-rail conveyors have one series of rollers held between frame

Table 31. Drag-line Scraper, Storing and Reclaiming Coal

Sauerman Bros., Inc.

Size of scraper bucket, cu. yd.	Average haul, ft.	Capacity, tons per hr.	Operating cost per ton (approximate)	Cost of machine (approximate)	H.p. of motor
½	200	18	$0.045	$ 2,500– 4,000	15
3	200	100	0.040	11,000–16,000	100
4½	200	170	0.035	13,000–18,000	125
7	200	265	0.030	20,000–30,000	200
10	200	365	0.028	26,000–40,000	300

These machines are made in 15 sizes from ¼-cu. yd. to 12-cu. yd. capacity. They operate between head and back posts, or between movable steel towers, and for spans up to 600 or 700 ft. The costs given are for certain representative machines, and cover both storing and reclaiming. Operating costs vary with distance material is moved, continuity of operation, etc. The first cost varies widely because of the great difference in the layouts of different storage yards.

rails on either side. Three-rail types have two sets of rollers, parallel, and held between three frame rails, one being in the center between the two sets of rollers.

Table 32. Drag-line Scraper, Storing and Reclaiming Crushed Stone

Sauerman Bros., Inc.

Size of scraper bucket, cu. yd.	Average haul, ft.	Capacity, tons per hr.	Operating cost per ton (approximate)	Cost of machine (approximate)	H.p. of motor
½	100	50	$0.06	$ 5,000	20
1½	200	90	0.08	10,000	75
3	300	150	0.09	20,000*	125
5	400	275	0.095	30,000*	200

These machines are made with 12 different sizes of buckets, ranging from ¼-cu. yd. to 8-cu. yd. capacity, operating between masts, or between towers, and for spans up to about 600 ft. Costs are for representative installations. Costs vary with distance material is moved, layout, continuity of operation, etc.

* Machines of 3-cu. yd. size and larger usually have a movable steel head tower and the above cost is figured accordingly.

Straight sections of roller conveyors are 4 or 8 ft. long as desired. Special lengths, curves, switches, etc., are made to order.

The light-duty roller conveyor has rollers of 1-in. outside diameter on light ball bearings. Length of rollers ranges from 4 to 20 in. for the two-rail type and from 6 to 20 in. for three rail. Capacity is 20 lb. continuous load for each roller.

The standard-duty roller conveyor has rollers of 1⅝-in., 1⅞-in., or 3¼-in. diameter, on ball bearings. The length of roller is 4 to 24 in. for the two-rail type and 6 to 24 in. for the three-rail type. Capacity is 100 lb. continuous load for each roller.

Heavy-duty roller conveyors with rollers from 2⁹⁄₁₆-in. to 4-in. diameter, and capacities up to 5,000 lb. continuous load per roller, are also made.

First cost of gravity roller conveyors runs from $1 per ft. to $25 per ft. depending on size, load capacity, and roller spacing.

Wheel Conveyors. Wheel conveyors are made with antifriction bearings and with steel, rubber, fiber, or molded plastic composition tires. Wheel diameters available vary from 2 to 8 in. and continuous load capacities from 20 to 300 lb. per wheel. The preferred application of these conveyors is in installations where conveyor lines must be readily portable and where the relatively light weight of wheel conveyor sections becomes a distinct advantage. Costs are slightly below those of gravity roller conveyors.

Redler Conveyors. Capacities of Redler conveyors go as high as 300 tons per hr. when handling heavy materials at high speed. However, with the light materials usually handled by these conveyors (weights less than 100 lb. per cu. ft.) and with usual size conveyors, capacity will be in the range of 1 to 75 tons per hr. Redler conveyors come in a number of designs, but for the design most usually employed, flight sizes are from 3 to 23 in. The smallest of these will handle lumps up to ¾ in. in unsized material, while the largest will handle lumps up to 5¾ in. in unsized material. Speeds range up to 180 ft. per min. for the largest size and up to 80 ft. per min. for the smallest size. Costs of Redler conveyors vary greatly, depending upon the job to be done, and the figures for estimating purposes should be obtained from the manufacturer.

Johns Conveyors. Sizes, capacities, and speeds of Johns conveyors are given in Table 29*A*. Cost figures for estimating purposes should be obtained from the manufacturer.

Pneumatic Conveyors. Capacities and list prices of pneumatic handling systems should be obtained from a manufacturer for each case that arises. For estimate purposes it may be assumed that in a suction system, to move 20 tons per hour through 50 ft. requires 50 h.p., the same quantity through 150 ft. requires 75 h.p., and through 250 ft., 90 h.p.

An installation for moving 10 tons per hour of material weighing 40 lb. per cu. ft., over a distance of 300 ft. cost approximately $10,000. The motor was 78 h.p. and the actual power used in handling was 70 h.p.

Systems are built for handling as much as 250 tons per hour when handling heavy, granular materials. For light, dusty materials the limit is about 50 tons per hour. Handling 10 tons per hour a distance of 150 ft., the cost for power and labor will ordinarily be in the neighborhood of $0.10 per ton.

Drag-line Scraper Conveyors. The capacity of drag-line scraper equipment depends on the rating of the scraper and the length of the haul. Tables 30, 31, and 32 show the capacity, costs, and other data for one design of this type of conveyor.

Table 33. Capacity, Lift, and List Prices of Differential Chain Hoists
Yale & Towne Manufacturing Company

Capacity, tons	Price complete	Standard lift,* ft.	Extra lift, price per ft.†
¼	$36	6	$4.80
½	42	7	4.80
1	56	8	5.00
1½	72	8½	5.40
2	90	9	5.60

* "Standard lift" means hook travel.
† Each foot of extra hoist includes 4 ft. of chain.

Table 34. Capacity, Lift, and List Prices of Screw-geared Chain Hoists
Yale & Towne Manufacturing Company

Capacity, tons	Price complete	Standard lift, ft.	Extra lift, price per ft.
½	$ 50	8	$2.50
1	60	8	2.60
1½	80	8	2.70
2	100	9	2.80
3	150	10	3.00
4	190	10	3.80

Hand-operated Chain Hoists. There are three types of hand-operated chain hoist: the differential, the screw geared, and the spur geared. Of these types the differential has an efficiency of 30 and the screw geared, 40 per cent, while the spur geared has an efficiency of 80 per cent. Tables 33, 34, and 35 give data as to capacity, lift, and list price for the three types of hoist.

These hoists are frequently supported by means of a trolley running on an I-beam. Trolleys are of various types, of which perhaps the most common is the steel-plate trolley, data for which are given in Table 36.

Motor-driven Chain Hoists. Chain hoists are equipped with electric motors for easy and quicker handling of loads. Table 37 gives data for one of the simpler types of this hoist. Speeds for motor-driven hoists are given in Table 38 for both direct and alternating current.

Drum Hoists. There are several designs of drum hoist of varying complication and cost. That for which data are given in Table 39 is a simple type of friction hoist equipped with pulley for drive from a motor or other power source. It is supplied in both single- and double-friction designs.

Monorail Hoisting and Conveying Systems. Monorails are of two general types, the hand operated and the motor operated. The hand-operated type is made up of a chain hoist, a carrier for the hoist, and the supporting track or rail. Rail of this type, in lengths of 21 ft., lists at $0.65 per ft. Table 40 gives data as to capacities, lifts, and list prices for this type. It covers

Table 35. Capacity, Lift, and List Prices of Ball-bearing, Spur-geared Chain Hoists

Yale & Towne Manufacturing Company

Rated capacity, long tons	Price complete	Standard lift, ft.	Extra lift, price per ft.
¼	$ 70	8	$ 1.80
½	70	8	1.80
1	90	8	1.90
1½	120	8	2.00
2	140	9	2.10
3	180	10	3.00
4	220	10	3.20
5	280	12	4.30
6	330	12	4.30
8	400	12	5.40
10	480	12	6.50
12	600	12	8.60
16	720	12	10.80
20	850	12	13.00

Table 36. Data for Steel-plate Trolleys

Yale & Towne Manufacturing Company

Capacity, tons	Standard size of I-beam,† in.	Greatest distance between supports, ft.	Diameter of tread wheels, in.	Price plain trolley
¼	4	10	3	$ 22
½*	5	14	3¾	25
1	6	14	4¾	29
1½	7	15	5½	39
2	8	16	6½	46
3	9	16	7¼	61
4	10	16	8¼	85
5	12	18	10	109
6	15	21	10	134
8	20	28	12	170
10	24	32	13	218
12	24	29	13	295
15	24	26	18	485
20	24	21	18	585

* ½ ton and larger are equipped with roller bearings.

† Adjustable without extra cost for I-beams three sizes larger than standard. Also adjustable for I-beams of non-standard slant.

Table 37. Data for Electric Motor-driven Chain Hoists with Manual Control

Yale & Towne Manufacturing Company

Combination of hoist with	¼-ton quick speed ½-ton standard speed		½-ton quick speed 1-ton standard speed		2-ton standard speed	
	Standard 10-ft. lift	Extra lift per ft.	Standard 10-ft. lift	Extra lift per ft.	Standard 10-ft. lift	Extra lift per ft.
Hook or clevis	$310	$0.92	$417.50	$1.05	$519.00	$1.85
Plain trolley	320	0.92	430.00	1.05	540.00	1.85
Plain trolley with current collectors	330	0.92	440.00	1.05	550.00	1.85
Geared trolley	342	1.62	448.75	1.75	561.50	2.55
Geared trolley with current collectors	352	1.62	458.75	1.75	571.50	2.55
Built-in trolley	330	0.92	440.00	1.05	550.00	1.85
Built-in trolley with current collectors	340	0.92	450.00	1.05	560.00	1.85

Table 38. Speeds of Motor-driven Chain Hoists with Direct and Alternating Current

Yale & Towne Manufacturing Company

Direct Current

Rated capacity, tons*	Speeds, ft. per min.				
	Hoisting			Lowering	
	No load	Half load	Full load	No load	Full load
¼ Q.S.	72	46	36	72	56
½	36	23	18	36	28
½ Q.S.	60	40	32	60	45
1	40	25	20	40	30
1 Q.S.	54	34	29	54	38
2	20	12½	10	20	15
2 Q.S.	27	17	15	27	18

Alternating Current

Rated capacity, tons*	Speeds, ft. per min.					
	Polyphase		Single phase			
			Hoisting		Lowering	
	Hoisting	Lowering	No load	Full load	No load	Full load
¼ Q.S.	32	34	72	40	72	66
½	16	17	36	20	36	33
½ Q.S.	35	38	54	32	54	45
½ H.S.	52	57				
1 L.S.	6	7				
1	18	19	36	17	36	28
1 Q.S.	26	29				
2 L.S.	3	3½				
2	9	9½	18	8½	18	14
2 Q.S.	13	14				

* Q.S. is used as abbreviation for quick speed; H. S. for high speed; L.S. for low speed.

only the simple design of hand-operated monorail. Many different special carriers, hoists, and attachments are available for specialized services.

Hand-propelled monorails with electrically operated hoists are available in many sizes. Table 41 gives data for a standard design of this type. Other special designs are on the market.

Electrically propelled, motor-driven monorails are operated from the floor or by an operator seated in the carrier. Data for the floor-operated type are given in Table 42, and for the type operated from a cab in Table 43.

Traveling Cranes. Traveling cranes are of two general types, hand operated and motor operated. The hand-operated type is supplied in many sizes by different makers. Data for one manufacturer's hand-operated cranes are given in Table 44. As all cranes are more or less specially built, list prices are not available.

Light-weight, high-speed, motor-operated cranes are operated from the floor, as are the hand-operated type. Table 45 gives data for these cranes.

Table 39. Data for Friction Drum Hoists

Stephens-Adamson Manufacturing Company

Single friction

Size of drum, in.	Rope pull, lb.	Speed, ft. per min.	Max. size of rope, in.	Diam. flanges, in.	Drive pulley			Weight, lb.	List price
					Diam., in.	Face, in.	R.p.m.		
12 × 19	1300	150	3/8	18	30	6	200	1300	$ 300.00
12 × 24	1300	150	3/8	18	30	6	200	1400	322.50
18 × 18	2000	300	1/2	24	36	8	300	2600	435.00
18 × 24	2000	300	1/2	24	36	8	300	2800	465.00
20 × 24	2000	300	9/16	26	36	8	270	3000	495.00
24 × 24	2300	300	5/8	30	36	8	250	3600	600.00
24 × 24	3000	300	5/8	30	36	10	300	4300	705.00
24 × 30	3000	300	5/8	30	36	10	300	4700	750.00
Double Friction									
24 × 24	3000	300	5/8	30	36	10	300	4700	$ 973.50
24 × 30	3000	300	5/8	30	36	10	300	4900	1006.50
24 × 36	3000	300	5/8	30	36	10	300	5200	1039.50
30 × 30	4000	300	3/4	36	36	12	300	5500	1105.50
30 × 36	4000	300	3/4	36	36	12	300	5700	1138.50
36 × 36	5000	300	7/8	42	40	12	300	7000	1320.00
36 × 42	5000	300	7/8	42	40	12	300	7200	1369.50

Table 40. Data for Hand-operated Monorails

Cleveland Crane & Engineering Company

Capacity, tons	Standard lift, ft.	Shipping weight, lb.	List price, carrier only	List price, hoist only	Price per extra ft. of lift
1/10	8	126	$20	$ 60	$1.80
1/4	8	135	20	60	1.80
1/2	8	140	20	70	1.80
1	8	165	20	90	1.90
1 1/2	8	295	55	120	2.00
2	9	300	55	140	2.10

Heavy-duty traveling cranes of the motor-operated type can be obtained to suit almost any requirement. Manufacturers of such equipment should be consulted for data for each specific application.

Table 41. Data for Hand-propelled, Motor-driven Monorails

Cleveland Crane & Engineering Company

Capacity, tons	Speed, ft. per min.	Max. lift, ft.	Motor, h.p.	List price*
¼	40–80	16–30	1	$246–364
½	20–80	16–28	1½	273–476
1	20–80	17–30	3	368–620
2	20–56	18–30	5	503–764

* Depending on control and lift.

Table 42. Electrically Propelled, Motor-driven Monorails, Floor Operated

Euclid Crane & Hoist Company

Capacity, tons	Standard lift, ft.	Lifting speed, ft. per min.	Minimum h.p. motor	Size rope, in.	110–440 volt 3 phase 60 cycle	110–220 volt direct current	Weight, lb.
½	18	25–30	1½	5/16	$ 325.00	$ 338.00	515
½	40	25–30	1½	5/16	350.00	363.00	625
1	9	15–20	2	5/16	377.00	390.00	515
1	20	15–20	2	5/16	400.00	415.00	625
1	30	20–25	2	3/8	513.00	513.00	880
2	10	10–15	2	5/16	460.00	475.00	650
2	15	15–20	3	3/8	528.00	528.00	900
3	13	10–15	3	7/16	578.00	568.00	900
3	20	10–15	3	3/8	749.00	749.00	1745
3	36	10–15	3	7/16	749.00	749.00	1745
5	18	10–15	5	7/16	840.00	890.00	1775
5	28	15–20	7½	7/16	1098.00	1040.00	3140
7½	24	10–15	7½	½	1098.00	1040.00	3140
10	16	10–15	10	½	1235.00	1189.00	3300
10	25	20–25	20	9/16	1585.00	1732.00	5000
15	16	15–20	20	9/16	1735.00	1882.00	5100

Table 43. Electrically Propelled, Motor-driven Monorails Operated from Cab

Euclid Crane & Hoist Company

Capacity, tons	Standard lift, ft.	Lifting speed, ft. per min.	Minimum h.p. motor	Size rope, in.	110–440 volt 3 phase 60 cycle	110–220 volt direct current	Weight, lb.
1	16	20–25	2	5/16	$1290.00	$1210.00	2180
1	30	20–25	2	3/8	1290.00	1210.00	2180
2	15	15–20	3	3/8	1305.00	1225.00	2200
3	13	10–15	3	7/16	1505.00	1415.00	2200
3	20	10–15	3	3/8	1720.00	1640.00	3720
3	36	10–15	3	7/16	1720.00	1640.00	3720
5	18	10–15	5	7/16	1830.00	1830.00	3760
5	28	15–20	7½	7/16	2218.00	2110.00	6045
7½	24	10–15	7½	½	2218.00	2110.00	6075
10	16	10–15	10	½	2597.00	2474.00	6300
10	25	20–25	20	9/16	2982.00	3182.00	7360
15	16	15–20	20	9/16	3132.00	3332.00	7450

Hand-operated Lift Trucks. Hand-operated lift trucks are available in a great variety of sizes and capacities. Table 46 gives information that may be used in estimating for this type of equipment.

Skid Platforms. The U. S. Department of Commerce, in cooperation with manufacturers and users, has standardized on 7-in. and 11-in. heights for lowered skid platforms. Two sizes have been selected as meeting general requirements. The smaller measures 33 by 54 in. and can be loaded to 4000 lb. if the height of the load is not too great. The larger measures 42 by 66 in. and is recommended for bulky material. The smaller skid can be loaded three abreast in standard boxcars, while the larger can be loaded two abreast. Skid platforms are listed at from $3.50 to $15 each, depending upon size, capacity, and type.

Table 44. Hand-operated Traveling Cranes
Whiting Corporation

Capacity, tons	Span, ft.	Maximum lift, ft.
5	20–50	32
10	20–50	32
15	20–50 or up	45
20	20–50 or up	45

Table 45. Data for Light-weight, Motor-operated Traveling Cranes
Whiting Corporation

Capacity, tons	Span, ft.	Capacity, tons	Span, ft.
1	15–20	3	25–35
2	20–25	5	35–40

Table 46. Data for Hand-operated Lift Trucks
Lewis-Shepard Company

Type	Maximum capacity, lb.	Minimum lift, in.	No. of strokes for maximum lift	List price, (depending on size)
"Single lift"	2,500	1¾	1	$ 60–130
"Jacklift"	2,500	1¾	4	113–435
"Jacklift"	5,000	1¾	7	
"Jacklift"	8,000	1¾	8	
"Jacklift"	10,000	1¾	8	

Portable Platform Elevators. Hand-operated and motor-operated portable platform elevators are used for raising and lowering loads. For loads not exceeding 500 lb., elevating up to 10 ft. in height, a gearless hand-operated machine is available at a list price of about $200. Higher priced, geared, hand-operated elevators, made for elevating up to 20 ft., are also available.

Combination hand-operated and motor-operated elevators in sizes of 500 lb. and up to 40 ft., 750 lb. and up to 27 ft., 1000 lb. and up to 20 ft., 1500 lb. and up to 13 ft., and 3000 lb. and up to 10 ft. are available.

Fully electrically operated portable platform elevators are available in sizes as shown in Table 47.

Data supplied by the Lewis-Shepard Company indicate that list prices for portable platform elevators range from slightly less than $200 to $1750 for an elevator of 4 tons capacity.

Permanently Installed Freight or Platform Elevators. For elevators of this type, permanently installed in buildings for the handling of all types of loads, specifications and prices should be obtained from a manufacturer, by supplying him with all necessary data.

Table 47. Data for Electric Portable Platform Elevators

Barrett-Cravens Company

Capacity, lb.	Speed, ft. per min.				Lifting height, max. ft.
	½ h.p.	2 h.p.	3 h.p.	5 h.p.	
500	60	..	..	..	40
750	40	53	..	..	27
1000	30	40	60	..	20
1500	20	27	40	66	13
2000	15	20	30	50	10
3000	10	13	20	33	20
4000	7½	10	15	25	20
5000	6	8	10	20	20

Table 48. Data for Electric Industrial Trucks

Baker-Raulang Company

Model	Capacity, lb.	List price less battery	List price of batteries
Platform	4,000– 6,000	$1700–2000	$400–1000
Lift	4,000– 6,000	2000–2250	
Lift	10,000–20,000	2800–3500	
High-lift	4,000– 6,000	2250–2750	
High-lift	10,000–20,000	3250–5000	
Fork	3,000– 8,000	2500–4500	
Crane	3,000– 6,000	3750–4200	

Electric Trucks. Many types and capacities of electric trucks are available. The 2-ton model is much used where standard types are suitable. Table 48 gives information as to capacities and prices.

Speeds of electric industrial trucks when loaded are from 4 to 6 miles per hour; and when unloaded, about 50 per cent greater. These figures are for operation on level concrete. On grades, speeds are somewhat less as also on rough surfaces. Larger model trucks, suitable for use in warehouses, wharfs, and terminals, have load capacities from 50 to 100 per cent over those in the table and operate at speeds from 7 to 10 miles per hour.

Skid platforms used with these trucks are similar to those used with hand-operated lift trucks.

Tractors and Trailers. According to the Mercury Manufacturing Company, the average cost of an electric industrial tractor with battery is about $2500; the average cost of a gas-engine industrial tractor is about $1350; and the average cost of a trailer is $60. It is usual practice to employ 50 trailers for each tractor. Table 49 gives data as to capacities and list prices of tractors. Ball-bearing platform trailers, 72 by 36 in. and 16½ in. high, weigh 500 lb. and have a capacity of 2 tons load.

Industrial Railways. Equipment of the industrial-railway type is made in a variety to suit every purpose. As installations of this equipment are always special and of considerable magnitude, a manufacturer should be consulted before any attempt is made to estimate costs.

Cost of Handling Materials

Reliable data from operating experience for estimating the cost of operation of handling equipment by conveyors, overhead equipment, or industrial railways are not available. This is due to the fact that no two installations are at all alike in results obtained. All the conditions affecting cost vary in each installation. Distances moved, quantities handled, use-factor of equipment, cost of power, and cost of labor, among other factors, are of such a wide range as to prevent the determination of accurate unit costs.

Handling by trucks and tractors has been the subject of much study and fairly reliable unit costs for estimating purposes are available.

Hand-operated lift trucks, handling miscellaneous material about a warehouse, showed a handling cost of about $0.45 per ton in one job. These trucks and their accompanying quota of skid platforms have a yearly maintenance cost of from 0.5 to 2 per cent of the first cost. Many lift trucks of this type average about $1 per year for maintenance.

Portable platform-elevator maintenance averages about $15 per year. Cost of handling can be estimated from the power required over the period of operation and the labor expense.

Table 49. Data for Electric Industrial Tractors

Baker-Raulang Company

Type	Capacity, lb. draw-bar pull	List price less battery
Four-wheel		
Two-wheel drive....	2000	$1850
Four-wheel drive....	3200	2050
Three-wheel...........	1800	1575

Electric industrial trucks, according to the Industrial Truck Association, have yearly operating charges as shown in Table 50. Handling costs for unloading 1,000,000 lb. of billets per day and moving these 150 ft. are $0.02 per ton. For handling 48 rolls of paper per hour over 300 ft. and stacking four high, costs are 3⅓ cts. per ton. This same association also finds that the cost of operating these trucks, in average service, can be figured at $0.50 per hr.

The Baker-Raulang Company gives the following costs per day for operating a truck in average service: platform trucks, $2.50; lift trucks, $3 to $3.50; high-lift trucks, $3 to $3.50; fork trucks, $3.25; crane trucks, $4.25.

Tractors, according to the Baker-Raulang Company, cost $3 per day to operate. The Mercury Manufacturing Company gives the same figure, $3, for electric tractors, exclusive of the operator's wages, and gives the cost of operating a gasoline industrial tractor, exclusive of the operator's wages, at $4 to $4.50 per day. This latter company finds that the total cost of owning and operating trailer units is $0.03 per day per unit.

Drag-line scraper conveyor costs have been obtained by Sauerman Bros., Inc., from a large number of installations of their equipment. These figures are given in Tables 31 and 32, and can be summarized here as follows: for storing and reclaiming coal, from \$0.028 to \$0.045 per ton for a 200-ft. haul; for storing and reclaiming crushed stone, from \$0.06 to \$0.095 per ton for a haul varying from 50 to 275 ft.

Costs of handling by conveyors or overhead equipment can most readily be estimated by considering each conveyor or other unit separately. From the load handled, with a reasonable assumption of power losses, the power required can be estimated. Cost of power per kilowatt-hour, multiplied by the number of kilowatts of power required, multiplied by the hours of use will give the cost of power for a given amount of handling. Dividing this cost by the pounds, tons, or other quantity units of the material handled will give the power cost per unit handled.

Cost of labor can be estimated in the light of the current wage rate in the locality where the equipment is to be located, by analyzing the handling operation and estimating as nearly as possible the amount of attendance necessary. It should be borne in mind that frequently one man can attend to several duties and divide his time between the handling equipment and other work. Often additional investment in automatic loading, unloading, and controlling devices will be justified by the decrease it causes in handling-labor expense.

Maintenance cost for equipment, unless the service be unusually severe, can usually be estimated at between 2 and 5 per cent of the cost of the equipment.

Table 50. Yearly Operating Charges for Electric Industrial Trucks

Industrial Truck Association

Depreciation of truck	\$200.00
Depreciation of battery	100.00
Depreciation of charging equipment	15.00
Interest, insurance, and storage charges for the above	245.00
Power (7500 kw.-hr. at 1½ cts.)	112.50
Maintenance and repairs, labor and material	250.00
	\$922.50

Depreciation can in most cases be estimated on the basis of a life of 20 years for permanently installed equipment and 10 years for movable or portable equipment.

Interest on investment, taxes, insurance, and the possible effect of obsolescence also should be considered in estimating costs of materials-handling equipment.

MOVEMENT OF LIQUIDS

Necessity for the movement of liquids occurs widely throughout the chemical engineering industries. Raw materials are delivered to the plant as liquids. Many materials are liquids during all or part of the manufacturing processes. Finished products are frequently liquids.

Some of these liquids are handled in containers and are therefore subject to the handling methods described for solid materials and materials in containers. Other liquids are handled in bulk and are therefore handled by some form of *pump* through pipe lines. This method of handling will be discussed in this section.

Raw materials handled by pumping include such materials as water, petroleum and its products, brine, acids, alcohols, liquid alkaline solutions, liquid alkalies, salt solutions and liquid salts, liquid chemicals such as tanning liquids, molasses, syrups, vegetable and animal oils, and tar and other liquid coal products.

Materials in manufacture that are handled in a liquid condition are in general the same as the raw materials, with the addition of such materials as slurries, sludges, and slips, which are mechanical mixtures of a solid and a liquid that are usually best handled as liquids. Various solutions and suspensions will be encountered during manufacture that are not found among either the raw materials or the finished products, but, generally speaking, the pumping problems caused by these are not different from those encountered with other liquids of similar characteristics.

Finished products that are pumped are in general similar in physical properties to the liquids encountered among the materials in process and the raw materials. Consequently the pumping problems encountered fall in the same general classifications.

Pumping Equipment. For the purposes of this section relating to the movement of liquids, a *pump* is defined as any device for the transference of liquids. More broadly speaking, pumps also include devices for handling gases, but the subject of the movement of gases is covered in the latter portion of this section.

Types of pumps that are of importance in the chemical engineering field include:

1. Centrifugal pumps, including sump pumps, and propeller or screw-impeller type pumps and deep-well pumps.
2. Reciprocating, or piston pumps, including steam pumps and power pumps.
3. Rotary pumps.
4. Air-pressure pumps, including air lifts; and displacement pumps such as acid eggs, automatic elevators, pneumatic pumps, pneumatic ejectors, inflammable-liquid pumps, Humphrey gas pumps.
5. Pulsometers.
6. Jet pumps, including injectors and ejectors, operated by steam or water.
7. Hydraulic rams and hydrautomats,
8. Special pumps, including the screws, the scoop wheel, etc.
9. Diaphragm pumps.

Centrifugal Pumps

A **centrifugal pump,** in its simplest form, consists of an impeller rotating within a casing. The **impeller** consists of a number of blades, either open or shrouded, mounted on a shaft which projects outside the casing at each side. **Impeller blades** may be curved forward (for low-head pumping), straight (for medium-head pumping), or curved backward (for high heads). Impellers may have their axis of rotation either horizontal or vertical, to suit the work to be done.

Casings are of four general types, but in any case consist of a chamber in which the impeller rotates, provided with inlet and exit for the liquid being pumped. The simplest form of casing is the **circular casing,** consisting of an annular chamber around the impeller, no attempt being made to overcome the losses that will arise from eddies and shock when the liquid leaving the impeller at relatively high velocities enters this chamber. Such casings are not used in the usual designs of centrifugal pumps but are widely used in special pumps as, for instance, in pumps made of acid-resisting materials, where pump efficiency is sacrificed in order to permit the use of the special material. **Volute casings** are a modification of the above, wherein the chamber takes the form of a volute, increasing in cross-sectional area as the outlet is approached.

This form is a distinct improvement over the previous form, in that it decreases the losses, although loss is still incurred due to the fact that the velocity of the liquid leaving the blades is higher than the velocity of the liquid in the volute chamber. The volute casing is the most used form. A third type of casing interposes a **whirlpool chamber** between the volute and the impeller, which effects a further decrease in losses but adds to the size and cost of the casing. In practice, a compromise is usually effected by cutting down both the volute and whirlpool chambers. A fourth type of casing, used where losses must be at a minimum, as in boiler-feed practice or high-head pumping, is called the **turbine pump.** In this type, guide vanes are interposed between the impeller and the circular casing chamber. In passing through these vanes, the velocity head of the liquid leaving the impeller blades is converted to pressure head, and shock or eddy losses in the casing are thus avoided.

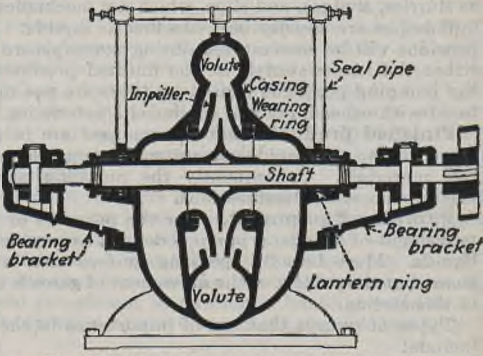


Fig. 32.—Single-stage volute centrifugal pump.

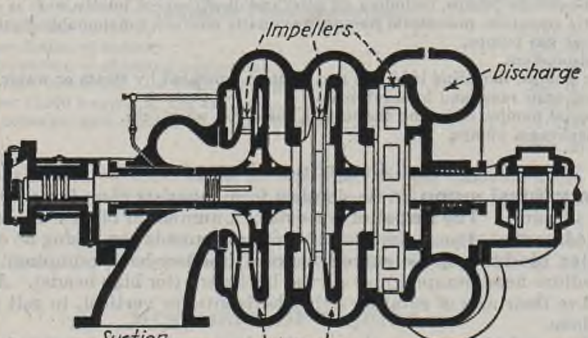


Fig. 33.—Multistage centrifugal pump.

Centrifugal pumps are available in two general classes: *standard* for all services involving liquids not highly corrosive, viscous, dense, or not carrying too large a quantity of solid or abrasive material; *special*, for liquids involving severe problems of corrosion or abrasion, liquids of high viscosity or density, or liquids carrying a large quantity of solids.

Standard Centrifugal Pumps. As generally applied, these pumps are divided into two groups: **single-stage volute pumps** (Fig. 32), for low or medium heads; **multistage pumps** (Fig. 33) for high heads.

Volute pumps are of four general styles: single suction, open impeller; single suction, closed impeller; double suction, open impeller; double suction, closed impeller. The double-suction pumps (Fig. 32) are the usual choice, because of their inherent hydraulic balance.

Multistage pumps have two or more impellers in series, in order to increase the head produced. These pumps may or may not be provided with vanes. Where vanes are not used the casings have circular or volute chambers, depending on the application.

Standard centrifugal pumps are generally designated according to the diameter of the discharge opening. While this figure is not a measure of pump capacity, which varies with the speed, the head pumped against, the area of the passages within the pump, and other factors, it does give a useful method of classification.

A simple explanation of the action of a centrifugal pump is given in Merriam's "Treatise on Hydraulics," reproduced below in condensed form. Figure 34 shows the arrangement of a centrifugal pump. Power is supplied to the shaft *A* from some outside source to rotate the impeller *B* in the casing *C*. The blades of the impeller in revolving produce a partial vacuum and hence take liquid from the suction pipe *D*. This liquid, entering at the axis of the impeller, is forced outward along the blades, leaving the blade tips with a velocity head which changes to pressure head as the liquid passes around the volute chamber toward the discharge *E*.

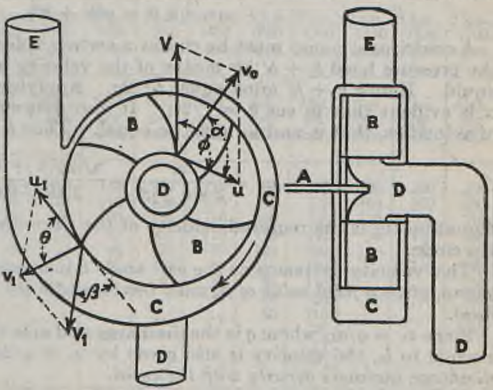


Fig. 34.—Diagram showing action of a centrifugal pump.

The velocities of rotation of the inner and outer circumferences of the impeller-blade tip circles are u and u_1, the absolute velocities of the liquid as it enters and leaves the impeller are v_0 and v_1, and the corresponding relative velocities of the liquid are V and V_1. The angles of entrance, approach, and exit are ϕ, α, and β, while θ denotes the angle between v_1 and u_1. Let H_0 be the pressure head at the point of entrance to the casing (suction) and H_1 at the point of discharge from the casing, while h_0 and h_1 are the heights of the suction and discharge lifts, estimated downward and upward from the center of the impeller. Let h_a be the height of the water barometer. Then

$$V^2 - u^2 - V_1^2 + u_1^2 = 2g(H_1 - H_0) \tag{1}$$

and, neglecting friction,

$$H_1 = h_a + h_1 - \frac{v_1^2}{2g} \tag{2}$$

and

$$H_0 = h_a - h_0 - \frac{v_0^2}{2g} \tag{3}$$

Combining these equations and replacing $(h_1 + h_0)$ by h (the total lift), the fundamental equation for centrifugal pumps, neglecting friction, results. To account for friction, replace h by $(h + h')$, where h' is the total head lost in all hydraulic resistances. Then

$$V^2 - V_1^2 - u^2 + u_1^2 + v_1^2 - v_0^2 = 2g(h + h') \tag{4}$$

which is the fundamental equation for the centrifugal pump.

Since liquid enters the blades radially, $\alpha = 90$ deg., and therefore $V^2 = u^2 + v_0^2$. Also, the parallelogram of velocities at exit indicates that $V_1^2 = u_1^2 + v_1^2 - 2u_1v_1 \cos \theta$. When these values of V^2 and V_1^2 are substituted in Eq. (4), we obtain

$$u_1v_1 \cos \theta = g(h + h') \tag{5}$$

A centrifugal pump must be run at a certain velocity in order to overcome the pressure head $h + h'$ by means of the velocity head, $v_1^2/2g$ of the issuing liquid. Hence $h + h'$ must equal $v_1^2/2g$. Applying this relation to Eq. (5), it is evident that $u_1 \cos \theta = 1/2v_1$. It also follows, from the parallelogram of velocities, that v_1 and u_1 must be equal. Then $\theta = 90$ deg. $- \frac{1}{2}\beta$ and

$$u_1 = \frac{v_1}{2 \sin \frac{1}{2}\beta} = \frac{\sqrt{2g(h + h')}}{2 \sin \frac{1}{2}\beta} \tag{6}$$

Equation (6) is the required velocity of the circumference of the outer blade-tip circle.

This velocity decreases as the exit angle β increases. *Hence the speed of the pump, given a fixed value of β, must increase with the square root of the pressure head.*

Since $v_1 = q/a_1$, where q is the discharge and a_1 is the area of the exit orifices normal to v_1, the velocity is also given by $u_1 = q/2a_1 \sin \frac{1}{2}\beta$. *Therefore, the discharge increases directly with the speed.*

Since the speed must increase with the lift, and the losses of head increase with the speed, the efficiency of a centrifugal pump in general decreases with the lift.

The theoretical horsepower of a centrifugal pump necessary to raise a given quantity Q, in cubic feet per second, of a liquid of density d lb. per cu. ft. through a head H_1 in feet, is expressed by the equation

$$\text{Horsepower} = \frac{QdH_1}{550} \tag{7}$$

Losses due to bearing friction, stuffing boxes, rotation loss, and short-circuiting of liquid through the pump increase the actual power requirement of the pump above the theoretical, so that the **shaft efficiency** of a well-designed pump is between 70 and 80 per cent. Some large pumps have shaft efficiencies close to 90 per cent.

The quantity discharged by a centrifugal pump is only directly proportional to the speed when there is no static head to be overcome. To regulate for constant quantity while maintaining constant speed, it is necessary to provide for constant velocity of flow in the discharge pipe or in the suction pipe, which can be done by means of a control valve actuated by changes in velocity.

It can be stated that, among single-stage pumps, the double-suction types are the usual choice because of their inherent hydraulic balance. This statement must be qualified by further stating that, for general work at low heads and also in the case of many special pumps for the chemical industry, single suction is used and the thrust is either entirely taken up in a thrust bearing, or a combination of a partial hydraulic balance obtained by special construction and a thrust bearing serves this purpose.

In multistage pumps, because of the difficulty of designing a double suction for each stage, partial hydraulic balance is obtained by varying the amount of shrouding on either side of the blades, by perforating the shrouding and providing pressure chambers on either side of the blade, or by similar means.

Table 51. Motor-driven, Single-suction Centrifugal Pumps

Worthington Pump & Machinery Corporation

Head, ft.	1 in.		1½ in.							2½ in.		
	½ h.p., 1450 r.p.m.	1 h.p., 1700 r.p.m.	1 h.p., 1450 r.p.m.	2 h.p., 1150 r.p.m.	2 h.p., 1450 r.p.m.	2 h.p., 1700 r.p.m.	3 h.p., 1450 r.p.m.	3 h.p., 1700 r.p.m.	5 h.p., 1700 r.p.m.	2 h.p., 1150 r.p.m.	3 h.p., 1450 r.p.m.	5 h.p., 1700 r.p.m.
	Gal. per min.	Gal. per min.	Gal. per min.	Gal. per min.	Gal. per min.	Gal. per min.	Gal. per min.	Gal. per min.	Gal. per min.	Gal. per min.	Gal. per min.	Gal per min.
10	35	40	100	110	110	120	110	130	130	180	200	250
15	30	40	90	100	110	120	110	130	130	160	200	250
20	25	40	70	80	100	110	110	130	130	120	200	250
25	10	30	35	50	85	100	110	130	130	50	190	250
30	..	25	20	30	60	80	100	120	130	...	160	250
35	..	20	...	...	35	60	90	110	130	...	110	240
40	..	10	...	...	...	40	70	90	130	...	...	220
45	..	..	...	...	...	...	40	70	120	...	...	190
50	..	..	...	...	...	...	...	50	110	...	...	150
55	..	..	...	...	...	...	...	...	90	...	...	90
60	..	..	...	...	...	...	...	...	75			
70	..	..	...	...	...	...	...	...	40			

List-price range for these pumps is from $25 to $100 (without motor).

Table 52. Belt-driven, Single-suction Centrifugal Pumps

Worthington Pump & Machinery Corporation

Head, ft.	1 in.			2 in.			3 in.			4 in.			8 in.		
	Gal. per min.	H.p.	R.p.m.	Gal. per min.	H.p.	R.p.m.	Gal. per min.	H.p.	R.p.m.	Gal. per min.	H.p.	R.p.m.	Gal. per min.	H.p.	R.p.m.
10	10	0.50	1100	65	1.00	625	125	1.50	500	224	2.00	428	780	5.50	270
	15	0.50	1200	80	1.00	660	165	1.75	535	304	2.25	460	1025	6.75	290
	20	0.50	1300	95	1.00	695	191	2.00	555	332	2.50	475	1200	7.50	302
	25	0.50	1390												
20	10	0.75	1510	90	1.50	885	176	3.00	710	316	4.00	605	1100	12.00	380
	17	0.75	1570	115	1.50	935	232	3.25	760	430	5.25	650	1450	14.50	410
	25	0.75	1690	135	1.75	980	270	3.75	780	470	5.75	670	1690	16.50	430
	35	1.0	1900												
30	10	0.75	1780	110	2.00	1090	216	4.50	865	388	6.75	740	1350	19.50	466
	17	1.0	1830	140	2.75	1150	285	5.25	930	528	9.00	795	1775	24.00	500
	25	1.2	1950	160	3.00	1200	330	6.25	960	575	9.50	820	2070	28.00	524
	35	1.3	2250												

These pumps are also supplied in 1½-, 2½-, 5-, and 6-in. sizes. List prices for these pumps range from $25 to $250.

Head, quantity, speed, horsepower and list price for typical centrifugal pumps of standard types are given in Tables 51, 52, and 53.

Special Centrifugal Pumps. Those considered here include sump pumps; propeller-type pumps, including centrifugal deep-well pumps; and special chemical pumps.

Sump pumps are used for draining relatively shallow pits or sumps and generally must handle dirty liquids. A typical sump pump embodies a standard single-stage, single-suction volute pump with open or unshrouded impeller. This pump is driven from above the pit by a direct-connected motor or belt. A shaft extends down to the pump in the pit from this drive, encased in a pipe which serves to prevent contact of the shaft with the liquid in the pit and to provide a means of support for the shaft bearings.

Table 53. Standard Horizontal Ball-bearing Centrifugal Pumps
Worthington Pump & Machinery Corporation

Pump size and type	Discharge, In.	Suction size, In.	60-cycle, syn., r.p.m.	Range of capacities, gal. per min.	Range of dynamic heads, ft.
1½ R-1	1½	2	3600	50–130	50–180
1½ R-1	1½	2	1800	40–70	25–45
1½ U-1	1½	2	3600	45–110	160–325
1½ U-1	1½	2	1800	25–60	35–75
2 R-1	2	2½	3600	120–215	155–215
2 R-1	2	2½	1800	55–110	40–55
2 U-1	2	3	3600	130–240	180–480
2 U-1	2	3	1800	50–210	80–120
2½ R-1	2½	3	3600	170–310	100–160
2½ R-1	2½	3	1800	80–165	25–40
2½ R-2	2½	3	3600	260–360	155–390
2½ R-2	2½	3	1800	125–200	40–100
3 L-1	3	4	3600	350–600	155–265
3 L-1	3	4	1800	200–330	40–75
3 L-1	3	4	1200	125–285	25–35
3 L-2	3	5	1800	330–600	85–135
3 L-2	3	5	1200	210–400	35–60
3 L-3	3	4	1800	220–460	100–145
3 L-3	3	4	1200	150–300	40–65
4 L-1	4	5	3600	610–870	110–140
4 L-1	4	5	1800	350–700	40–75
4 L-1	4	5	1200	250–500	25–35
4 L-2	4	6	1800	400–800	85–125
4 L-2	4	6	1200	265–520	35–55
4 L-3	4	6	1800	500–750	125–185
4 L-3	4	6	1200	310–490	55–80

List-price range, 1½- to 8-in. sizes, bed plate and motor coupling included, $225 to $1000. Type R pumps are single suction, single stage. Type U pumps are single suction, two stage. Type L pumps are double suction, single stage.

The propeller pump, called by some manufacturers the "screw pump," a name here reserved for the Archimedes screw, is primarily used for handling large quantities at very low heads, work such as is encountered in drainage and irrigation.

Deep-well pumps are vertical-axis centrifugal pumps. The impellers of these pumps are of two types: one, a vaned type similar to that of many simple types of standard volute pumps; and the other of the propeller type. In deep-well pumps the motor may be mounted above the well or may be submerged in the well with the pump, or drive may be by belt, steam turbine, or engine. The pumping unit, and with one design the motor also, are submerged in the water in the well, from which point a discharge pipe extends to the surface, acting also as a support for the pump. When the motor or other means of driving the pump is at the surface, the pump is driven by a shaft, encased in a pipe, and extending downward through the discharge pipe.

A typical range of capacities is

Size, Pump and Well Casing Inside Diameter, In.	Gallons per Minute, Minimum to Maximum
6	100–300
8	200–600
10	450–1200
12	800–2000
14	1200–3000
16	1800–4000
18	3000–6000

Centrifugal chemical pumps are available in a great variety of sizes and designs. The materials used range through a wide selection of metals to hard rubber, glass, and acid-proof stoneware.

Hard-rubber pumps, with rubber impeller and casing of metal lined and covered with rubber, are recommended by the makers for hydrochloric acid, acetic acid, hydrofluoric acid, alkalies, and corrosive salts of all concentrations, nitric acid up to 22 per cent, sulfuric acid up to 62 per cent, hydrogen peroxide, and chlorine liquors. One well-known design of hard-rubber pump, with an impeller diameter of 13 in., has capacities up to 200 gal. per min.

Glass pumps are made with all parts that come into contact with the liquid being pumped constructed of Pyrex glass. This glass can be provided for use with temperatures up to 200°F. These pumps can be used in the handling of many acid products and other liquids that do not attack glass, and they are particularly suitable for use with food products and other liquids when metallic contamination must be avoided. Glass pumps are designed for easy and thorough cleaning.

Chemical stoneware pumps, used for pumping acids, are usually simple in design, made for low heads, and have relatively large capacities.

Special metal pumps for chemicals are available in a wide variety of metals such as lead lined, lead-antimony alloy, nickel, bronze, stainless steel, aluminum alloy, chrome alloy, copper, tin-lined, and various proprietary alloys. These pumps are available over the entire range of heads and capacities ordinarily encountered in chemical manufacture and are also designed to meet conditions of high temperature and high pressure found in some modern manufacturing processes.

Centrifugal-pump Data and Costs

Selecting chemical pumps cannot be properly done from catalogue data, nor can ranges of prices, horsepower, etc., be given in any handbook. The only method of selecting such pumps that assures the procurement of a suitable unit is to assemble complete data as to the conditions under which the pump must operate and then purchase, upon guarantee, after consultation with several manufacturers of this type of equipment.

Reciprocating Pumps

There are two general classes of piston or reciprocating pumps, **steam pumps** and **power pumps.** In general the action of the liquid-transferring parts of these pumps is the same, a cylindrical piston, plunger, or bucket being caused to pass back and forth in a cylinder. The device is equipped with valves for inlet and discharge of the liquid being pumped, and the operation of these valves is related in a definite manner to the motions of the piston.

The water end of a steam-driven double-acting pump is shown in Fig. 35. A is the plunger actuated by means of the shaft B, which passes out through one end of the cylinder C, through a stuffing box which prevents leakage. The inlet or suction valves are shown at D and the discharge valves at E.

In considering the operation of a reciprocating pump, several efficiencies must be taken into account. These are volumetric efficiency, hydraulic efficiency, indicated efficiency, and mechanical efficiency.

Volumetric efficiency is the relation of the water actually pumped to that which theoretically should be moved on a basis of the piston displacement. It indicates the percentage loss and, when stated as $1 - \text{eff.}_{\text{vol.}}$, is called the **slip.** In good practice slip should not be over 5 per cent, while in new pumps, or those kept in good condition, it will be as low as 1 per cent.

Hydraulic efficiency is the ratio of the actual head pumped to the theoretical head and is expressed by the equation

$$\text{Eff.}_{\text{hyd.}} = \frac{H}{H + \text{hydraulic losses}}.$$

The hydraulic losses are the losses in head in the suction and discharge lines. In the suction line, these consist of (*a*) velocity head, (*b*) entrance head, (*c*) friction in suction pipe, (*d*) losses in bends, and (*e*) losses in suction valves. The loss in the discharge line consists of (*f*) loss in discharge valves, (*g*) velocity head, and (*h*) friction in discharge pipe.

Indicated efficiency is the relation of the horsepower required to move the water actually pumped against the total head to the horsepower calculated from the indicator card of the water end.

Mechanical efficiency is the relation of the indicated water horsepower of the pump to the indicated steam horsepower. It shows the losses of power transmission incurred in operating the pump. This efficiency varies from around 50 per cent for small pumps up to 90 per cent for the larger sizes.

Total efficiency of a steam-driven reciprocating pump is the product of the volumetric, hydraulic, and mechanical efficiencies.

The suction lift is, theoretically, the atmospheric pressure less the vapor pressure of the water. The pump must develop sufficient suction head to overcome this lift, and in practice a head of 12 ft. is found to be sufficient.

Reciprocating pumps are usually provided with an air chamber, as shown in Fig. 35. This serves to smooth out irregularities in the discharge of the pump and gives a uniform flow. It is always used with single steam pumps and power pumps. With low-pressure direct-acting duplex steam pumps, it is not needed. With duplex pumps its volume should be four times the displacement of one piston per stroke; with all other types, eight times.

Types of Reciprocating Pumps. The ordinary types of reciprocating pumps are four.

1. Simplex, Double Acting. These may be direct acting (*i.e.*, direct connected to a steam cylinder) or power driven (through crank and flywheel from the cross head of a steam engine). Figure 36 is a pump of this type, designed for use at heads up to 200 ft. In this figure, the piston consists of the disks *A* and *B*, with the packing rings *C* between. A bronze liner for the water cylinder is shown at *D*. Suction valves are E_1 and E_2. Discharge valves are F_1 and F_2. In the steam end, a pilot valve *L* is operated by a rod, actuated by the piston rod *M*. This pilot operates the main valve *N* to cover or uncover the steam ports *P*.

With the reciprocating pump, either piston or plungers are used. In Fig. 35 a plunger is shown. It operates past stationary packing, while the piston shown in Fig. 36 carries its own packing.

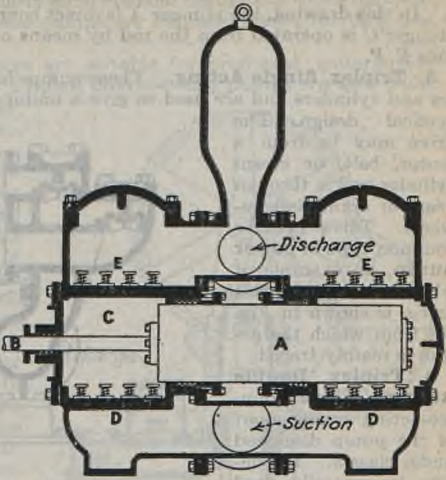


Fig. 35.—Water end of a double-acting steam-driven reciprocating pump.

2. Duplex, Double Acting. These pumps differ primarily from those of the simplex type in having two water cylinders whose operation is coordinated. These pumps may be either direct acting steam driven, or power driven with crank and flywheel.

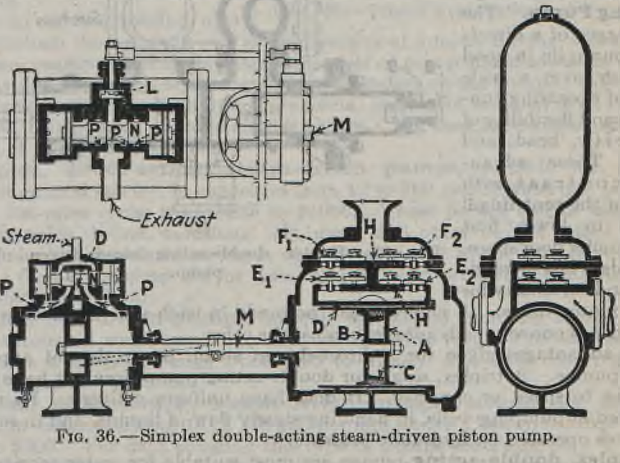


Fig. 36.—Simplex double-acting steam-driven piston pump.

A duplex, outside-end-packed plunger pump with pot valves, of the type used with hydraulic presses and for similar service, is shown in Figs. 37*a* and *b*. In this drawing, the plunger *A* is direct connected to the rod *B*, while the plunger *C* is operated from the rod by means of the yokes *D*, *D* and the tie rods *E*, *F*.

3. Triplex, Single Acting. These pumps have three single-acting plungers and cylinders and are used to give a uniform flow. They are usually of vertical design. The drive may be from a motor, belt, or steam cylinder and is through gears or crank and flywheel. This is the common type of **power pump,** an example of which, arranged for belt drive, is shown in Fig. 38, from which the action is readily traced.

4. Triplex, Double Acting. This is a double-acting arrangement of the pump discussed under class 3. The design is generally used for horizontal triplex pumps. Other features are similar to class 3.

Choice of a Reciprocating Pump. The advantages of a direct-drive pump lie in good efficiency over a wide range of operating conditions and flexibility of capacity, head, and speed. These advantages contrast with those of the centrifugal pumps in lower first cost, smaller floor space, less noise, less attendance, and suitability for direct motor drive. A choice must be made in each individual case by a balancing of one set of advantages against the other.

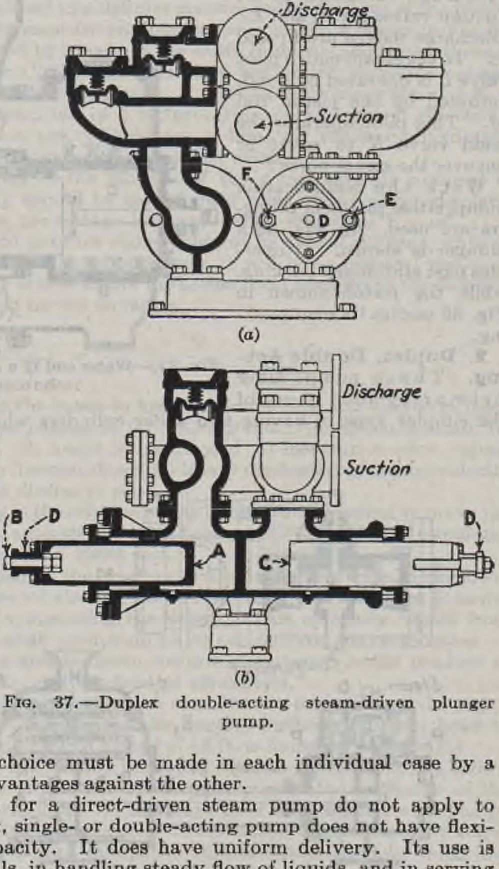


Fig. 37.—Duplex double-acting steam-driven plunger pump.

The advantages given for a direct-driven steam pump do not apply to power pumps. A triplex, single- or double-acting pump does not have flexibility as to speed or capacity. It does have uniform delivery. Its use is indicated in pumping wells, in handling steady flow of liquids, and in serving machines operated by hydraulic pressure.

Simplex, double-acting pumps are most suitable for water service and boiler feed.

Duplex, double-acting pumps are universally adaptable. When made of the proper materials, particularly as to cylinder linings and valve material, and when of correct design, these pumps are frequently used for tars, oils, and other viscous liquids.

Triplex, single-acting pumps are suitable for producing uniform flow when conditions are constant. Triplex, double-acting pumps are used under the same conditions as triplex, single-acting pumps but are generally of horizontal design.

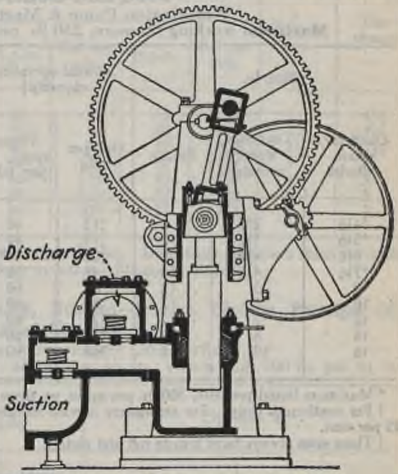


FIG. 38.—Triplex single-acting power pump.

Reciprocating-pump Data and Costs. Reciprocating pumps, particularly of the direct-acting, steam-driven design, are available in a great variety of designs and sizes, to fill every need for which such a pump is suitable. Because of this fact, it is not possible to give within the compass of a handbook full tabular data as to all sizes and styles. In the following data as to costs and in the accompanying tables of size and capacity, the pumps in ordinary service are covered. For information concerning pumps outside the range for which data are given, the chemical engineer should refer to the pump manufacturers.

Chemical engineering operations include the handling of a great diversity of liquids, having a range of properties requiring the use of many materials of pump construction in order to withstand the action of the liquid being handled on the pump. However, in the great majority of pumping problems, the liquid handled is relatively harmless, being water, oil, or of similar nature. For such service the ordinary designs of pumps are suitable.

Simplex, direct-acting, steam-driven pumps, suitable for boiler feed and general service, in capacities from 12 to 360 gal. of water per minute have a list-price range from $125 to $1000. These pumps can be used for handling acids, brines, solutions, oils, dyes, and other liquids (other than water), the liquid ends being made of a metal suited to the service requirements. Capacities given are for water and will vary with the liquid pumped (see Tables 54 and 55).

Duplex, belt-driven pumps, suitable for general pumping service, belt driven through a single-gear reduction, in capacities from 30 to 450 gal. per min., have a list-price range from $300 to $1100. For data covering this type of pump see Table 56.

Duplex, direct-acting, steam-driven pumps, suitable for general service, in capacities from 14 to 170 gal. per min., have a list-price range from $100 to $500. For data covering this type of pumps see Table 57.

Duplex, direct-acting, steam-driven pumps for low-pressure service, in capacities from 35 to 295 gal. per min., have a list-price range from $130 to $550. For data covering this type of pump see Table 58.

Triplex, single-acting pumps, for service up to 200-lb. pressure, or somewhat higher in the belt-driven design with gear reduction, in capacities from 2¾ to 220 gal. per min., have a list-price range from $90 to $1000. These pumps as designed are suitable for general water supply or tank service

Table 54. Simplex, Direct-acting Steam Piston Pumps for Boiler Feed and General Service

Worthington Pump & Machinery Corporation
Maximum working pressure, 250 lb. per sq. in. steam and water ends

Size, in.			Normal operating capacity†		Pipe size, in.			
Diameter, steam cylinder	Diameter, water cylinder	Stroke	Gal. per min.	Piston speed, ft. per min.	Steam	Exhaust	Suction	Discharge
*4½	2¾	6	‡12	40	½	¾	1¼	1
*5½	3¼	7	‡19	44	½	¾	1½	1¼
6½	4⅛	8	‡32	47	¾	1	2½	2
*7½	4½	10	48	58	1	1¼	3	2½
*8	5	12	61	60	1	1¼	3½	3
10	6	12	88	60	1¼	1½	4	3½
12	7	12	120	60	1½	2	5	4
14	8	12	157	60	2	2½	5	4
16	10	18	367	90	2½	3	8	6

* Maximum liquid pressure, 300 lb. per sq. in. on these sizes.

† For continuous duty. For emergency service increase 30 per cent; for boiler-feed service, decrease 35 per cent.

‡ These sizes always have bronze rod and piston.

Table 55. Simplex, Direct-acting Steam Piston Pumps for Low-pressure Service

Worthington Pump & Machinery Corporation
Maximum working pressure: steam end, 250 lb. per sq. in.; liquid end, 8-in. diameter or below, 100 lb. per sq. in.; larger sizes, 75 lb. per sq. in.

Size, in.			Normal working capacity*		Pipe size, in.			
Diameter, steam cylinder	Diameter, water cylinder	Stroke	Gal. per min.	Piston speed, ft. per min.	Steam	Exhaust	Suction	Discharge
†3½	3½	4	17	33	⅜	½	1½	1
†4½	4⅛	6	28	40	½	¾	2½	2
5½	5½	7	54	44	½	¾	3	2½
6	6	12	88	60	¾	1	3½	3
7½	7	10	117	58	1	1¼	4	3
8	8	12	157	60	1	1¼	5	4
10	10	12	245	60	1¼	1½	6	6
10	10	18	367	90	1¼	1½	6	6
10	12	12	352	60	1¼	1½	8	6
10	12	18	528	90	1¼	1½	8	6
12	12	12	352	60	1½	2	8	6
12	12	18	528	90	1½	2	8	6

* For continuous duty. Capacities may vary, dependent on liquid handled.

† These sizes always have bronze rod and piston.

Table 56. Belt-driven Duplex Piston Pump

Worthington Pump & Machinery Corporation

Size, in.		Maximum pressure, lb. per sq. in.	Displacement*					Pipe size, in.	
				Normal		Maximum			
Diameter, piston	Stroke		Gal. per revolution	R.p.m.	Gal. per min.	R.p.m.	Gal. per min.	Suction	Discharge
3	4	150	0.48	65	31	102	49	2	1½
3¾	4	125	0.73	75	54	102	74	2½	1½
4	6	150	1.26	60	75	87	109	2½	1½
5	6	100	2.00	60	120	87	174	3	2
6	6	75	2.88	60	173	87	250	4	3
7	6	100	3.94	60	236	87	342	6	5
8	6	75	5.16	60	309	87	448	6	5

* Liquids which are viscous or hot require speed below normal. Do not use maximum speed for applications where quiet operation is essential, or where suction conditions are difficult.

Table 57. Duplex, Direct-acting, Steam-driven Piston Pumps for General Service

Worthington Pump & Machinery Corporation

Maximum working pressure: liquid end, 250 lb. per sq. in.; steam end, 200 lb. per sq. in.

Size, in.			Normal operating capacity		Pipe size, in.			
Diameter, steam cylinder	Diameter, water cylinder	Stroke	Gal. per min.	Piston speed, ft. per min.	Steam	Exhaust	Suction	Discharge
*3½	2¼	4	14	33	⅜	¾	1½	1¼
*4½	2¾	4	21	33	½	¾	2	1½
5¼	3½	5	37	38	¾	1¼	2½	1½
6	4	6	52	40	1	1¼	3	2
7½	4½	10	97	58	1½	2	4	3
9	5¼	10	131	58	2	2½	4	3
10	6	10	171	58	2	2½	5	4

* These sizes always have bronze piston rods and water pistons.

on hot or cold water and for feeding small boilers. With slight changes these pumps can be adapted to the pumping of oil and other liquids. For data on pumps of this type see Tables 59, 60, and 61.

Rotary Pumps

Rotary pumps differ from centrifugal and reciprocating pumps in that they will deliver a positive quantity of liquid under conditions of varying head or pressure. These pumps will handle any liquid that contains no grit or abrasive material.

This type of pump consists of a stationary casing in which are located one or more rotating members. When one rotating member, or impeller, is used, it is mounted eccentrically on the shaft. The impeller in this type of

Table 58. Duplex, Direct-acting, Steam-driven Piston Pumps for Low-pressure Service

Worthington Pump & Machinery Corporation

Maximum working pressure: liquid end, 125 lb. per sq. in.; steam end, 200 lb. per sq. in.

Size, in.			Normal operating capacity		Pipe size, in.			
Diameter, steam cylinder	Diameter, water cylinder	Stroke	Gal. per min.	Piston speed, ft. per min.	Steam	Exhaust	Suction	Discharge
*4½	3¾	4	35	30	½	¾	2½	1½
5¼	4¾	5	63	34	¾	1¼	3	2
6	5¾	6	97	36	1	1¼	4	3
7½	6	10	154	52	1½	2	5	4
7½	7	10	200	52	1½	2	6	5
9	8½	10	295	52	2	2½	6	5

* This size always has bronze piston rods and water pistons.

Table 59. Triplex, Single-acting Plunger Pumps

Worthington Pump & Machinery Corporation

Belt drive,* with tight pulley and single-gear reduction, suitable for 250 lb. per sq. in. pressure

Size, in.		Displacement					Pipe size, in.	
			Normal		Maximum			
Diameter, plungers	Stroke	Gal. per revolution	R.p.m.	Gal. per min.	R.p.m.	Gal. per min.	Suction	Discharge
†1¼	2	0.032	86	2.75	129	4.12	1	¾
1¾	2½	0.078	75	5.85	120	9.36	1	1

* Can also be driven by pulley on crank shaft, or with double-reduction gears and direct-connected motor.

† This size can be operated safely to 300 lb. per sq. in. pressure on intermittent service.

pump is usually circular in section and is provided with a reciprocating blade or blades or a horizontal abutment. Figure 39 shows the cross section of a pump of this type. While there are many designs, an explanation of that shown in Fig. 39 will serve to indicate how these pumps operate. *A* is a cylindrical impeller mounted eccentrically on a shaft. As it rotates, its wearing surface *B* wipes around the circumference of the casing and the blade *C* moves up and down in the sleeve *F*, which is free to rotate in a socket. Water enters as shown by the arrow. When the impeller is at the top of the stroke, water flows into the casing. As the impeller wipes down, past the suction opening, this water is forced around the casing and is discharged at *D*, through opening *E*, and out through discharge *G*.

When two or more impellers are used in a rotary-pump casing, the impellers will take the form of toothed-gear wheels as in Fig. 40, of helical gears, or of lobed cams. In either case, these impellers rotate with extremely small clearance between each other and between the surface of the impeller and the casing. Referring to Fig. 40, the two toothed impellers rotate as indicated by the arrows. The suction connection is at the bottom. As the spaces between the teeth of the impeller pass the suction opening, water is impounded between them, forced around the casing to the discharge opening, and then forced out through this opening. The arrows indicate this flow of water.

Wherever the pumping problem involves the delivery of a specified quantity of liquid in a definite time, the rotary pump should be considered, although, if the head does not vary, a centrifugal may be found more suitable in certain cases. Advantages of the rotary pump are low first cost, low attendance cost, durability, and simplicity.

Rotary pumps are available in two general classes, **interior bearing** and **external bearing**. The interior-bearing type is used for handling liquids of a lubricating nature, while the exterior-bearing type is used with non-lubricating liquids. The interior-bearing pump is lubricated by a part of the liquid being pumped, while the exterior-bearing type is oil lubricated by an attendant.

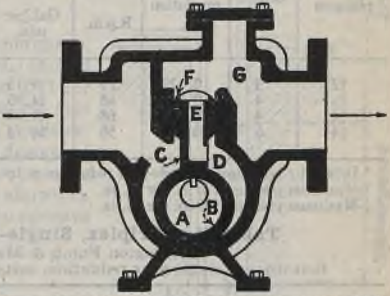


Fig. 39.—Single impeller rotary pump with plunger or reciprocating blade.

Among the liquids handled by rotary pumps are mineral oils, vegetable oils, animal oils, greases, glucose, molasses, paints, varnish, shellac, lacquers, alcohols, catsup, brine, mayonnaise, sizing, soap, tanning liquors, vinegar, and ink. It should be kept in mind that rotary pumps are not suitable for handling liquids carrying grit or abrasive material.

Rotary-pump Data. With rotary pumps of the simple cam or two-lobed impeller design, operating against heads of 10 to 200 ft., the efficiency is said to range from 75 to 85 per cent.

With rotary pumps of the gear type, volumetric efficiencies range from 90 to 95 per cent and over-all efficiencies range up to 70 per cent. This class of pump will operate at pressures up to 350 lb. per sq. in.

Rotary pumps of the gear type, made by a well-known manufacturer, and one of the most common types employed in chemical work, are available in capacities from 4 to about 850 gal. per min. when operating at 60 lb. per sq. in. pressure. These pumps have a list-price range from $75 to $800, without motor.

Fig. 40.—Gear-type rotary pump, having two impellers.

Table 60. Triplex, Single-acting Plunger Pumps*

Worthington Pump & Machinery Corporation

Size, in.		Displacement					Pipe size, in.	
			Normal		Maximum			
Diameter, plungers	Stroke	Gal. per revolution	R.p.m.	Gal. per min.	R.p.m.	Gal. per min.	Suction	Discharge
†2	3	0.122	75	9.15	112	13.6	1¼	1¼
†3	4	0.367	68	24.95	102	37.4	2	1½
‡4	4	0.652	68	44.33	102	66.5	2½	2
‡4	6	0.979	58	56.78	87	85.0	3	2½

* Driven by belt through single-gear reduction or by motor with double-gear reduction.

† Maximum pressure 250 lb. per sq. in.

‡ Maximum pressure 150 lb. per sq. in.

Table 61. Triplex, Single-acting Plunger Pumps

Worthington Pump & Machinery Corporation

Belt-driven, single-gear reduction, suitable for 200 lb. per sq. in. pressure

Size, in.		Displacement			Pipe, in.	
			Maximum			
Diameter, plungers	Stroke	Gal. per revolution	R.p.m.*	Gal. per min.	Suction	Discharge
3½	4	0.50	102	51	3	2½
4	6	0.97	87	84	4	3
5	6	1.53	87	133	4	4
6	8	2.93	75	220	†5	†5

* Liquids that are hot or viscous require lower speeds.

† Has flanged openings.

Handling Liquids with Fluid Pressure

In addition to the liquid-handling devices, such as pumps, that depend on the mechanical action of pistons, plungers, or impellers to move the material, there is a large group of devices for this purpose that employ fluid pressure to move the liquid. This group includes air lifts, acid eggs, jet pumps, pulsometers, and diaphragm pumps among others.

The Air Lift. This is a device for raising liquids by means of compressed air. Its ordinary use is in pumping wells. A simplified sketch of an air lift is shown in Fig. 41. Referring to this sketch the running submergence H_s is the distance from the water level to the point of an air inlet, and the total lift H_t is the distance from the working surface of the water to the point of discharge. An empirical formula to express the volume of free air required to lift 1 gal. water has been developed by the Ingersoll-Rand Company from practice. According to this formula

$$V_a = 0.8 \frac{H_t}{C \log \dfrac{H_s + 34}{34}}$$

where V_a is the volume of free air in cubic feet and C is a constant, values for which, varying with the total, head are

H_t, ft.	C
10–60	245
61–200	233
201–500	216
501–650	185
651–750	156

In designing an air lift, the submergence, as expressed by the ratio $H_s/(H_t + H_s)$ should vary from 0.66 for a lift of 20 ft. to 0.41 for a lift of 500 ft. The results given by this formula approximate those found in practice, but variations occur caused by the design of the foot pieces. Two designs for foot pieces that are employed with air lifts are shown in Fig. 42. The air pressure required to operate an air lift is given by

$$P = B - 0.434s$$

where

P is in lb. per sq. in. abs.

B is the barometric pressure, lb. per sq. in.

s is the submergence, ft.

The efficiency of an air lift, *i.e.*, the ratio of the water horsepower to the indicated air horsepower of the compressor, is about 70 per cent, except at starting, when it is considerably lower. Higher air pressure is also required for starting, when the pressure must be equivalent to the height of the water level, at rest, above the end of the air pipe.

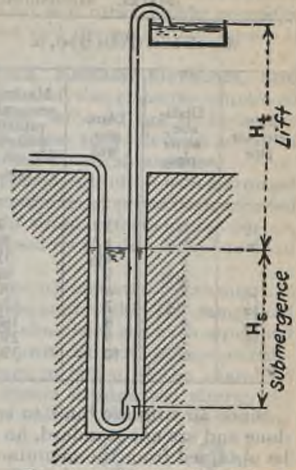


FIG. 41.—Simplified sketch of an air lift, showing submergence and total head.

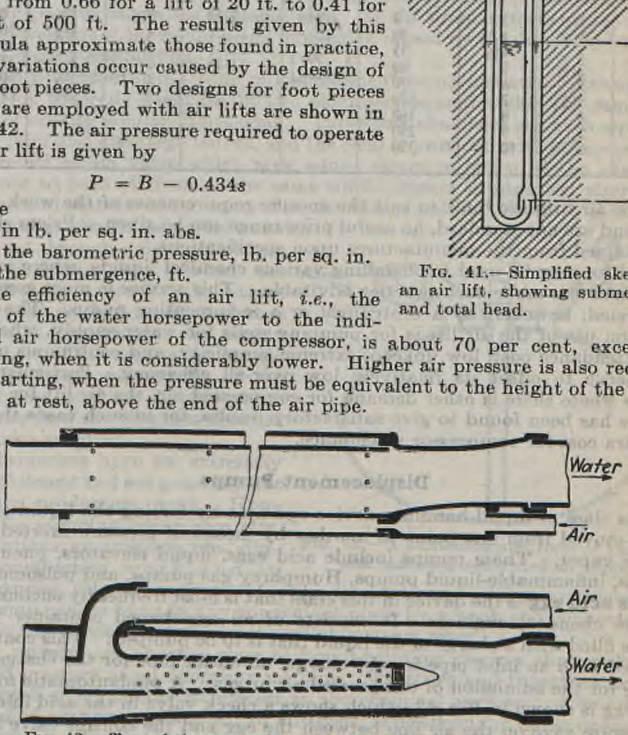


FIG. 42.—Two of the many designs of foot pieces used with air lifts.

Referring to Fig. 42, the foot piece shown at the top is a modification of the Saunders type, in which a central water discharge pipe is surrounded by an air passage. That at the bottom of the figure is a modification of the Pohle type of foot piece, in which air enters the water discharge pipe at the side and the air and water pipes are placed side by side in the well. The accompanying Table 62 shows values for capacities and pipe sizes for air lifts.

Table 62. Pipe Sizes and Capacities of Air Lifts

Marks, "Mechanical Engineers' Handbook," McGraw-Hill

Side air inlet (Pohle type), in.				Concentric air pipe		Central air pipe		
Diameter, air pipe	Diameter, water pipe	Diameter, well	Maximum economical capacity moderate lift, gal. per min.	Lift, ft.	Gal. per min. per sq. in. of water pipe, cross section	Diameter, casing, in.	Diameter, air pipe, in.	Capacity, gal. per min.
½	1	3	7	25	15–20	3½	1¼	80– 100
¾	1½	4	20	...	.....	4	1½	100– 150
1	2	4½	35	50–125	.....	5	2	150– 250
1	2½	5	60		12–15	6	2	275– 375
1¼	3	6	90		.....	8	2½	500– 665
1½	3½	7	120	...	.....	10	2½	775–1000
1½	4	8	160					
1½	5	9	250					
2	6	10	350					

Since air lifts are built to suit the specific requirements of the work to be done and are not standard, no useful price range can be given. Prices should be obtained from the manufacturer upon specifications.

Air lifts may be used for handling various chemical liquids, where special circumstances make such practice advisable. This service is more generally performed, however, by a centrifugal or a reciprocating pump. The most common use of the air lift is for pumping wells for water supply, where its low attendance cost, low upkeep, extreme simplicity, and continuous operation serve to offset its relatively low over-all efficiency. Particularly in plants where there is other demand for compressed air, the air lift for water service has been found to give satisfactory results, for in such cases there is no extra cost for compressor attendance.

Displacement Pumps

This class of liquid-handling device operates by forcing the liquid that is to be moved from one point to another by means of pressure exerted by a gas or vapor. These pumps include acid eggs, liquid elevators, pneumatic pumps, inflammable-liquid pumps, Humphrey gas pumps, and pulsometers.

The acid egg is the device in this class that is most frequently encountered by the chemical engineer. It consists of an egg-shaped container which can be filled with a charge of the liquid that is to be pumped. This container is fitted with an inlet pipe for the charge, an outlet pipe for the charge, and a pipe for the admission of compressed air or gas. A semiautomatic form of acid egg is shown in Fig. 43, which shows a check valve in the acid inlet and a pressure gage on the air line between the egg and the control valve in the air line. This arrangement permits the operator to close the air line when the egg is emptied, for the gage will then show a lower pressure than when the egg is operating and a pressure is registered due to the height of the acid in the discharge pipe.

Acid eggs are also designed to be fully automatic, turning the air and the acid on and off as required. However, all designs of this equipment suffer under the disadvantage of being low in over-all efficiency and intermittent.

With the development of the centrifugal pump in materials of construction suitable to withstand the action of sulfuric acid and other highly corrosive chemicals, the need for the acid egg has passed and the centrifugal pump is usually chosen for such services.

Automatic elevators, pneumatic ejectors, Kestner elevators, and inflammable-liquid pumps are all variations of the principle employed in the acid egg. The same reasons that cause the acid egg to be rarely employed in modern chemical plant practice cause these devices to be of little beyond theoretical interest to the practicing chemical engineer.

The Humphrey gas pump is a type of displacement pump which aroused considerable interest some twenty years ago but which never has been adopted to any great extent in the United States. In the simplest form of this device, the movement of the liquid being pumped is caused by the direct expansion of a gas during combustion.

The pulsometer is a form of displacement pump operated by steam. It consists, generally speaking, of two chambers formed within the same shell and of exactly similar dimensions. These two chambers are each equipped with inlet and discharge valves, and the shell is provided with a flap- or ball-controlled steam valve which may admit steam to either of the chambers but not to both of them at the same time. Steam is admitted alternately to these chambers under control of the steam valve. When it enters one of the chambers, it forces the water in it out through a check valve into the discharge pipe. When the chamber is emptied, the steam enters the discharge pipe and thus causes a suction in the empty chamber. This results in reversing the steam valve and, at the same time, sucking a fresh charge of water into the chamber that has just been emptied.

Pulsometers have an extremely low efficiency and are generally not used in production work. However, for pumping water from pits, foundations, excavations, etc., during construction work and in emergencies, they are very useful, as no foundation or setting is required and the pump can be hung in any convenient place by a hook and connected by flexible steam hose to the nearest available steam main.

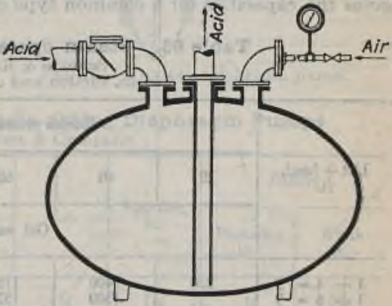


FIG. 43.—A semiautomatic form of acid egg, with manually controlled air supply.

Jet pumps are another class of liquid-handling devices that make use of fluid pressure as an operating medium. **Ejectors** and **injectors** are the two types of jet pumps of interest to the chemical engineer. The ejector, also called **siphon, exhauster**, or **eductor**, is designed for use in operations where the head pumped against is low and is less than the head of the fluid used for pumping. The **injector** is a special type of jet pump, operated by steam and used for boiler-feed and similar services, in which the fluid being pumped is discharged into a space under the same pressure as that of the steam which is used to operate the injector.

Figure 44 shows a simple design of jet pump of the **ejector** type. The pumping fluid enters through the nozzle at the left, passes through the venturi nozzle at the center and out the discharge opening at the right. As it passes into the venturi nozzle, it develops a suction which causes some of the fluid in the suction chamber to be taken into the venturi nozzle and, entraining with the steam passing through the discharge, be delivered through this discharge.

The efficiency of an ejector or jet pump is low, being around 1 or 2 per cent. The head developed by the ejector is also low, except in special types. The device has the disadvantage of diluting the fluid pumped by mixing it with the pumping fluid. In steam injectors for boiler-feed and similar services, where the heat of the steam is recovered, the efficiency is close to 100 per cent. Types of jet pumps have also been developed for use as vacuum pumps, in which service high efficiency is obtained.

The simple ejector or siphon is widely used, in spite of its low efficiency, for transferring liquids from one tank to another, for lifting acids, alkalies, or solid-containing liquids of an abrasive nature, and for emptying sumps. Table 63 gives the capacities for a common type of steam-operated water ejector.

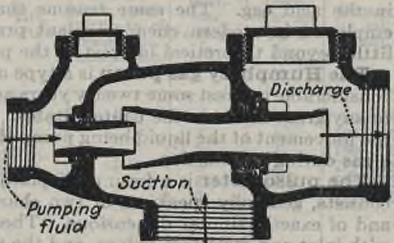


FIG. 44.—Simple type of ejector for water service.

Table 63. Steam-operated Water Ejectors

Schutte & Koerting

1¼-in. suction and discharge pipes

Lift + head, ft.	Steam pressure, lb. per sq. in., gage					
	20	40	60	100	140	200
	Gal. water pumped per hr.					
1 + 4 = 5	220	1400	1150			
1 + 8 = 9	...	1300	1520	1300	1150	900
1 + 30 = 31	...	....	1450	1300	1150	
1 + 70 = 71	....	....	....	1300		
8 + 0 = 8	700	1430	1250	1060	920	780
16 + 0 = 16	...	1000	920	740	600	
19 + 0 = 19	...	800	680	540	450	

Miscellaneous Liquid-handling Devices

In addition to the forms of liquid-handling equipment already described, which include all the most used forms for chemical engineering work, there are many other types in existence, including diaphragm pumps, hydraulic rams, hydrautomats, screw or spiral pumps, spiral circulators, scoop wheels, chain elevators, and many others.

Of these, the **diaphragm pump** is the only device of much present importance to the chemical engineer. A simple diaphragm pump of the vertical type is shown in Fig. 45. In this pump, *A* is a flexible diaphragm, bolted to the casing as shown. This diaphragm is movable, up and down, through the action of the yoke and rod shown. *B* is a rubber discharge valve and *C* is the inlet valve. The action of the pump is obvious from inspection of the figure.

Diaphragm pumps of this simple type are used for emptying tanks and sumps, pumping out excavations, and similar services.

Diaphragm pumps for handling chemical liquids, particularly those of a corrosive or abrasive character, are designed so that the liquid being pumped does not come in contact with the moving metallic parts of the pump. In such pumps, a plunger works back and forth into a fluid-filled chamber, one side of which is formed by the diaphragm. This causes the diaphragm to move back and forth, thus actuating the liquid to be pumped, which is in a chamber on the other side of the diaphragm. Table 64 gives capacities and other data for four sizes of this latter type of diaphragm pump.

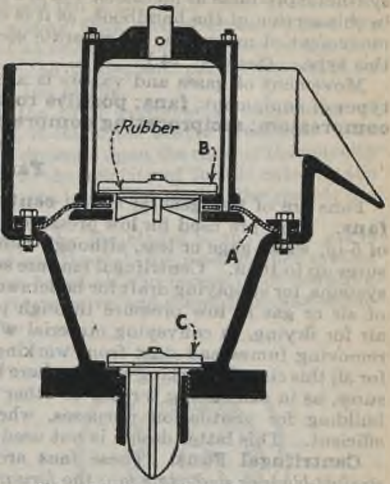


Fig. 45.—Vertical-diaphragm pump.

Table 64. Capacities of Single-acting Diaphragm Pumps
F. W. Brackett & Company

Gal. per hr.	Diameter of plunger, in.	Stroke of plunger, in.	R.p.m.	H.p.	Pipe size, in.	Pulleys	
						Diameter in.	Width in.
280	2½	3½	90	1¼	1¼	16	2¼
450	3	4½	80	1¾	1¾	21	2½
620	3½	5	70	2¼	2¼	24	3½
830	4	6	60	2½	2½	30	4½

MOVEMENT OF GASES AND VAPORS

The movement of gases and vapors is basic to such chemical engineering industries as fuel-gas manufacture, by-product coking, and petroleum refining. The manufacture of many other chemical products involves the movement of gases and vapors, which may be the materials processed, or may be used in the processing of other materials, as air for agitation, gas for bleaching, ammonia or other refrigerants for cooling, gas for fuel, and many other applications.

These gases may be raw materials, as in the case of natural gas or air; they may be materials undergoing processing, as in the case of unstripped coke-oven gas; or they may be finished products such as manufactured gas, carbon dioxide, or acetylene.

The compression of gas which occurs as part of the manufacture of such synthetic products as methanol and synthetic ammonia will not be considered in this section of the handbook, as it is essentially a step in processing, not a movement of materials. A separate section covers high-pressure technic of this type. (See Sec. 14.)

Movement of gases and vapors is accomplished by means of five general types of equipment: **fans; positive rotary blowers; centrifugal or turbo-compressors; reciprocating compressors;** and **jets.**

Fans

Fans are of two general types: **centrifugal fans** and **disk or propeller fans.** Fans are used for low pressures, generally speaking for pressure heads of 5-in. water gage or less, although some heavy-duty types operate at pressures up to 10 in. Centrifugal fans are suitable for ventilating work with duct systems, for supplying draft for boilers and furnaces, for moving large volumes of air or gas at low pressure through piping or duct systems, in supplying air for drying, in conveying material while suspended in a stream of air, in removing fumes and dust from working spaces. Centrifugal fans are used for all this class of working except where large volumes are handled at low pressures, as in exhausting a room or other open space, or in blowing air into a building for ventilation purposes, when propeller or disk fans are more efficient. This latter design is not used with piping or ducts.

Centrifugal Fans. These fans are made in three general styles: the *straight-blade or steel-plate fan;* the *forward-curved-blade fan;* and the *backward-curved-blade fan.*

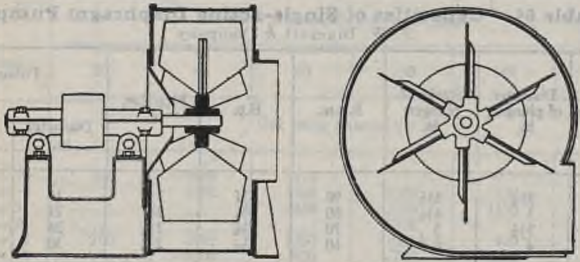


Fig. 46.—Straight-blade or steel-plate fan used for exhausting and draft.

The *forward-curved-blade fan* finds its greatest use where large volumes of air are moved at slow rotative speeds. In ordinary industrial work it is not now as frequently encountered as formerly, although still widely used. The *straight-blade* or *steel-plate fan* (Fig. 46) is now generally built in the type called the **mill exhauster.** This fan has relatively few blades, from six to ten as a general rule. These blades are relatively long and narrow. The design is most used for exhaust work and handling industrial wastes such as shavings,

Many of the centrifugal fans that are now designed for ventilating, draft production, gas handling, and general industrial work are of the *backward-curved-blade* type, as, when properly designed, such fans are found to have the best characteristics over the widest range for general purposes. These fans as now designed tend to have a relatively large number of relatively shallow, wide blades. They are often equipped with inlet vanes or guides which improve the working characteristics and decrease the sound of operation.

The theory of operation of a centrifugal fan is much like that of a centrifugal pump, the pressure developed arising from two sources. These are centrifugal force due to the rotation of an enclosed volume of air or gas, and velocity imparted to the air or gas by the blades and partly converted to pressure by the volute or scroll-shaped fan casing.

The centrifugal force developed by the rotor produces a compression of the air or gas which, in fan engineering, is called the **static pressure.** The amount of this static pressure developed depends upon the ratio of the velocity of the air leaving the tips of the blades to the velocity of the air entering the fan at the heel of the blades. Therefore, the longer the blades, the greater the static pressure developed by the fan.

The kinetic energy or velocity imparted by the blades to the air or gas is more than sufficient to carry the air or gas through the pipe or duct system served by the fan. Casing designs are based on the necessity of reducing this velocity, changing some of it to static pressure, and thereby increasing the pressure of the air delivered by the fan.

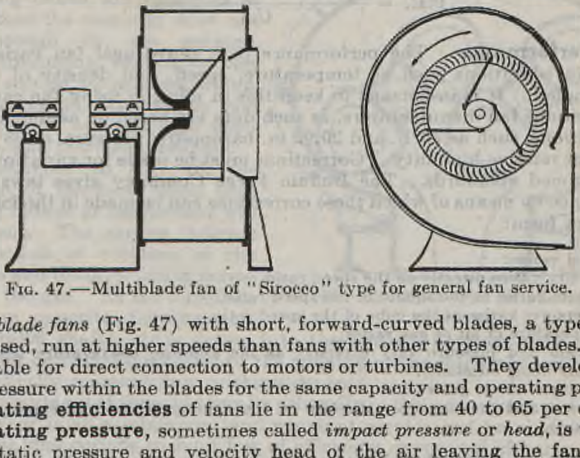


FIG. 47.—Multiblade fan of "Sirocco" type for general fan service.

Multiblade fans (Fig. 47) with short, forward-curved blades, a type that is widely used, run at higher speeds than fans with other types of blades. They are suitable for direct connection to motors or turbines. They develop more static pressure within the blades for the same capacity and operating pressure.

Operating efficiencies of fans lie in the range from 40 to 65 per cent.

Operating pressure, sometimes called *impact pressure* or *head*, is the sum of the static pressure and velocity head of the air leaving the fan. It is generally expressed in inches of water gage or in ounces per square inch.

1 oz. per sq. in. = 1.732 in. water with air at standard density (0.075 lb. per cu. ft.).

Head of 1 in. water = head of 69.3 ft. air.

The **total pressure** for a straight-bladed fan is given by

$$P = \frac{(U_2^2 - U_1^2) + MU_2^2 - (1 - M)V_2^2 - (NV_0)^2}{V_p^2}$$

where V_0 = velocity of air through inlet.
V_p = velocity corresponding to unit pressure.
V_2 = radial velocity of air leaving blade tips.
U_1 = linear velocity at heel of blades.

U_2 = linear velocity at tip of blades.
M = percentage velocity pressure conversion in scroll.
N = ratio of actual to effective area of inlet.

The **air horsepower** of a fan is given by

$$\text{Air horsepower} = \frac{\text{cubic feet per minute} \times \text{total pressure in pounds}}{33{,}000}$$

$$= 0.000157 \times \text{cubic feet per minute} \times \text{total head in inches}$$

The **mechanical efficiency** of a fan is given by

$$\text{M.E.} = \frac{\text{air horsepower}}{\text{brake-horsepower}}$$

Pressure efficiency of a centrifugal fan is the pressure developed by the fan divided by the pressure against a plane surface due to a velocity equal to the peripheral speed of the wheel, or

$$\text{P.E.} = \frac{g \times \text{head of air in feet}}{U_2}$$

Fan Performance. The performance of a centrifugal fan varies with changes in conditions such as temperature, speed, and density of the gas being handled. It is important to keep this in mind in using the catalogue data of various fan manufacturers, as such data are based on assumed standard conditions, such as 70°F. and 29.92 in. barometric pressure, or 68°F. and 50 per cent relative humidity. Corrections must be made for variations from these assumed standards. The Buffalo Forge Company gives laws of fan performance by means of which these corrections can be made in the following convenient form:

When speed varies:
1. Capacity varies directly as the speed ratio.
2. Pressure varies as the square of the speed ratio.
3. Horsepower varies as the cube of the speed ratio.

When temperature of air or gas varies:
Horsepower and pressure vary inversely as the absolute temperature (speed and capacity being constant).

When density of air or gas varies:
Horsepower and pressure vary directly as the density (speed and capacity being constant).

Selection of Centrifugal Fans. It is a common practice among fan manufacturers to publish complete data in tabular form showing capacities, pressures, speeds, and horsepowers of their fans under standard conditions of temperature and air density. These tables are of great use to the heating and ventilating engineer and to others who specialize in fan engineering. Those who do not specialize along these lines, including the chemical engineer, should not attempt to select fans from these tables. The proper course to follow is to put full data concerning the job to be done in the hands of fan manufacturers and allow them to specify the fan which they are willing to guarantee to do the required work at the best obtainable economy. A comparison of several such proposals from manufacturers will indicate the best choice.

Disk or propeller fans should be chosen by the chemical engineer in the same manner as has been indicated for centrifugal fans.

Special fans for chemical service can be obtained from various fan manufacturers on specification to handle practically any gas or vapor. Such fans are, in the main, similar to standard fans but are either constructed of, or have the working parts covered with, a material that will resist the corrosive or abrasive action of the material being handled. Such fans are not standard, and each such case should be placed fully before several manufacturers of fans before a choice is made.

Positive Rotary Blowers

Positive rotary blowers and compressors are similar in action and design to rotary pumps. In fact, any cycloidal, gear, or other rotary pump can be used as a blower and, if properly designed, as a compressor.

These blowers have a pressure range from ½ to 15 lb. per sq. in. in single stages and an equivalent vacuum range when used as exhausters.

Rotary blowers and exhausters find among their important applications handling gases, drying, supplying blast for furnaces, forges, and ovens, aerating and agitating, pneumatic conveying (when the blower is so placed that the material does not pass through it), ore refining, sewage treatment, ventilation, low vacuum service, by-product coking, gas boosting in distribution and transportation systems, gas compression, and petroleum refining.

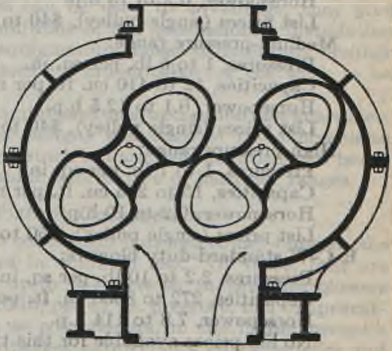


FIG. 48.—Cycloidal type of positive rotary blower.

By reference to Fig. 48, the operation of the most usual type of rotary blower or compressor will be evident. The arrows indicate the direction of rotation of the impellers and the direction of movement of the gas. As the impellers revolve in opposite directions on their parallel shafts, air is drawn into the pockets between them and the casing and delivered positively to the opening of the discharge. The result is a practically constant flow, without surges.

The advantages of these blowers lie primarily in their simplicity and ruggedness. Wear is at a minimum, and constant service can be obtained from them over long periods without loss of efficiency or renewal of parts.

Within the ranges for which different types of rotary blowers are built, their efficiencies are relatively high and their power requirements low. Generally speaking, at pressures from 8 oz. per sq. in. up, rotary blowers are more efficient than fans. Also the blower has an advantage in that it will handle a constant volume under varying pressure conditions, which a fan will not do. At pressures below 10 lb. per sq. in., the rotary blower is more efficient than the reciprocating compressor, but above that point the compressor has the advantage. The efficiency of the rotary blower is best at 3 or 4 lb. pressure or vacuum, when it lies in the range of from 84 to 86 per cent in the case of large machines. For small machines the efficiency is less, as the leakage past the impellers is relatively greater.

Horsepower for rotary blowers varies with volume and pressure, being roughly about 5 h.p. per 1000 cu. ft. free air delivered per minute at 1 lb. per sq. in. pressure.

Positive rotary blowers are available in many types. In general, the range of working pressures or suctions for which they are designed runs from 1 lb. up to 15 lb. per sq. in. The capacity range will run from 10 cu. ft. up to about 10,000 cu. ft. free air per minute in standard sizes, while special machines will have capacities up to 25,000 or 30,000 cu. ft. free air per minute. Compound, two-stage units can be obtained in capacities up to 50,000 cu. ft. per min. and for pressures up to 30 lb. per sq. in.

In the most common type, having two two-lobed impellers (of which the Roots-Connersville-Wilbraham is a well-known example), the following ranges are available:

- Victor-Acme blowers for general service:
  - Low-pressure range:
    - Pressures, 1 to 3 lb. per sq. in.
    - Capacities, 15 to 700 cu. ft. per min.
    - Horsepower, 0.2 to 13 h.p.
    - List prices (single pulley), $40 to $440.
  - Medium-pressure range:
    - Pressures, 1 to 5 lb. per sq. in.
    - Capacities, 12 to 410 cu. ft. per min.
    - Horsepower, 0.1 to 12.5 h.p.
    - List prices (single pulley), $40 to $425
  - High-pressure range:
    - Pressures, 1 to 7 lb. per sq. in.
    - Capacities, 15 to 245 cu. ft. per min.
    - Horsepower, 0.2 to 10 h.p.
    - List prices (single pulley), $50 to $420.
- R-C-W standard-duty blowers:
  - Pressures, 2.2 to 10 lb. per sq. in.
  - Capacities, 272 to 8050 cu. ft. per min.
  - Horsepower, 7.8 to 114 h.p.
  - No list prices available for this type.
- R-C-W heavy-duty blowers:
  - Pressures, 3 to 10 lb. per sq. in.
  - Capacities, 295 to 7920 cu. ft. per min.
  - Horsepower, 12.2 to 152 h.p.
  - No list prices available for this type.

Centrifugal (Turbo) Compressors

Centrifugal compressors are similar to centrifugal pumps in design, except that proportions and strength of parts are varied to suit the requirements of handling gases rather than liquids. The theory of operation is in general the same for the two machines, except that the compressor is handling a readily compressible fluid, whereas the pump handles fluids which are but slightly compressible. Centrifugal fans (see pp. 2266 to 2269) are in reality but a special case of the centrifugal compressor, designed to operate at good efficiency and economy at low pressures. The machines usually known as centrifugal compressors are high-speed machines operating in the pressure range from 1 to 5 lb. per sq. in. when built in single stages and having capacities up to 12,000 cu. ft. per min. Multistage types of these compressors are built for pressures up to 175 lb. per sq. in. The usual range for the multistage machines is up to 30 lb. per sq. in. and 50,000 cu. ft. per min.

Centrifugal compressors have the advantage of being suitable for direct connection to electric motors or steam turbines. The space requirements in such cases are relatively small. They maintain approximately constant pressure when operated over a broad range of capacity, if the speed is maintained constant. With intermittent delivery, the flow may be shut off without stopping the compressor or wasting gas.

Centrifugal compressors find their chief uses in the chemical engineering industries in manufactured-gas plants, at by-product coke works, and at blast-furnace and steel plants. In these industries they are used for blast work and gas handling. Conditions of service in these industries are frequently ideal for centrifugal compressors and permit them to operate, within their suitable pressure range at efficiencies that cannot be matched by other compressing devices. However, in many other industries, where conditions vary from the ideal by a greater amount, the efficiencies obtained are lower and do not compensate for the relatively high first cost of the centrifugal compressor. Where these machines can be used to advantage, they give extremely good service, and the conditions of any gas-handling installation should be examined carefully before equipment is chosen to determine whether the centrifugal compressor's characteristics are suitable for the work to be done.

Centrifugal compressors should be purchased on specification and guarantee. While manufacturers have standard designs and sizes of these machines, they are not generally manufactured to stock but are built on order, and hence it is not possible to give detailed tabular information or list-price ranges.

Reciprocating (Piston) Compressors

These compressors constitute, with centrifugal fans and rotary blowers, the three classes of gas-handling devices in common use. Of these, fans are most used in the low-pressure range, up to ½ lb. per sq. in. pressure, blowers in the medium-pressure range up to 10 lb. per sq. in. pressure, and reciprocating compressors, usually referred to as compressors, in the higher pressure range. Each of these machines has its best performance characteristics and its most economical applications in the range indicated.

A compressor is similar to a piston pump for water, except that it performs work on a readily compressible fluid, gas, whereas the pump handles a fluid that is but slightly compressible. In pumping water, the work done in compression is negligible. The main source of heat production is friction, and the heat generated in this way is small relative to the mass of the liquid being handled. The heat is absorbed by the liquid with no important temperature rise. In compressing air or gas, however, the work done on the gas in decreasing its volume is considerable and appears as heat, added to that due to friction losses. As the final volume of the gas is relatively small, the temperature rise is relatively large. For good efficiency in compression, the gas must be delivered at a temperature as near as possible to that of the entering gas, *i.e.*, the compression must be as near to the isothermal as can be attained. For this reason most compressors are water jacketed, or in small sizes are provided with exterior fins or other means to facilitate air cooling. Such cooling is adequate for low pressures, not over 90 or 100 lb. per sq. in., and hence such compression is usually carried out in a **single-stage** compressor, *i.e.*, one in which the compression is completed in one cylinder. For higher pressures, the cooling effect is increased by carrying out the compression in two or more cylinders, the discharge from the first and largest cylinder passing into the next cylinder, and so on. Between each two cylinders, cooling additional to that of the jackets is provided in an *intercooler*, a tubular heat exchanger, water cooled, through which the heated air passes. Multistage compression will usually range from 100 to 500 lb. per sq. in. in **two-stage** compressors, from 500 to 2500 or somewhat less in **three-stage** compressors, and up to 3000 lb. per sq. in. or even higher in **four-stage** compressors.

When intercoolers are used in staged compression, the air usually enters each stage at about the original temperature, the compression approaching the isothermal. If the compression were actually isothermal, it would follow the law

$$\frac{P_2}{P_1} = \frac{V_1}{V_2} = \text{constant}$$

If there were no cooling, the compression would be adiabatic, expressed by

$$\frac{P_2}{P_1} = \left(\frac{V_1}{V_2}\right)^n$$

Actually, the compression line on an indicator card from an air compressor lies between the adiabatic and isothermal curves, with leakage as well as cooling tending to bring it toward the isothermal. Assuming that the leakage is kept at a minimum and the cooling brings the discharge as near to the original temperature as possible, Edward D. Thurston, Jr., gives the following theoretical discussion of the work and capacity of air compressors (condensed from his discussion of this subject in Peele, "Mining Engineers' Handbook," 3d ed., John Wiley, New York, 1941). Note that the pressure to which the air or gas is compressed, termed *receiver pressure*, affects the saving due to staging. Maximum saving occurs in two-stage compression when:

$$\text{Receiver pressure} = (\text{low pressure} \times \text{high pressure})^{0.5}$$

(Receiver pressure is the pressure in the receiver, which is a surge tank into which the delivery of an air compressor is discharged in order to flatten out the surges due to the reciprocating action of the machine and give an approximately constant pressure in the system beyond the receiver.)

Maximum saving occurs in three-stage compression when:

$$\text{First receiver pressure} = (\text{low pressure})^{0.67} \times (\text{high pressure})^{0.33}$$
$$\text{Second receiver pressure} = (\text{low pressure})^{0.33} \times (\text{high pressure})^{0.67}$$

Since there must be clearance in a compressor, not all the air in the cylinder can be expelled on each stroke and the volume remaining must reexpand to the low pressure before any fresh air can be drawn in. Consequently, the air compressed is always less than the piston displacement. The ratio of the low-pressure capacity to the displacement is the **apparent volumetric efficiency.**

The entering air comes into contact with the warm cylinder walls and the pistons and expands. Hence the quantity of air drawn in, when measured under external conditions of pressure and temperature, is less than would appear from the apparent volumetric efficiency. The ratio of the actual volume drawn into the cylinder to the displacement is the **true volumetric efficiency,** or

$$\text{True vol. efficiency} = \frac{\text{apparent volumetric efficiency}}{1 + (0.3 \text{ to } 0.5)\left(\dfrac{\text{abs. temp. air delivered}}{\text{abs. temp. air supplied}} - 1\right)^*}$$

---

* After Lucke, "Engineering Thermodynamics," McGraw-Hill.

Air delivered in other than isothermal compression is at a higher temperature than that of the supply and when cooled to the original temperature decreases in volume. Volume actually delivered is termed the **high-pressure capacity hot.** When cooled, the volume delivered is called the **high-pressure capacity cold.** These volumes are equal for isothermal compression.

Expressions for work and capacity of single- and two-stage compressors, given below, use the following symbols:

l. p. = low pressure, lb. per sq. in., absolute.
h. p. = high pressure, lb. per sq. in., absolute.
r. p. = receiver pressure, lb. per sq. in., absolute.
c = clearance, as a fraction of displacement.
cap. = capacity, cu. ft.
d = diameter of cylinder, in.
l = length of stroke, in.
n = cycles per minute.
D = displacement, cu. ft. per min.
s = value of exponent of volume in equation for the compression curve $PV^s = K = 1.406$ for adiabatic compression.
T_1 = temperature of entering air, absolute, °F.
T_2 = temperature of delivery air, absolute, °F.
δ_1 = weight per cu. ft. of entering air.
R_p = ratio, high over low pressure.
E_v = apparent volumetric efficiency.
C_v = specific heat at constant volume.
w = weight of air.

In general, subscripts 1 and 2 refer to low-pressure and high-pressure cylinders, respectively.

Work required to compress a given quantity of air is independent of the clearance, but the size of the cylinder required varies with the clearance. A comparison of the expressions for work for the entire cycle and for the compression alone,

$$\left\{W = \frac{P_2V_2}{s-1}\left[\left(\frac{P_1}{P_2}\right)^{\frac{s-1}{s}} - 1\right]\right\}$$

shows that the work for the entire cycle is s times that for the compression cycle alone, *i.e.*, in Fig. 49 area $ABCD$ is s times area $BCEF$.

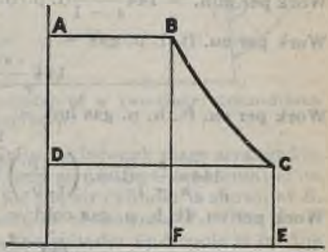


Fig. 49.—Ideal compressor card.

Work of a complete cycle for adiabatic compression $= JsC_vw(T_2 - T_1)$ where J = Joule's equivalent = 778 (approximately).

For single-stage compressors, non-isothermal compression:

$$\text{Work per min.} = 144\frac{s}{s-1}(\text{l. p.})(\text{l. p. cap.})\left[(R_p)^{\frac{s-1}{s}} - 1\right]$$

or

$$= sC_vw(T_2 - T_1) \text{ for adiabatic compression only}$$

Work per cu. ft. l. p. gas $= 144\frac{s}{s-1}(\text{l. p.})\left[(R_p)^{\frac{s-1}{s}} - 1\right]$

Work per cu. ft. h. p. gas hot $= 144\frac{s}{s-1}(\text{l. p.})\left[(R_p)^{\frac{s-1}{s}} - 1\right]R_p^{\frac{1}{s}}$

Work per cu. ft. h. p. gas cold $= 144\frac{s}{s-1}(\text{h. p.})\left[(R_p)^{\frac{s-1}{s}} - 1\right]$

M.E.P. lb. per sq. in. $= \frac{s}{s-1}(\text{l. p.})E_v\left[(R_p)^{\frac{s-1}{s}} - 1\right]$

Displacement per min., cu. ft. $= \frac{d^2ln}{2200}$

Low-pressure cap. per min. $= DE_v = D\left[1 + c - (cR_p)^{\frac{1}{s}}\right]$

High-pressure cap. per min. hot $= \frac{DE_v}{R_p^{1/s}} = \frac{\text{l. p. cap.}}{R_p^{1/s}}$

High-pressure cap. per min. cold $= \frac{DE_v}{R_p} = \frac{\text{l. p. cap.}}{R_p}$

Apparent volumetric efficiency $= E_v = 1 + c - cR_p^{1/s} = \frac{\text{l. p. cap.}}{D}$

For two-stage compression, with perfect intercooling and any receiver pressure:

Work per min. $= 144\frac{s}{s-1}(\text{l. p.})(\text{l. p. cap.})\left[\left(\frac{\text{r. p.}}{\text{l. p.}}\right)^{\frac{s-1}{s}} + \left(\frac{\text{h. p.}}{\text{r. p.}}\right)^{\frac{s-1}{s}} - 2\right]$

Work per cu. ft. l. p. gas =

$$144\frac{s}{s-1}(\text{l. p.})\left[\left(\frac{\text{r. p.}}{\text{l. p.}}\right)^{\frac{s-1}{s}} + \left(\frac{\text{h. p.}}{\text{r. p.}}\right)^{\frac{s-1}{s}} - 2\right]$$

Work per cu. ft. h. p. gas hot =

$$144\frac{s}{s-1}(\text{l. p.})\left(\frac{\text{h. p.}}{\text{r. p.}}\right)^{\frac{1}{s}}\left(\frac{\text{r. p.}}{\text{l. p.}}\right)\left[\left(\frac{\text{r. p.}}{\text{l. p.}}\right)^{\frac{s-1}{s}} + \left(\frac{\text{h. p.}}{\text{r. p.}}\right)^{\frac{s-1}{s}} - 2\right]$$

Work per cu. ft. h. p. gas cold =

$$144\frac{s}{s-1}(\text{h. p.})\left[\left(\frac{\text{r. p.}}{\text{l. p.}}\right)^{\frac{s-1}{s}} + \left(\frac{\text{h. p.}}{\text{r. p.}}\right)^{\frac{s-1}{s}} - 2\right]$$

Displacement per min. $= \frac{d_1^2l_1n}{2200}$ for l. p. cylinder

Low-pressure cap. per min. $= D_1\left[1 + c_1 - c_1\left(\frac{\text{r. p.}}{\text{l. p.}}\right)^{\frac{1}{s}}\right]$ for l. p. cylinder

High-pressure cap. per min. hot $= (\text{l. p. cap.})\left(\frac{\text{l. p.}}{\text{r. p.}}\right)\left(\frac{\text{r. p.}}{\text{h. p.}}\right)^{\frac{1}{s}}$

High-pressure cap. per min. cold $= \dfrac{(\text{l. p. cap.})}{R_p}$

M.E.P., referred to first stage:

$$\frac{s}{s-1}(\text{l. p.})E_{v_1}\left[\left(\frac{D_1E_{v_1}}{D_2E_{v_2}}\right)^{\frac{s-1}{s}} + R_p\left(\frac{D_2E_{v_2}}{D_1E_{v_1}}\right)^{\frac{s-1}{s}} - 2\right]$$

Apparent volumetric efficiency, first stage $= \dfrac{\text{l. p. cap.}_1}{D_1}$

Apparent volumetric efficiency, second stage $= \dfrac{\text{l. p. cap.}_2}{D_2}$

A section through the high-pressure stage of a two-stage compressor, with intercooler between stages, is shown in Fig. 50. This compressor is steam driven, with the steam cylinder and the air cylinder for each stage arranged in tandem. The crosshead A, steam piston B, and air piston C are mounted on the single piston rod D. The water jacket for the air cylinder is shown at E. The low-pressure cylinder, not cut by this section, is shown at F. The intercooler G receives the air from the low-pressure cylinder and cools it by the water in the tubes H, and then it passes into the pipe J and so to the inlet valves of the high-pressure cylinder. These valves, shown in the inset on Fig. 50, consist of a light steel ring K, held against a seat by springs L. The similarly constructed discharge valves are shown at M and the discharge connection at N.

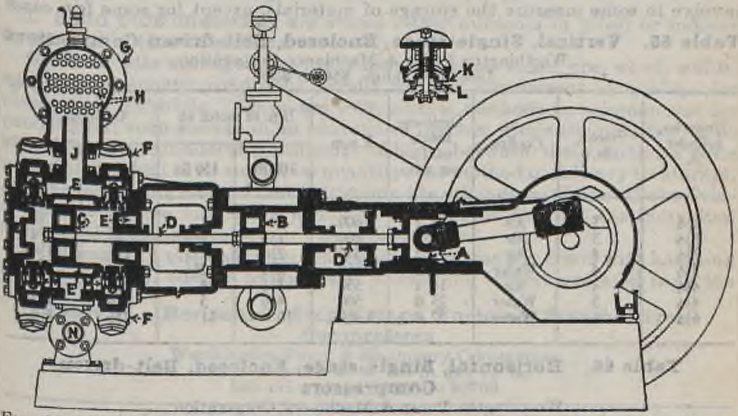


FIG. 50.—Section through the high-pressure cylinder of a two-stage steam-driven reciprocating air compressor.

Not shown in Fig. 50 is the unloading arrangement, which is for the purpose of unloading the pressure, either for starting or for regulating while running. Unloaders may be controlled by hand, mechanically, or magnetically. Compressors are also supplied with the necessary governing and regulating devices and lubricating arrangements.

Moisture in compressed air is to be avoided because it may freeze in, or corrode the pipe lines, or it may cause line and meter troubles. Such moisture is condensed when the vapor in the heated compressed air condenses out as the air cools after leaving the final stage. To obviate this, **aftercoolers** are introduced between the compressor and the compressed-gas piping system to remove the moisture that condenses when the compressed gas is cooled.

Standard air compressors are also used for compressing gas to ordinary pressures and for gas handling. For compressing ammonia and for some other work, such as the compression of gases to extreme pressures for synthetic chemical manufacture, special designs of compressors are available. Dry vacuum pumps, in general similar to air compressors, are also available in many designs and makes. Tables 65 to 71 give sizes, capacities, and other data for several commonly used types of compressors.

STORAGE OF MATERIALS

All problems relating to the handling and movement of materials also involve in some measure the storage of materials, except for some few cases

Table 65. Vertical, Single-stage, Enclosed, Belt-driven Compressors

Worthington Pump & Machinery Corporation
List-price range, $50 to $300

Diameter, cylinder, in.	Stroke, in.	Cooling	Displacement, cu. ft. per min.	R.p.m.	H.p. of motor at		Openings, in.	
					100 lb.	150 lb.	Inlet	Outlet
2½	3	Air	5.2	600	1	1	¾	¾
3⅛	3	Air	8.0	600	1½	1½	¾	¾
3½	4	Air	12.2	550	2½	2¾	1¼	1¼
3½	4	Water	12.2	550	2½	2¾	1¼	1¼
4¼	4	Air	18.1	550	3¾	4	1¼	1¼
4½	5	Water	23.0	500	4½	5	1½	1½
6½	6	Water	52.0	450	10	11	2½	2½

Table 66. Horizontal, Single-stage, Enclosed, Belt-driven Compressors

Worthington Pump & Machinery Corporation
List-price range, $600 to $1800

Diameter, cylinder, in.	Stroke, in.	Displacement, cu. ft. per min.	R.p.m.	Air pressure, lb. per sq. in.	Brake-h. p. at rated pressure	Openings, in.	
						Inlet	Outlet
7½	6	106	350	80–100–125	15.9–17 –18	2½	2½
8½	9	170	300	80–100–125	25 –27 –29	3	3
10	10	250	285	80–100–125	36 –38.5–41	3½	3½
11	12	350	270	80–100–125	51 –57 –60	. . .	4
8½	6	136	350	40–60	15–18.5	. . .	3
10	9	245	300	40–75	27–34	3½	3½
11	10	312	285	40–75	34–43	4	4
13	12	495	270	40–75	54–70	5	5
12	9	350	300	20–45	30–42	4	4
13	10	435	285	30–45	42–52	6	6
15	12	660	270	30–50	59–74	7	7

where the movement is incidental to the process, or where the movement of raw materials or finished products is direct. It is therefore necessary to give some consideration to the storage methods commonly employed in the chemical engineering industries and an indication of how the movement into, and out of, storage can be conveniently made.

The materials commonly stored have the following characteristics:

A. Solid materials.
  1. Bulk.
  2. Containers.
B. Liquid Materials.
  1. Bulk.
  2. Containers.
C. Gases.
  1. Bulk.
  2. Containers.

A. Storage of Solid Materials

1. Solid bulk materials are stored either outdoors in piles; or indoors in piles, or in bins or bunkers.

Outdoor pile storage of solids is usual for coal, stone, ore, wood, sulfur, and, where climatic conditions permit without impairment of quality, for some other materials. There are four general methods in common use for carrying out such storage in an economical manner where the quantity to be stored is of a considerable amount. Choice between these methods rests upon the material to be stored, its quantity, the method of delivery to storage, and the reclaiming method best fitting into the other operations at the individual plant. These methods are subject to variation to meet the individual needs of the storage problem. The methods are:

a. Storage in a pile under a traveling bridge, or gantry, crane with handling into, and out of, storage by means of a traveling bucket operated from the

Table 67. Horizontal, Single-stage, Enclosed, Steam-driven Compressors

Worthington Pump & Machinery Corporation
Steam pressure, 90 to 150 lb. per sq. in.
List-price range, $1200 to $2700

Diameter steam cylinder, in.	Diameter air cylinder, in.	Stroke, in.	Displacement, cu. ft. per min.	R.p.m.	Air pressure, lb. per sq. in.
7	7½	6	106	350	80–100–125
8	8½	9	170	300	80–100–125
9	10	10	250	285	80–100–125
10	11	12	350	270	80–100–125
7	8½	6	136	350	40–60
8	10	9	245	300	40–75*
9	11	10	312	285	40–75*
10	13	12	495	270	40–75*
8	12	9	350	300	20–45†
9	13	10	435	285	20–45†
10	15	12	660	270	20–50†

* 110-lb. steam necessary for maximum air pressure.
† 125-lb. steam necessary for maximum air pressure.
All machines have piston-type steam valves.

Table 68. Duplex, Horizontal, Belt-driven Compressors

Ingersoll Rand Company

For single-, two-, three-, or four-stage compression.

Type XRB-1 has duplex, single-stage air cylinders and is automatically regulated by the five- or three-step, free-air controller when specified.

Type XRB-2 has compound, two-stage air cylinders and is automatically regulated by the three-step free-air controller when specified.

Type XCB-1 has duplex, single-stage air cylinders and either five- or three-step clearance control regulation.

Type XCB-2 has compound, two-stage air cylinders and either five- or three-step clearance control regulation.

Diameter cylinder, in.		Stroke, in.	R.p.m.	Capacity, cu. ft. per min. piston displacement at 100 lb. per sq. in. discharge pressure		
Low pressure	High pressure			Sea level	5,000-ft. altitude	10,000-ft. altitude
12	7½	8	335	348		
13	7½	8	335	.....	409	
14	7½	8	335	.....	.....	475
13½	8	10	300	494		
14	8	10	300	.....	530	
15	8	10	300	.....	.....	610
15	9¼	12	277	676		
16	9¼	12	277	.....	770	
17	9¼	12	277	.....	.....	870
17	10½	12	277	870		
18	10½	12	277	.....	975	
19	10½	12	277	.....	.....	1,087
18	11	14	257	1,052		
19	11	14	257	.....	1,173	
20	11	14	257	.....	.....	1,302
20	12½	14	257	1,302		
21	12½	14	257	.....	1,435	
22	12½	14	257	.....	.....	1,575
22	14	14	257	1,575		
23	14	14	257	.....	1,723	
24	14	14	257	.....	.....	1,875

Table 69. Two-stage Horizontal, Belt-driven Compressors

Ingersoll-Rand Company

Air pressures 150 to 400 lb. per sq. in.

Diameter, cylinder, in.		Stroke, in.	R.p.m.	Piston displacement, cu. ft. free air per min.
Low pressure	High pressure			
4	2⅛	4	500	28
6	2⅞	6	350	65
8	3⅜	8	300	133
10	4⅞	10	275	241

crane. This bucket can unload from cars, barges, or other means of transport and can deliver to some means of transport, directly to a secondary storage in the plant, to a conveyor, or directly to such equipment as furnaces.

b. Storage in piles on either side of a track, served by a locomotive crane. The crane is used to place the material in storage, taking it from a car, truck, or barge, or from a track hopper in which it has been dumped from hopper-bottom cars, or to which it has been delivered by some other means of handling. In reclaiming from storage, the crane is used for loading the material into cars, delivering it into a track hopper served by a conveyor, or to the boot of a bucket elevator, or to the loading hopper of a skip hoist.

c. Overhead systems are used, employing a monorail car and bucket or a cable way and bucket. These systems can be controlled from a central point, or the operator can travel in a car accompanying the bucket. Overhead systems can unload and pile from any means of transport and can serve any type of conveyor or transport to take the material out of storage. These systems can also be arranged to deliver the material directly to the point of use within the plant, or to bins or bunkers in the plant.

d. Drag-scraper systems are useful in many problems of bulk outdoor storage. These systems consist of a scraper bucket to which both ends of a cable are attached. The cable passes over a pulley and also around the drum of a hoisting engine. The pulley is detachably mounted on one of several posts located at suitable points around the storage area. The scraper can then be caused to move back and forth across the storage area, piling and reclaiming material. Drag scrapers usually deliver to a conveyor, a bucket elevator, or skip hoist.

Table 70. Belt-driven Dry Vacuum Pumps
Ingersoll-Rand Company

Size, cylinder, in.		R.p.m.	Piston displacement, cu. ft. per min.
Diameter	Stroke		
10	4	500	181
14	5	400	355
18	7	350	720
22	9	275	1085
26	11	275	1853
31	12	220	2295

Table 71. Steam-driven Dry Vacuum Pumps
Ingersoll-Rand Company

Cylinders, diameter in.				Stroke, in.	R.p.m.	Piston displacement, cu. ft. per min.
Steam pressure, 125–175 lb.	Steam pressure, 80–120 lb.	Steam pressure, 35–75 lb.	Vacuum			
6	7	11½	18	6	350	615
8	9	13½	22	8	275	965
10	12	18	26	10	275	1680
12	14	21	31	12	220	2295

Pile storage for wood, used in paper mills and other industries, differs from the general cases mentioned. Such storage is carried out with conveying devices that have been specially developed for the purpose.

Contents of outdoor storage piles cannot be calculated exactly unless the pile is within walls and is kept at an even thickness, as is sometimes done. Large random piles can be estimated with sufficient accuracy for most purposes by obtaining the contours of sections evenly spaced over the storage area, say 5 to 10 ft. apart, and then figuring the cubic contents of the material between sections. Such sections are easily determined by the use of surveying instruments.

Smaller piles are frequently estimated by means of soundings, but this method leaves much to be desired as to accuracy, whatever its convenience.

Indoor pile storage is usually employed for materials that must be kept dry or protected from the weather for some reason. Among these are ceramic materials, scrap metals, glass sand, agricultural products, minerals, and chemical products. The usual method is to have a large storage shed or room within one of the buildings of the plant for each material; or, when quantities are relatively small, to divide a single storage space into several sections by means of partitions. Frequently such storage spaces are served by an overhead crane or a monorail crane, which is used for both storing and reclaiming the materials. Such a crane can serve the processing equipment direct where that is convenient, or conveyors may be interposed when such an arrangement is better.

Indoor bulk storage may be served entirely by conveyors. For example, two belt conveyors can be used, one at the top of the storage space for taking material from an elevator and distributing it to the storage spaces, and the other at the bottom, beneath the floor, fed through a feeding device or loading hopper, for reclaiming. Another conveyor application useful in such cases is the pivoted bucket carrier, which will serve for both storing and reclaiming.

Where the quantities stored are small, it may not be economical to handle otherwise than with manual labor.

Intermediate quantities in bulk storage of dry solids are often served by electrical industrial trucks or tractors and trailers. The trucks or the trailers, for this purpose, can be equipped with dump bodies, which serve to eliminate much hand labor that would otherwise be necessary.

Quantities in indoor pile storage are easily estimated by obtaining the contour of the pile by soundings. This method is not accurate but serves most purposes. Where piles can be kept level, *i.e.*, evenly distributed over the whole storage floor, more accurate estimates can be made from the dimensions of the storage space and the height of the pile on the wall. Where this method is used, it is well to have an indicating scale in feet and fractions of feet painted on the wall.

Indoor storage of bulk solids in bins or bunkers is frequently employed. Particularly where such storage can be arranged to feed by gravity to the production equipment or shipping facilities, this system finds an economical application. Hopper bottoms or parabolic bottoms are usual for such bins or bunkers. The material will then feed directly to a chute or spout by gravity, whenever the outlet gate is opened. In the storage, by this method, of some materials that are likely to clog in the outlet, it is customary to install mechanical or air agitators, designed to keep the material loose and free flowing. The chute or spout can feed direct to the equipment as is usually the case with furnaces; or it can charge a larry car or similar transport device which serves to move the material to the point of use, as is usual in gas or coke practice. Bins or bunkers can also be arranged to feed to a belt or other conveyor, or to load into industrial trucks or cars.

Material is placed in bins or bunkers through the open top, or through gates in a closed top. One method in common use for this purpose is to employ an overhead crane or monorail crane with a bucket. A belt conveyor passing over the top of the storage, with suitable unloading arrangements, and fed by a bucket elevator or skip hoist, is another common method of storing. A third method employs a pivoted bucket conveyor, which can also be used for reclaiming the material.

Indoor storage of bulk solids in silos is an alternate method that finds much favor in some industries, particularly in the lime, cement, and ceramics fields. Silos are cylindrical structures of concrete, tile, steel, or wood. Their advantages lie in the economy of floor space occupied for a given amount of storage and their relatively inexpensive construction. Compared to bins and bunkers, silos are less adaptable as to possible locations about the plant, and they have a relatively greater height for the amount of storage space provided. Because of this great height, and because the bottoms are seldom raised far above the ground level, silos must usually be loaded by means of an elevator or skip hoist and unloaded by means of a conveyor or by trucks working in a pit.

Where several silos are in use, they can be grouped; and a belt conveyor can be arranged to take material from an elevating device and unload it into any one of the silos desired. Reclaiming from silos can best be done by an arrangement of conveyors suited to the movements of the materials reclaimed, or by means of wheeled transport fed from hopper gates under the silos.

Quantities in storage in bins, bunkers, or silos are figured with sufficient accuracy for most purposes from the known dimensions of the storage and the depth of the material stored. Another method, more accurate than the first, is to weigh the material as it is stored and then to weigh the amounts taken from storage. This method incurs considerable first cost for the purchase of weighing devices but is often justified where a close check on quantities is desired.

2. Solid materials stored in containers present few problems not already considered in this section on the movement of materials. Such containers are usually boxes, bags, barrels, or steel drums. The storage methods employed usually consist of piling or storing these containers outdoors when the material stored and the weather conditions permit, or indoors in other cases.

One of the most useful methods of handling materials into and out of such storage employs the electrical industrial truck, equipped with the high-lift or tiering feature when stacks or piles are to be used, and, if the loads to be handled are heavy, equipped with a crane.

Another method widely used employs an overhead crane or monorail. This is particularly useful for handling bags, several of which can be taken at one time in a rope sling.

For barrels and drums, much special equipment is available, including barrel elevators and barrel stackers.

In storing these containers within buildings, care should be taken not to overload the storage floors.

Quantities of barrels, bags, and drums in storage can be easily estimated by counting the number in some unit of space and then extending the count to cover the whole space occupied, making proper allowance for those areas where the space is not filled to an even level.

No mention has been made of the storage of finished solid products in containers. In general, such materials will be stored in the manner that is described under Liquid Materials Stored in Containers (see p. 2283).

B. Storage of Liquid Materials*

1. Bulk storage of liquid materials employs reservoirs where the material can be exposed to the elements, and tanks for all other purposes. The opportunities for using reservoirs are few, because most liquids must be protected from contamination or dilution. Water is the liquid commonly stored in this way in chemical engineering industries. Where the quantities dealt with are large, the reservoir becomes a problem for the civil engineer, and adequate assistance of this nature should be employed. For small reservoirs, concrete-walled excavations or concrete tanks without tops, either sunken in the ground or raised above it, are in common use. Such tanks should be constructed with reinforced walls adequate to hold the contents when the reservoir is completely filled. The concrete should be waterproofed so as to prevent any possibility of leaking. Quantities stored in this way can be easily figured from the known dimensions of the storage and the measured depth of the liquid.

Storage of the liquid materials of chemical engineering industry is usually carried out in tanks. Such tanks are classified as *vertical*, *horizontal*, *rectangular*, or *spherical*. Vertical tanks are cylindrical tanks with the axis in a vertical plane. Horizontal tanks are cylindrical tanks with the axis in a horizontal plane. Rectangular and spherical tanks are shaped as indicated by the names. Vertical tanks are most commonly used for outdoor storage, in tank fields, for such materials as petroleum, tar, and asphalt. These tanks are also sometimes used for indoor storage. Another frequent use for vertical tanks is for elevated water storage. Horizontal and rectangular tanks are the most usual types for all sorts of storage of liquids. Spherical tanks are used for the storage, generally outdoors, of volatile liquids, such as natural-gas gasoline, which are likely to develop a pressure in the tank due to the volatilization of part of the liquid.

Tanks are constructed of metal, generally steel plate, wood staves, or concrete. In cases where the liquid being stored is of a nature to attack the material of the tank, it may be lined with rubber or some other protective material, or coated on the inside with some resistant coating. Wood-stave and concrete tanks are frequently treated to prevent leaking.

Filling and drawing from liquid-storage tanks are usually done with pumps. In many cases, however, tanks can be so located that one or both of these operations can be accomplished by gravity. Obviously, tanks can be readily connected by pipe lines to any point in the plant.

Quantities stored in tanks can be readily determined from the dimensions of the tank and the depth of the liquid in the tank which can be obtained by means of a liquid level gage or by sounding. The horizontal cross-sectional area of a vertical tank is the area of the circle having the same diameter as the tank, while that of a rectangular tank is the length multiplied by the breadth.

When figuring stored quantities from the dimensions of a tank, the simplest method is to divide the volume in cubic inches by 231, the number of cubic inches in 1 gal.

With horizontal tanks the contents will be influenced by the shape of the tank ends, which may be convex, straight, or concave. For ordinary estimat-

---

* See p. 2353.

ing, the effect of the ends may be neglected. The content of the horizontal tank, when filled, is found by multiplying the area of the end of the tank by the length and dividing by 231. When the tank is only partly filled, a deduction must be made from the total contents of the tank equal to the unfilled portion above the liquid. If the tank is filled to a point above the axis, then subtract from the total contents an amount equal to the contents of a space having the same length as the tank and an end area equal to the unwetted segment of the tank end. If the tank is filled to a point below the axis, then the contents become equal to that obtained by multiplying the length of the tank by the area of the wetted segment of the end of the tank.

When it is desired to figure the contents of a horizontal tank with curved ends more accurately, it becomes necessary to determine the radius of the curvature or to obtain this from the tank maker. This is often inconvenient, and the alternate method of calibration is recommended. This consists in completely filling the tank with water, then drawing off the contents in steps, reducing the depth of the liquid by even amounts and accurately measuring the quantity of liquid withdrawn for each of these unit reductions in depth. In this way a table can be compiled giving the tank contents for each unit of depth of liquid in the tank. In practice, with such a table and a liquid level gage, the quantity in a tank can be readily obtained.

The calibration method recommended for horizontal tanks is also best for use with spherical tanks, as its use avoids the recurrence of tedious calculations.

2. Liquid materials stored in containers are generally packed in barrels, kegs, drums, cans, or in glass containers. The storage of the barrels, kegs, and drums will be the same as that described for solids packed in similar containers. For cans, one of the most convenient methods is to stack the cans several tiers high on palettes or skid platforms and then handle them into and out of storage by means of electric or hand-operated industrial lift trucks. Where small cans are to be stacked to a considerable height, machinery developed for service in food canneries will be found useful for both stacking and reclaiming.

Storage for glass containers, such as carboys or bottles, may be either on skids or palettes, or directly on the floor. Movement into and out of storage is best carried out by means of hand-operated or electrical industrial trucks. Special bodies are available for these trucks for handling carboys efficiently.

Quantities in storage, of large-sized containers for liquids, are quickly estimated by counting the number on a unit area and then extending this quantity to cover the whole storage. With cans or other small containers, such a method is not so convenient. In this case it is better to use some kind of indicators in the stack. Such indicators can, for instance, be used to mark every gross in the stack, and then quantities are obtained by counting the number of indicators and multiplying by the unit.

Floor loadings must, of course, be considered with any kind of storage in buildings, but are particularly important with small containers such as cans because of the ease with which such containers can be stacked to a height that overloads the floor. Ordinarily, building floors can be loaded to 150 lb. per sq. ft. Mill-type buildings and reinforced-concrete structures are usually constructed for loads up to 250 lb. per sq. ft. In many cases, buildings of special construction are good for higher loadings. In any case, the chemical engineer should ascertain the load that the floors of the buildings in his plant can safely support and take pains to see that this loading is not exceeded.

C. Storage of Gases

1. Gases stored in bulk are usually stored in **holders or tanks.** Holders are of two types: telescopic and stationary.

Telescopic gas holders are the familiar type of gasometer, exclusively employed until recently for the storage of city gas supplies. These holders consist of a number of concentrically arranged steel plate bands or rings. The bottom ring has the largest diameter, is fixed in position, and has a gas-tight bottom. The top ring is the smallest in diameter and is furnished with a tight top. Intermediate rings are evenly graded in diameter between the top and bottom rings. Where these rings fit one within the other, liquid seals are provided to prevent the escape of gas. The whole structure is carried within an open supporting frame, upon the columns of which rest guide wheels or rollers attached to the rings, holding them in a fixed horizontal position with relation to the frame and to each other. These rings are, however, free to move up and down, impelled by the gas pressure within the holder.

Such telescopic holders are constructed in very large capacities, as well as in small sizes, holders of 10,000,000 and 15,000,000 cu. ft. capacity being not uncommon. Since these holders usually involve a considerable investment, the proper procedure for obtaining one is to obtain competitive bids upon specifications from several manufacturers who specialize in this equipment.

Stationary gas holders are a more recent development than the telescopic type. These are sometimes called "waterless" holders, which term is hardly appropriate, as a liquid seal must also be employed in this design. Holders of this type consist of cylindrical or polygonal tanks with vertical axes. The bottom and sides are gas-tight, while the top, usually rounded or slightly conical, is vented. Within the tank is a horizontal partition, carried in a liquid seal at the point where its circumference is in contact with the holder wall and arranged to float up or down on top of the gas. As gas is forced into the tank, the partition is raised by the gas pressure created, and, as gas is taken from the tank, the partition automatically lowers, maintaining even pressure in the gas.

Bulk storage of gas is usually carried out in tanks of the types just described, which permit the pressure to be maintained fairly constant. Where storage is carried out in ordinary tanks, arrangement must be made to prevent the gas becoming diluted with air, and also to prevent too great a pressure from building up within the tank. Ordinarily, such tanks are used only for compressed air, in which case they are called *air receivers.*

Gas storage in holders or tanks is accomplished by means of compressors or blowers, taking the gas from its source and feeding to the storage through pipe lines. Reclaiming of gas from storage is also done by means of a compressor or blower, or by utilizing the gas pressure in the storage when this is great enough. Such reclaiming is done through piping systems. Frequently the gas pressure in the storage tank is sufficient to partly serve, and it can then be supplemented by additional pressure imparted to the gas by means of a blower or compressor.

Quantities of gas stored in holders must be calculated for some standard conditions, due to the changes that occur in gas volumes with variations in pressure and temperature. The gas placed in storage and that removed from storage may be metered in one of many types of gas meters available and in this way a running check kept on the quantity stored.

2. Gas stored in containers is usually stored in what are called gas "cylinders." These are heavily constructed tanks of drum or bottle shape. Heavy construction is used in order that the gas may be stored under considerable pressure, thus permitting a relatively large amount of gas (when considered from the standpoint of its volume under ordinary pressures) to be stored in a relatively small container. Container storage for gas is usually employed for the storage and shipment of such gases as oxygen, carbon dioxide, acetylene, propane, butane, and hydrogen. It provides a convenient method for gas shipment. When such gas cylinders are stored in the plant, their relatively great weight, due to the heavy construction used, should be kept in mind so that storage floors will not be overloaded.

FEEDERS AND FEEDING MECHANISMS

BY GEORGE A. PROCHAZKA, JR.

The need for obtaining a uniform flow of materials into and out of process with a minimum of manual attendance, has led to the design of a wide variety of automatic feeding devices. These have been developed in greatest number for handling solids. Liquids and gases are readily handled by means of pressure vessels, pumps, and fans. Design in these fields, therefore, has been largely a matter of coordinating pumps, compressors, fans, valves, etc., with meters, floats, thermometers, photoelectric cells, and other indicating devices.

In the selection of a suitable feeder, the physical state of the material is a most important factor. Thought usually is given to size and uniformity of product, also whether it is sticky or free flowing. Other factors considered are tonnage, accuracy of feeding, constant or intermittent use, power consumption, first cost, and maintenance. A wider selection generally is possible for materials which are easy to handle than for those having troublesome characteristics. For the application of feeding devices to countercurrent decantation see Sec. 15 pp. 1651 to 1652.

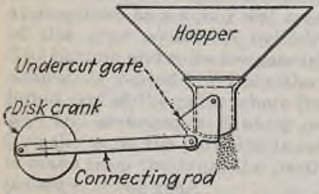


FIG. 51.—The undercut-gate feeder.

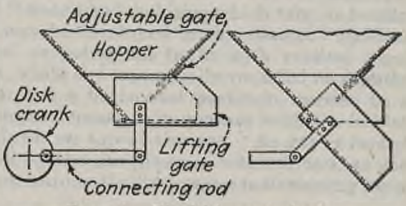


FIG. 52.—The lifting-gate feeder.

C. Kemble Baldwin (*Trans. Am. Soc. Mech. Eng.* May, 1909) made an analysis of the principles used in the design of automatic bulk feeders and found eleven general types. It appears necessary now to add only three, namely, vibrating, chain, and flight feeders. Figures 51 to 59, inclusive, and much of the accompanying discussion have been reproduced from Baldwin's presentation which has been rearranged to include swinging plate with plunger feeders.

The undercut-gate feeder, Fig. 51, made of cast iron or steel plate, is pivoted near the top, the gate being swung back and forth by means of an eccentric or crank. This type of feeder is best adapted to fine-sized, free-flowing material. Material containing lumps is likely to bridge. As the discharge is intermittent, the feeder is generally used for chain or bucket conveyors, the strokes being timed with the buckets. The capacity may be changed only by varying the length or the number of strokes. For the drive, an adjustable crank is preferable; no decided advantage accrues from the quick return obtained with the more costly eccentric drive. The so-called **stirrup feeder** is a variant using a transverse baffle above the swinging gate.

The lifting-gate feeder, Fig. 52, also gives an intermittent feed suitable principally for a chain or bucket elevator or conveyor. The chute is hinged so that, when down, the material will flow out of the hopper and, when raised, the discharge is stopped. Motion is obtained from a crank or eccentric. This feeder will handle material regardless of size, but it must be sufficiently free flowing to move by gravity when the chute is lowered. The capacity may be adjusted by varying the number of strokes and also in a measure by increasing the length of stroke.

The screw-conveyor feeder, Fig. 53, will deliver a constant stream of material, but it must be of such form as to flow by gravity to the screw. The capacity can be changed, on a machine of given size, only by altering the speed of the screw shaft. This type of feeder has been widely used for handling pulverized material such as coal, cement, etc.

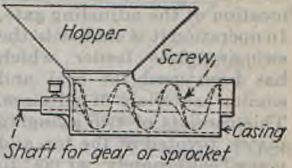


FIG. 53.—The screw-conveyor feeder.

The roll feeder, Fig. 54, has been used extensively in the mineral industries for handling large and small material from bins to weigh cars, etc. The roll is located under the hopper so that the material will not flow when the roll is stationary but will carry forward when the roll rotates. The capacity is determined by the speed and width of the roll and the thickness of the stream, as fixed by the adjustable gate. The disadvantage of the feeder is in the large amount of head room required as a roll 6 to 8 ft. in diameter must be used to handle run-of-mine material.

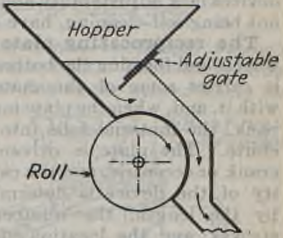


FIG. 54.—The roll feeder.

The rotary-paddle feeder, Fig. 55, acts both as a feeder and measuring device. It is used for fine material which flows readily from the blades. The capacity is fixed by the speed of the paddle shaft. The **pocket feeder**, also called the star or revolving-door feeder, is a variant in which the paddle is tightly housed to permit delivery against vacuum or pressure.

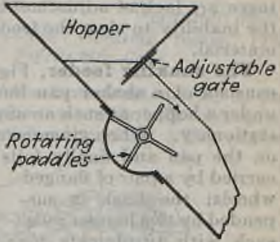


FIG. 55.—The rotary-paddle feeder.

The revolving-plate feeder, Fig. 56, was developed for feeding stamp mills in the recovery of gold from ore. The inclined plate driven by gears, either from above (as shown) or from below, moves the material out of the hopper where it is scraped off by the skirt board. With an adjustable skirt board sticky material may be handled, as the curved plate will scrape the material off the revolving plate and into the chute. The capacity is fixed by the speed of the plate and the location of the adjustable gate.

The apron-conveyor feeder, Fig. 57, may be of any of the various types of apron flights or a rubber or canvas belt supported on idlers. The capacity per foot of width is fixed by the speed of the apron and the position of the adjustable gate. Apron, pan, and belt feeders have been designed in almost unending variety and frequently are used because they have the advantage that the material can be moved over a greater distance than with most other types of feeders.

The plunger feeder, Fig. 58, pushes the material along the bottom plate and off into the chute. The plunger may be built in one or in two parts, in the latter case the parts are moved alternately by a rocker shaft. The capacity is fixed by the length and number of strokes and the location of the adjusting gate. In operation it is similar to the swinging-plate feeder which has been used for coal and similar materials of all sizes. This consists of two swinging plates, pivoted at the top and moved alternately so as to push the material along the bottom plate. Both these devices lack adjustability and, not being self-cleaning, have a tendency to freeze in the wintertime.

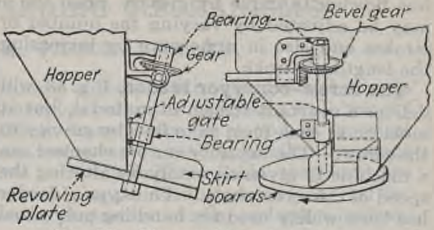


FIG. 56.—The revolving-plate feeder.

The reciprocating-plate feeder consists of a plate mounted on four wheels and forming the bottom of a hopper. When the plate moves forward, it carries some of the material with it, and, when the plate moves back, the material falls into the chute. The plate is driven by crank or eccentric, and the capacity of the device is determined by the length, the number of strokes, and the location of the adjusting gate. The disadvantages are lack of adjustment and the inability to clear the feeder of material.

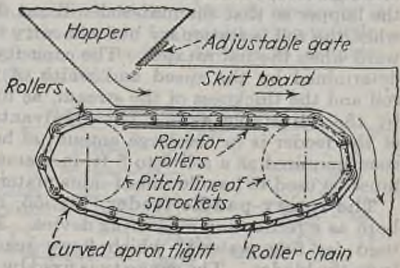


FIG. 57.—The apron-conveyor feeder.

The shaking feeder, Fig. 59, consists of a shaker pan located under a hopper at such an angle that the material will not flow when the pan is stationary. When given a reciprocating motion, the material moves forward on the pan and finally falls into the chute. The front end of the pan is carried by a pair of flanged wheels; the back is suspended by two hanger rods, each with turnbuckle for adjusting the angle of the pan. As the number of strokes is difficult to change, these feeders usually are designed for about 75 per min., a number determined by experiment. The length of stroke varies from 4 to 12 in. The angle of the pan is fixed by the capacity desired and the nature of the material handled. For coal, stone, ore, etc., 8 to 10 deg. slope is sufficient, but clay and other sticky substances require from 15 to 20 deg. The shaking feeder has many advantages, particularly its flexibility and self-cleaning feature. It will handle most materials regardless of size or condition.

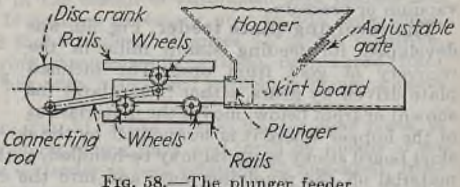


FIG. 58.—The plunger feeder.

The vibrating feeder, Fig. 60, is a Jeffrey-Traylor design. The Jeffrey Manufacturing Co. makes various styles of feeders such as a pan type, a tubular type, and one jacketed for heating, drying, or cooling. Materials ranging from 300 mesh to 3-ft. cubes as well as mixtures of earth, sand, and rock have been handled. Capacities range from 10 lb. to 1250 tons per hr. The power consumption with a vibrating feeder is exceptionally low, 4 watts being reported for sand fed at the rate of 1500 lb. per hr. Motion is obtained by means of an electromagnet anchored to the main frame of the machine. The oscillating armature or keeper is attached by means of a clamp to a set of springs or vibrator bars. These are clamped at their ends to the main frame but are free to flex at the center where the keeper clamp, which also carries the material handling deck, is located. A vibrating feeder, driven by 60-cycle alternating current, makes 7200 strokes per minute, and control is obtained by adjusting the current input, thereby regulating the pull of the electromagnet and thus the length of the stroke. The Syntron Co. and Allis-Chalmers Manufacturing Co. also make vibrating feeders.

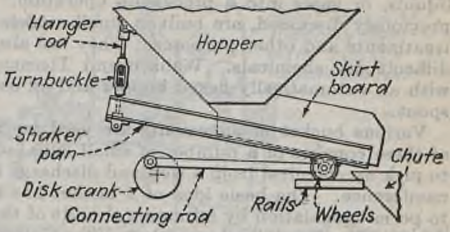


FIG. 59.—The shaking feeder.

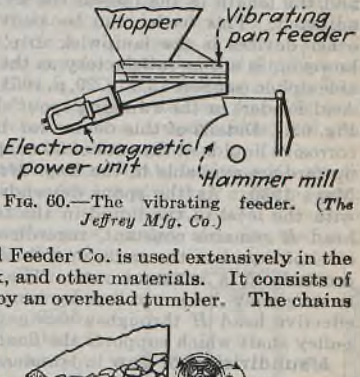


FIG. 60.—The vibrating feeder. (*The Jeffrey Mfg. Co.*)

The Ross feeder, Fig. 61, a patented device, made by the Ross Screen and Feeder Co. is used extensively in the mineral industries for handling ore, rock, and other materials. It consists of a curtain of heavy, endless chain driven by an overhead tumbler. The chains are suspended so as to lie on the material and travel with it. These feeders are made in widths of 6 in. to 20 ft., capacities from 1/4 to 2000 tons per hr., and in styles fitted with ship-anchor chain ranging from small steamer to heavy battleship size. Crushed material can be fed from storage with an accuracy of about 2 per cent variation in tonnage flow. The feeder will handle any granular material of any size from primary steam shovel to coal slack.

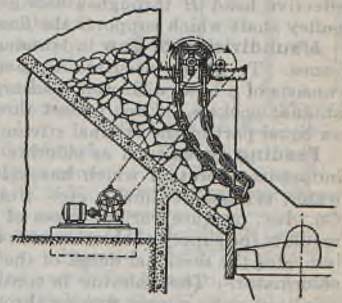


FIG. 61.—The Ross feeder. (*Ross Screen and Feeder Co.*)

The flight-conveyor feeder consists of a series of flights, vertical scrapers, or chains of suitable design, moving in a stationary trough. In that a moving pan effect is thus produced, the feeder might be considered similar in principle to the apron or pan conveyor, Fig. 57. A great variety of material can be handled in a flight conveyor by altering

Level. The measurement of the position of the interface between gas and liquid at high pressures may be accomplished by balancing the head of liquid against a column of mercury. Liquid level gages identical with the types of mercury-manometer flow meters described above are used to indicate the level of the meniscus. As the operation of these level gages in no wise differs from those used at low pressures, there is no need for describing them at length. A second very useful method consists in mounting on scales either the main receptacle or a smaller vessel connected with the main body by flexible tubing. A third method involves mounting one or more suitable conductivity cells in the vessel. These operate signal lights or electrical indicating instruments when wet with process liquid.

The level may be controlled by pneumatically or electrically operated valves actuated by a gage. This kind of control makes it possible to separate a gas and a liquid under pressure, drawing off one and recirculating the other. There is often a definite advantage in this: it saves recompression costs, the removal of a reaction product often permits the reaction to be carried further in another vessel so that the concentration of the desired product is built up, and there is less handling loss due to recirculation of unconverted material later in the process. Moreover, if 90 per cent of the gas can be removed at a total pressure in comparison with which the vapor pressure of the liquid is negligible, the vapor carried off with the remaining 10 per cent when the pressure is released will be a relatively small amount.

Pressure Drop. Pressure drops in a high-pressure synthesis system may be very significant, and simply placing two record pens on the same chart may not be sufficiently accurate or direct-reading. For moderate pressure drops the usual flow meter serves as a direct-reading pressure-drop instrument, but at higher drops range tubes 2 in. high per pound differential become unwieldy. In such cases readings transmitted from scattered static-pressure instruments can be shown on a single direct-reading pressure-drop recorder.

Safety

When reasonable precautions are taken, high-pressure plants can be made as safe as any. There are a few hazards peculiar to work at high pressures that it seems worth while to enumerate.

1. At least one fatality and several accidents are known to have resulted from split Bourdon tubes. The fatality is believed to have been the result of a rush of compressed inflammable gas into a Bourdon tube containing air. The minor explosion which occurred split the tube and a particle flying from the gage case was responsible for the death of a man. To permit the escape of gases, the cases of Bourdon gages should be provided with large vent openings covered with paper or foil for dust protection. Whenever possible the gage glass should be removed and the face left open or, for dust protection, the glass replaced with thick transparent sheeting or with safety glass so that the danger from flying particles will be avoided. Bourdon gages should be placed above eye level. In some laboratories it is required that Bourdon gages be placed behind a barrier and read only by their reflection in mirrors so that the observer is at all times out of the direct line of danger. The usual procedures should be followed, such as the use of oil seals to prevent corrosion, throttling down to prevent fatigue, and the use of built-in checks to prevent the pressure from rising or falling too suddenly.

2. With inflammable gases at high pressures, the Armstrong effect is a possible source of trouble: when a gas containing finely divided liquid or solid particles passes at high velocity over an insulated metallic object, that object

becomes electrically charged. The charge may become sufficiently great to cause a spark which in turn ignites the gas.

3. When metallic containers rupture, frictional effects may cause very high local temperatures and the ignition of inflammable gases.

4. It is rarely possible to cool a direct-fired vessel and its setting in a hurry. Such apparatus should be followed by a by-pass leading to the atmosphere so that if a failure or a plug occurs farther along in the process the by-pass can be opened and material kept running through the direct-fired apparatus while it cools.

5. Where it is not possible to use a self-seating safety valve, it is sometimes possible to have two safety valves: one set slightly above the operating pressure with a shut-off valve between it and the body to be protected, and the other set somewhat higher and without the shut-off valve. When the lower safety valve releases, it can be reseated with the shut-off valve closed, while close observation and the higher safety valve provide the necessary security.

6. A variety of pressure relief that has met with favor because of its positive action and because it cannot readily be prevented from blowing off when the pressure exceeds a certain value is the rupture disk, which consists of a plate held in place over an opening by means of flanges. When the pressure rises too high, the disk ruptures. The disadvantages of the method are that the entire contents of the pressure system are discharged, and that corrosion may so weaken the disk that unless frequently inspected and renewed it may burst at too low a pressure; on the other hand, since the working pressure produces stresses close to the yielding stress of the material of the disks, they tend to become strain hardened with continued use and may then fail to relieve the pressure at the desired point.

7. Carbon monoxide rapidly attacks pure nickel, forming nickel carbonyl, and even attacks iron, though much more slowly. When carbon monoxide is to be used, the proper selection of alloys is important. It is also necessary to warn against the use of mercury in any apparatus in which it can come in contact with copper, brass, or other metals with which it can amalgamate. The general belief that mercury does not wet steel or form alloys with it is incorrect: Bridgman found that mercury can wet surfaces made by breaking steel in mercury. This is of great importance where there is any possibility of a crack in a mercury container opening slightly under pressure, allowing the mercury to wet the steel and start to undermine the walls.

8. Some catalysts, notably very active nickel or iron powders, are highly pyrophoric. Where there might be a danger of their leaking through valves or stuffing boxes, drying out and glowing, it is desirable to provide a constant stream of water to wash them away so that any inflammable vapors or gases that accompany them may not catch fire.

9. The larger and hotter pieces of high-pressure equipment may be placed to advantage behind heavy barricades, for in case of failure, heavy walls and light roofs give protection to the surroundings. At the same time it should be urged that anything permitting the formation of gas pockets is to be avoided: the buildings in which high-pressure apparatus is housed should permit the free passage of air through all parts, especially up under the roof. Where poisonous or inflammable gases are used, additional protection for the operators should be provided by the installation of forced ventilation. This system is, however, worse than useless if the draft is too strong or its temperature too high or too low.

10. Since leaks are a serious fire or health hazard, they should be repaired as promptly as possible, especially as erosion quickly makes the leak worse and

loaded sound is heard, the device closes down the feeder. Later, when the mill begins to sound empty, the feeder is automatically started. If the "Electric Ear" is used in conjunction with a wet grinding mill, the relay in the "Electric Ear" circuit can be made to open and close a solenoid valve in the water-supply line, in addition to controlling the feeder delivering dry material to the mill. Thus a constant density of pulp is maintained in the mill and classifier circuit.

The electric eye has been used extensively for automatic batch weighing and for other purposes. The possibilities for controlling feeding devices through the use of instruments seem almost unlimited (see also Sec. 17, Measurement and Control of Process Variables).

The packaging of bulk and liquid chemicals is a highly specialized branch of feeding. By using a semiautomatic packaging machine, *i.e.*, one requiring constant attendance, it is possible to pack from 10 to 20 units per minute. With fully automatic machinery, *i.e.*, where an attendant can supervise a battery of machines, the output can be raised to as much as 125 units per minute when free-flowing material is fed into bags, bottles, cans, cartons, and sacks. Of course, to warrant the investment, a very large production is essential with fully automatic machinery. With some material, it is possible to obtain a weight accuracy of 0.05 oz. to the 1-lb. package. On this type of work, weights range from ½ oz. to 50 lb. With bottles it is customary to fill to a given height rather than on a volumetric basis. The capacities of bottles vary as much as 5 per cent and thus volumetric filling would give the consumer the impression that the bottles had not been properly filled.

BULK-PACKING EQUIPMENT

BY R. W. LAHEY

Packaging and Filling Equipment for Small Containers. There are so many machines for packaging, filling, weighing and measuring, closing, labeling and wrapping bottles, cans, boxes, bags, etc., that this is a subject in itself and will not be considered here. The Packaging Institute, which is composed of machinery manufacturers and production managers interested in small containers, is an authoritative source of information. The "Packaging Catalog," published by Modern Packaging Magazine contains worthwhile data on this subject.

Definitions. The term **bulk** as used here, excludes all so called "retail" packages and refers to the "wholesale" containers which comprise large bags of 50 lb. minimum capacity, drums, barrels, kegs, etc.

Purpose. It is the function of packaging equipment to prepare a product for shipment.

It is unfortunate that a plant engineer cannot choose equipment by a mathematical or scientific formula, but most of these problems can be solved with assurance if all the following factors receive proper and detailed consideration:

1. Physical characteristics of product.
    - *a.* Variation in specific gravity.
    - *b.* Degree of gravity flow.
    - *c.* Effect of moisture absorption.
    - *d.* Effect of temperature change.
2. Plant layout.
3. Maximum and minimum production rates.
4. Choice of container.
5. Factory storage in bulk (bins or tanks) or in containers.

In most instances these factors vary for each product and for each production installation. This requires a special consideration for every problem, and often existing machinery must be altered to fit a special situation. It is therefore difficult to supply useful information. A general statement of the known machinery and processes is probably of greater utility than any other form of presentation.

Bulk-packaging equipment is not always nationally advertised, and it is therefore impossible to be sure that the following completely covers the available equipment. There is an urgent need for the accumulation of these data by an agency or some central bureau which the engineer can consult with the assurance that he has up-to-date and complete information.

Storage in Bulk or in Containers. After choosing the container for a new product, the next question which must be settled before packaging machinery can be considered is to decide whether the product is to be stored in the factory warehouse in bulk or in containers.

If the quality of the product is not affected, it is less expensive to store finished products in bulk and pack them in containers just before shipping. It is obvious that this system requires less handling than it does to store in containers. Less storage space is required, the inventory of filled and empty containers can be reduced, and their appearance is usually better than containers which are held in storage for a considerable time. Even under the best of storage conditions, coatings on steel containers become marred, stenciling is often obliterated, and labels become dirty and torn.

Its disadvantages include additional cost of tanks or bins for storage and the possible extra cost of additional packing and filling equipment which may be required to take care of peak shipments if the business is seasonal.

In the rock products and the fertilizer industries where the products are inexpensive and extra handling costs are liable to make the difference between profit and loss, the almost universal practice is either to store in bulk or only to manufacture against shipping orders.

Storage-bin Construction for Dry Materials. The classification "dry materials" includes all degrees of particle size from the finest powder to large lumps which vary in degree of flow from free flowing to, and including, those products which will not flow. Some free-flowing materials lose this desirable characteristic on absorption of moisture in humid weather or on excessive changes in temperature.

Free-flowing Materials. Even though products are stored in containers, there must be an intermediate bulk storage just before the packing operation to absorb the fluctuations in both production and packaging.

The basic requirement for accurate weighing is to provide a uniform stream of material to the packaging equipment. Pulsating or uneven flow will seriously handicap proper functioning of packing equipment, and it is therefore of paramount importance to so design storage bins or tanks that this uniform feed will result.

Usually the best shape for a storage bin for even gravity flow of material is conical. Under any conditions, avoid square or angled corners where material has an opportunity to pack and then arch. The inside of the bin should be free from any projections, braces, or stay rods which interfere with the free movement of the product.

As the fundamental function of a storage bin is to take up the slack between manufacturing and packaging, it is obvious that the level of material in the bin will fluctuate, causing differences in the rate of flow of material.

In connecting the outlet of a storage bin to a filling machine of any kind, it is always desirable to avoid the possibility of the variation of the head load in the storage bin from directly affecting the uniformity of the flow. This is sometimes accomplished by baffles placed in the discharge spout, but more frequently by various types of offset connections and mechanical means such as are shown in the various sketches.

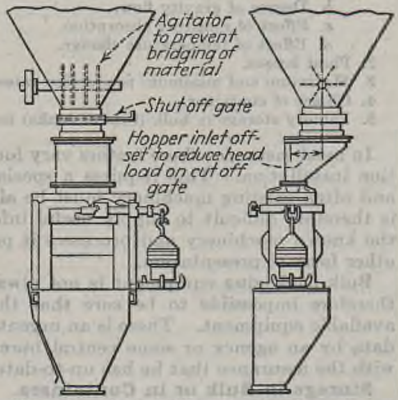


FIG. 65.—Automatic scale showing connection to bin outlet, with agitator, to prevent bridging of material.

Other Than Free-flowing Materials. This classification includes all types of dry materials from those which will arch in a bin and then flow with a surge to the type which will not flow except by mechanical assistance.

Some type of agitation in the bin or controlled feeding to the packaging machine is required. The type material which arches and then surges can probably be handled by agitation in the bin to prevent arching, while the strictly non-free-flowing type will require a mechanical feeding attachment between the bin and the packer which will regulate and mechanically convey the proper amount of material. Manufacturers of weighing and bagging equipment have devices for regulating flow of materials, and they should be consulted on this problem.

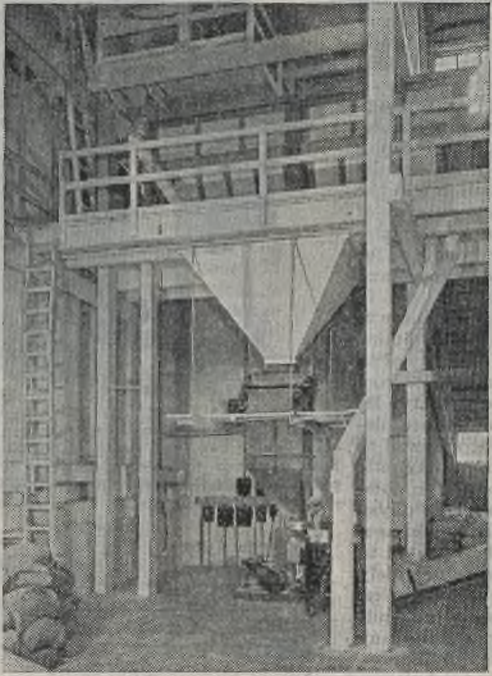


FIG. 66.—Richardson enclosed dustproof sacking scale weighing ground material from overhead storage bin. Scale is fitted with spike type of agitator in feed chute to keep material moving freely from bin to scale.

Weighing. In the bulk packaging field there are generally two systems of weighing in use.

Net Weighing. As the name implies, this system weighs the material before it is packed into the container. The product flows into a bucket or hopper which is attached to a balanced scale beam. When the bucket is filled, the movement of the scale beam actuates a shutoff to stop the flow of material. There is a gate for discharge at the bottom of the bucket which, when opened, either automatically or manually dumps the material into the container.

Gross Weighing. The bag is attached to the weighing mechanism, and a shutoff gate is closed when the desired amount has flowed into the container. This shutoff gate is operated either automatically or manually.

There are other measuring systems such as the volume method and a combination of the volume and weighing method, but these are largely used for small retail packages and need not be expanded here.

Weighing and Bagging Equipment. *Automatic Scales.* The choice of scales depends on the rate of production and the physical characteristics of the product. Automatic scales are best adapted for volume production, and they can be arranged in batteries of two or more to take care of large volume. The bagging capacity of an automatic scale varies from five to ten bags of 100 lb. capacity per minute depending on the material to be bagged. Automatic scales are built for commodities of specific physical characteristics and usually cannot be adapted to commodities of widely different characteristics. These scales can be built to handle practically any type of dry material.

Automatic scales are often used in conjunction with equipment for packing products into containers such as valve bags where the opening is small and some device must be used to carry the material through the valve into the bags. Another instance of this is the use of auger packers which are used to pack material which must be compressed.

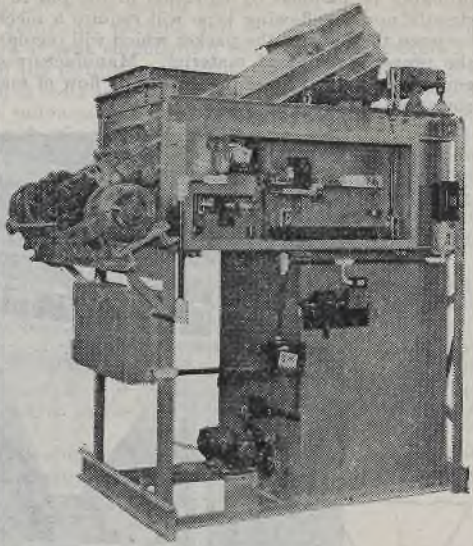


FIG. 67.—Richardson Duo-Screw feed bulk weighing scale having two feed screws, each fitted with special flush governor to control delivery of powdered or pulverized materials from bin to scale weigh hoppers.

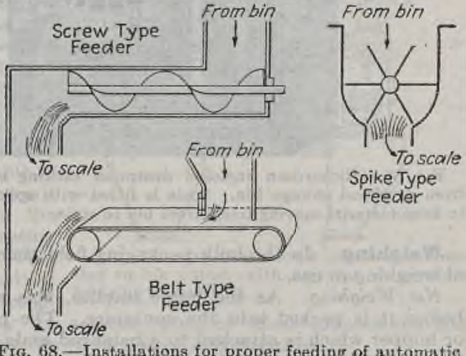


FIG. 68.—Installations for proper feeding of automatic scales.

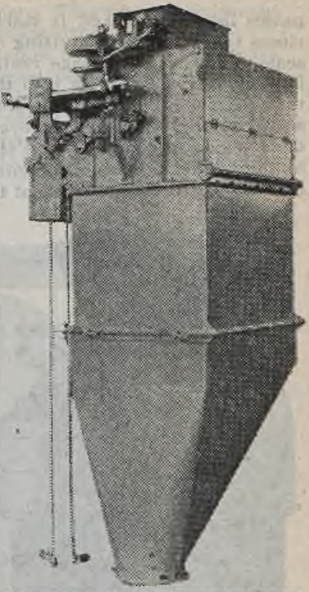


FIG. 69.—Richardson dustproof enclosed type of sacking scale for weighing powdered, ground, and dusty materials.

The illustration shows a Richardson net-weighing, automatic-bagging scale for granular and ground materials. It has a power feed with dustproof housing, and the principal operating levers and other parts are mounted outside the casing to be away from dust, etc. There is an agitator in the feed chute, the style of which is varied to suit the material. The agitator is connected with the feed gate which stops agitation while the feed gates are closed. It has a hand pull and an automatic discharge mechanism. The manufacturer claims that it will weigh batches of 50 to 300 lb. as required and, depending on the free-flowing characteristics of the product, will pack from 1 to more than 12 bags of 100 lb. capacity per minute.

The hopper shown at the top of the photograph of the Hoepner scale is connected to the outlet of the bin discharge. The bucket itself is suspended on scale beams in the usual manner. This intake hopper is equipped with two openings, one a large size intended for the flow of the bulk stream and the other a small one intended for the dribble stream. At each of these openings there is a gate operated by the bucket. The bottom of the scale bucket is equipped with a gate which likewise is controlled by the movement of the bucket itself.

With the bottom gate closed and the top gates open, the flow of material passes into the bucket. When about 90 per cent of the load to be weighed

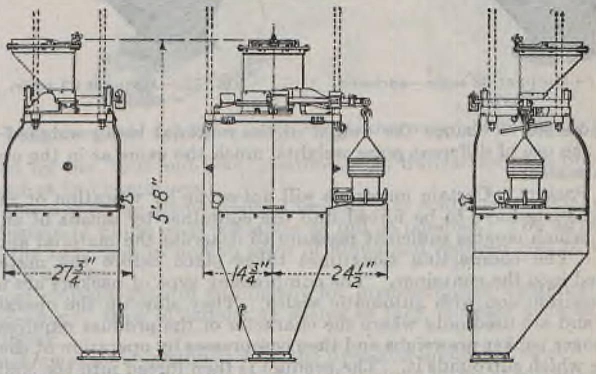


FIG. 70.—Hoepner 02 scale—enclosed type.

passes into the bucket, it starts to descend, and the start of this descent closes the main gate, shutting off the flow from the larger opening into the scale. The small passage remains open, permitting a very much reduced flow of material to pass into the bucket, and the continued weight of this material passing through the secondary opening, causes the bucket to descend a still greater distance, and eventually trips a mechanism which causes the dribble gate to be closed. At the same time the discharge gate at the bottom of the bucket is opened, permitting the material to flow into the container beneath the discharge spout of the scale.

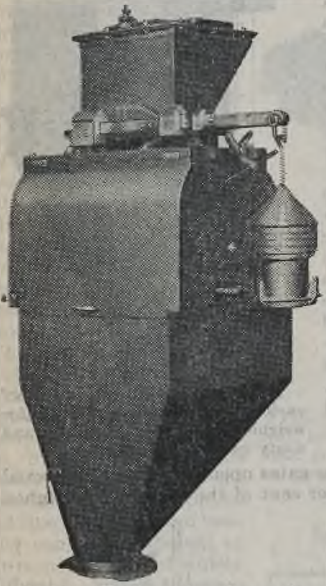


Fig. 71.—Hoepner 02 scale—enclosed type.

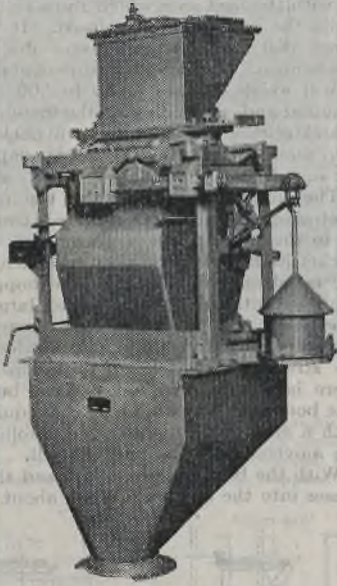


Fig. 72.—Hoepner 02 scale.

It is possible to change the weight of the material being weighed in the scale by the use of different poise weights, much the same as in the ordinary platform scale.

Auger Packers. Certain materials will not settle by vibration or shaking. These products have to be forced into the container by means of an auger or screw which creates sufficient pressure to deaerate the material and compress it. The compacting sometimes takes place before the material is discharged into the container. The compressing type of packers are usually used in conjunction with automatic scales. They slow up the operation of packing and are used only where the character of the product requires it.

This auger packer preweighs and then compresses by operation of the auger in a tube which surrounds it. The product is then forced into the container.

There are other types of compressing packers devised to meet special conditions; for instance, there is a packer which weighs the product after compressing it and before packing in the container. This type of equipment is not satisfactory for products which tend to aerate after compression.

Hand Weighing. In this operation the container is attached to a scale, and the operator closes the hopper's filling gate as the container is filled to the desired height or weight. The accuracy of this operation depends upon the skill of the operator but, on the average, is not as accurate as automatic weighing. Often the container is filled to the approximate amount required, and then the container is transferred to a platform scale where the weight is adjusted by the "put-and-take" method. Additional speed is obtained, but it requires two operators.

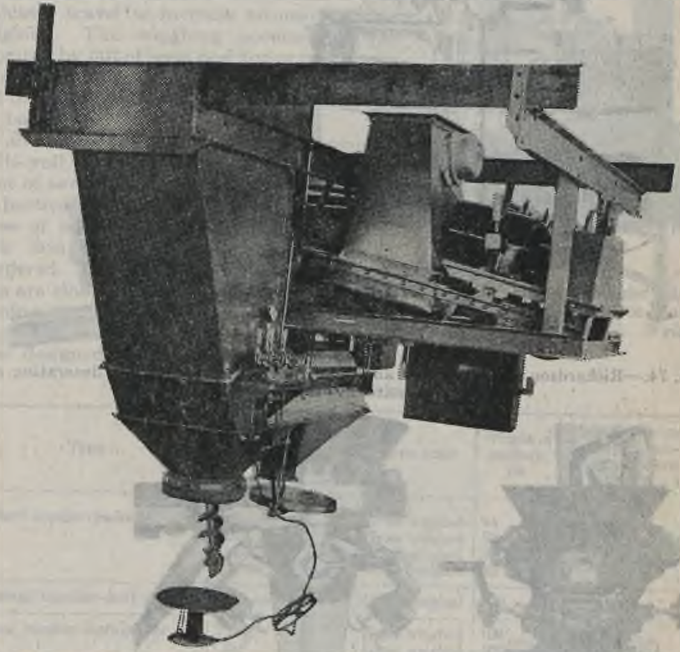


FIG. 73.—Richardson scale and Risco packer for packing material into bags or drums.

The practice of filling predetermined weights into containers is fast replacing the custom of filling each container as full as possible. The advantage of this is the reduction in human error of filling and reduced cost of record keeping, billing, etc.

The most efficient type of equipment for predetermined weighing is the "over-and-under" type of indicating scales as they increase accuracy and speed of filling.

This sacking scale is equipped with the over-and-under indicator. It will handle loads up to 200 lb. at a rate of speed of three to six bags per minute. It is designed to fit directly under a storage hopper outlet and occupies a total

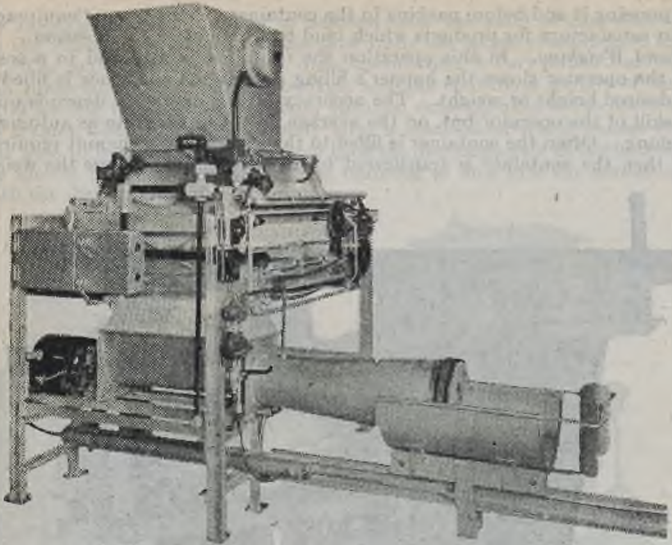


FIG. 74.—Richardson conveyometer and Risco horizontal packer for deaerating and packing material into paper bags.

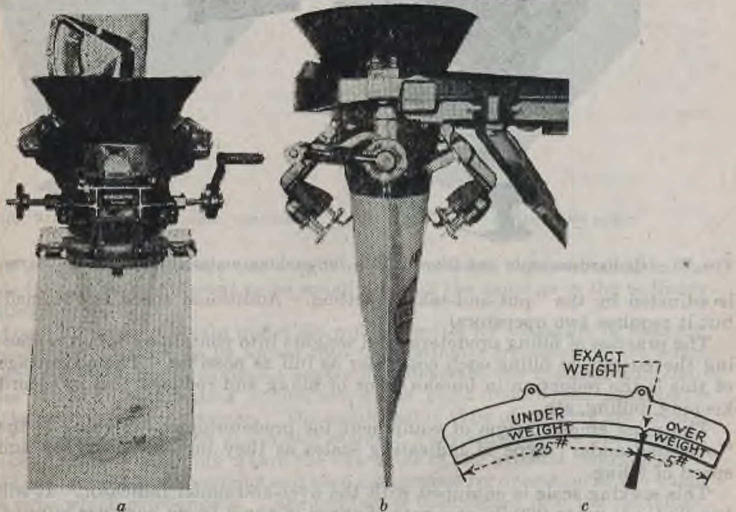


FIG. 75.—Exact Weight sacking scale. *a*, front view; *b*, side view; *c*, over-and-under indicator. (*The Exact Weight Scale Co.*)

space of 48 by 21 by 58 in. It can be floor mounted, swing mounted, or suspended. This scale is equipped with a dust-tight bag holder, elliptical in shape, which allows for retention in shape of bag—an important consideration when closing open-mouthed paper bags by machine sewing.

FIG. 76.—Exact Weight platform scale for check or hand weighing. (*The Exact Weight Scale Co.*)

This platform scale is also equipped with an over-and-under indicator which has a long indicator travel to increase accuracy in fast weighing. The weighing accuracy is not impaired by out of level position or machinery vibration. All working parts are enclosed, and the rail attached to the platform supports the bags.

Valve-bag Packers. The introduction of multi-wall paper bags has led to the development of several filling and weighing machines of particular interest. There are several types of equipment which will weigh and pack into valve bags a wide variety of powdered and granular products. Valve bags are closed at both ends save for a small opening or "valve" in one corner through which the material is injected. This valve is so designed as to be self-closing due to

Valve Bag Packers

Type	Typical products	Type scale	Weight of package, lb.	Approximate production, one operator, tons per hr.
Standard impeller packer	Cement Lime Limestone Plaster	Gross weighed Gross weighed Gross weighed Gross weighed	94 50 50 to 100 80 to 100	25 to 70* * * *
Horizontal impeller-shaft packer	Soda ash	Gross weighed	100	15 to 20*
Vertical impeller-shaft packer	Talc Clay Bicarbonate of soda	Gross weighed Gross weighed Gross weighed	100 100 100	10 to 15*
Belt packer	Alum Bicarbonate of soda Granular soda ash	Gross weighed Gross weighed Gross weighed	100 100 100	25 to 40 on 100 lb.* two tube
Belt packer	Calcium chloride Sugar Salt Charcoal Alum Dog food	Preweighed Preweighed Preweighed Preweighed Preweighed Preweighed	100 25 to 100 25 to 100 25 to 100 25 to 100 25 to 100	15 to 25 on 100 lb.* two tube
Tapered screw packer	Flour Calcimine Starch	Preweighed Preweighed Preweighed	24 to 98 24 to 98 24 to 98	8 to 10 on 98 lb.*

[*] Production varies dependent on type of installation, size of bags being packed, physical properties of product, etc.

the internal pressure of the contents as the bag is withdrawn from the filling tube. For granular products the valves are equipped with paper "sleeves" which, folded manually, prevent sifting.

Machines used for the filling of valve bags are of the impeller, belt, and screw types and are equipped with either preweighing or gross weighing scales. The tabulation p. 2301 shows the available types of valve bag filling machines with their rates of production.

The standad impeller-type machine consists essentially of a continuously running shaft equipped with feeder blades which, mounted in a housing, force the material through the filling tube into the valve bag.

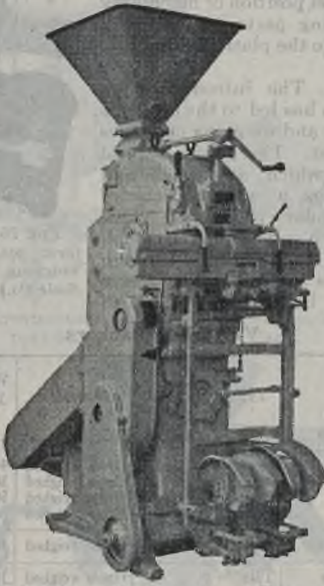


FIG. 77.—Shifting-tube belt packer.

The other impeller-type machines (horizontal and vertical shaft) are similar in principle, the differences being in the arrangement of the impeller blades and that the impeller shaft is started and stopped at the beginning and end of the filling cycle for each bag rather than turning continuously.

The belt-type machines are used primarily for granular products and especially for crystalline products whose physical characteristics may be changed in passing through an impeller-type machine (see Illustration).

With the belt-type machines the packing principle is as follows: The material flows or is fed from a hopper over the packing machine through an inlet leading to a deep groove in a rotating pulley to which it is confined by an endless belt pressing against the outer flanges of the pulley. As the pulley rotates (at a peripheral speed of from 1500 to 2100 ft. per min.), the material is pressed against the belt by centrifugal force and is rapidly accelerated.

After rotating through an arc of 90 deg., the material is discharged at a tangent to the pulley through the filling tube and into the bag.

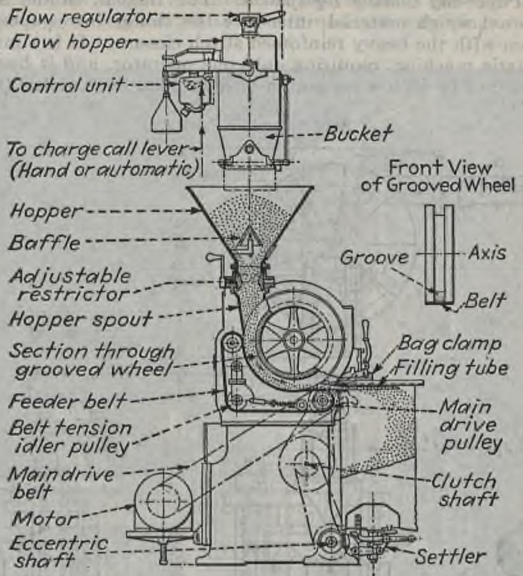


FIG. 78.—St. Regis belt packer.

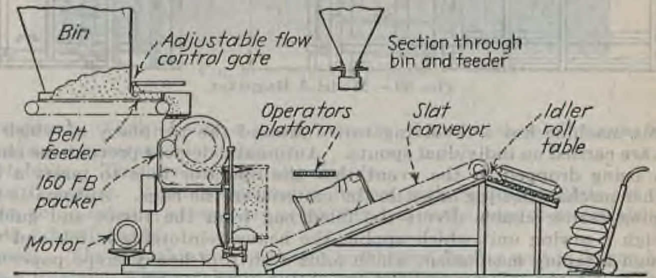


FIG. 79.—Typical belt-packer installation with belt feeder.

On the two-tube gross weighing machines of the belt type, a settler arrangement is provided to agitate the bag and settle the contents during a part of the filling cycle. On the preweighing machine of the belt type the settler provides such agitation during the complete filling cycle. Belt-type machines are ordinarily equipped to pack 100-lb. bags but special attachments are furnished for the filling of 25-, 50-, and 80-lb. bags as well.

Bag-closing Equipment. *Combined Automatic Scales and Open-mouthed Multi-wall Paper-bag Closing Equipment.* The Bagpak Model A has been designed to net weigh material, introduce the charge into paper bags, and to close them with the heavy reinforced stitch closure. It is a multi-station fully automatic machine, requiring only one operator, and it has a varying production capacity with a maximum of over 1000 bags of 100 lb. capacity per hour.

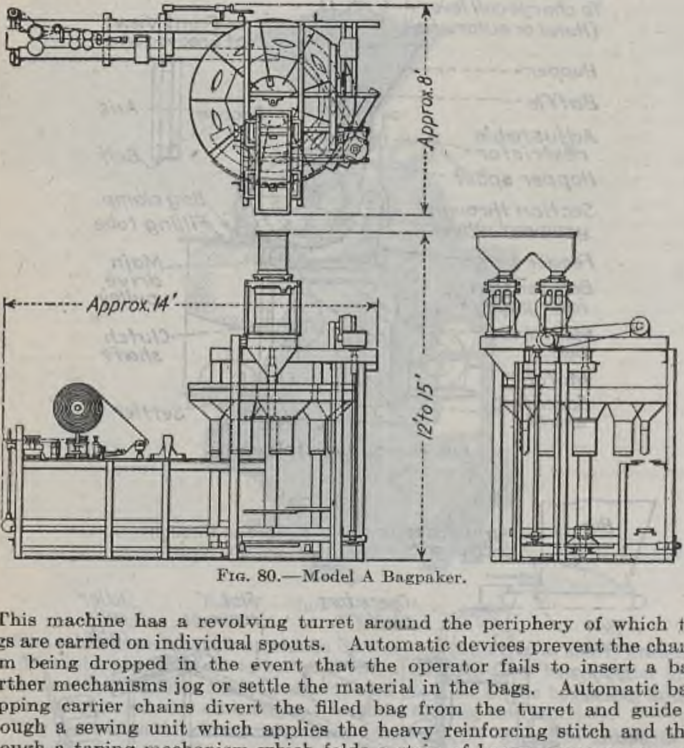


Fig. 80.—Model A Bagpaker.

This machine has a revolving turret around the periphery of which the bags are carried on individual spouts. Automatic devices prevent the charge from being dropped in the event that the operator fails to insert a bag. Further mechanisms jog or settle the material in the bags. Automatic bag-gripping carrier chains divert the filled bag from the turret and guide it through a sewing unit which applies the heavy reinforcing stitch and then through a taping mechanism which folds a strip of heavy crepe paper over the end and glues and presses it into place. The sewing unit and taping mechanism are mounted over a straightaway bag conveyor at the discharge end of the machine. This conveyor delivers the filled and closed bags to take-away conveyors or hand trucks.

Sewing Unit (Fig. 81): Hoepner No. 150 heavy-duty type, gear driven from main drive shaft. Sewing head equipped to apply various types of double or single stitch seam, including the Cushion Stitch Closure.

Conveyors. Circular steel roller conveyor underneath turret—slat type on the straightaway, separate motor driven.

Controls. Continuous operation motivated by two control switches.

Scales. This machine is equipped with two or three scales, dependent upon the production speed required. These are the Hoepner automatic bucket scales. On such materials as are not readily free flowing, scale feeders are required. They ensure a steady stream of material at all times, thus preventing inaccuracies in weighing.

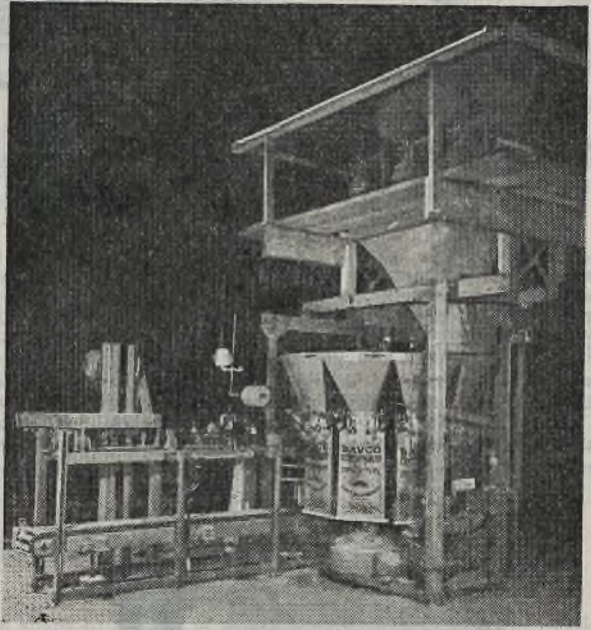


Fig. 81.—Model A Bagpaker.

Portability. This machine is not portable.

Dust Collectors. Dust hoods and collectors with suitable outlets for connection to an exhaust system are recommended in the case of materials that tend to "dust" easily.

Motors. The straightaway conveyor and entire closing unit is driven by a $1\frac{1}{2}$-h.p. motor. The revolving turret and settling devices are driven by a 1-h.p. motor. Scale feeders, when required, are driven by a motor the size of which is dependent upon the material handled but in any case will not be over 5 h.p. All motors are of dustproof type.

Bag Holders. The problem of holding bags on automatic and manually operated bag hoppers often requires special equipment. The round hopper with a holding device, usually a ring, is most often used for textile bags. Sometimes the hopper operator merely holds the textile bag over the lip of the hopper with one hand while the bag is filling.

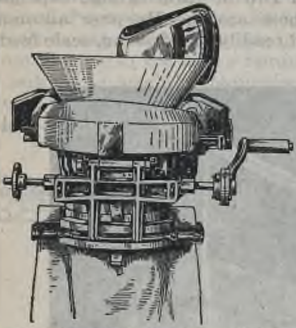


FIG. 82.—Dustite bag holder. A conveniently placed spring clip holds the bag preparatory to clamping and the entire bag opening is completely closed and held tight in one operation.

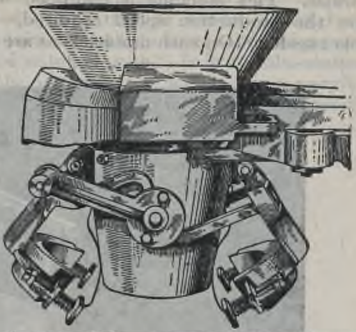


FIG. 83.—Side view of Dustite bag holder with bag removed.

The most desirable type of bag holder for paper bags is oval in shape, and, where the bag must be suspended above the floor, grippers are used as shown in Figs. 82. and 83. It is important to preserve the oval contour of paper bags if they are to be closed by machine sewing or difficulties will be encountered in that operation.

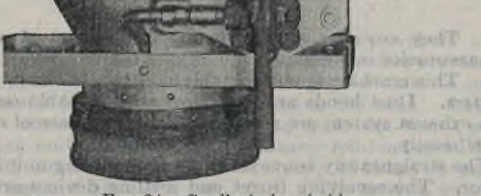


FIG. 84.—Swellgrip bag holder.

Recently, a new type of container holder has been introduced which holds the container by means of inflating a rubber tube that is attached to the periphery of the bagging spout. It uses 10 to 15 lb. of air pressure and one valve inflates and exhausts the air. It is adjustable for different sizes by adjusting a pressure regulator. This device is available in diameters from 6 to 16 in. in two widths. The smaller width (3¼ in.) handles bags varying not more than 2½ in. in width and up to 100 lb. in weight. The large size (8 in. wide) handles bags that vary 6 in. in width and up to 400 lb. in weight.

This holder reduces escape of dust and, by suspending the bag from the top, it allows filling without voids.

Liner Insertion. The insertion of crepe-paper liners into bags appears on the surface to be a very simple operation, but, if the paper liner is to be of any value, it must be properly and carefully inserted. Bearing in mind that the function of the liner is to prevent sifting and to eliminate contamination of the product by the container, it must depend for its strength on the container. This, therefore, requires that the liner be inserted to conform completely to the contour of the outer container or liner breakage will result from the filling operation or the abuses of handling or transportation. The illustration shows a simple and efficient table to be used for bag liner insertion. In this instance the inserting boards should be made adjustable so they can be accommodated to bags of various sizes. The lined bag should be removed by grasping at center of the bottom of the bag, thus forcing paper liner into corners of bag.

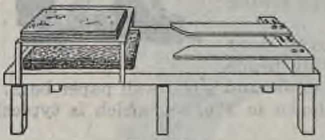


FIG. 85.

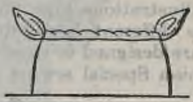


FIG. 86.

When the container is filled, the bag liner should be folded or rolled down before closing the bag, being sure that there is enough slack in the liner to conform in shape to the top of the bag after closing, otherwise shifting of the contents will rupture the liner. The liner should never be closed with the bag proper either by tying or sewing, as it is impossible to make a combined closure without setting up stresses which will result in liner ruptures.

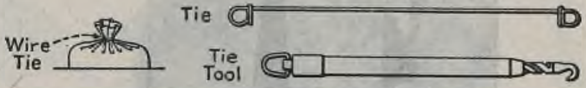


FIG. 87.

Sleeve-valve Bags. The valve bag is described under the heading of Weighing and Bagging Equipment. This container is preclosed at the bag factory. Sometimes these containers are equipped with sleeves to prevent the slight sifting that occurs through the valve from finely powdered products. This sleeve, or paper tube extension of the valve, must be tucked into the valve hole by hand after the bag has been filled.

Hand sewing for textile bags only. The mouth of the bag is rolled down (Fig. 86) and hand-sewed with a cooper's needle and heavy jute twine.

Wire tying for textile and paper bags. The bag mouth is gathered by pleating and wire tie applied by a twisting tool (see Fig. 87).

Machine Sewing. Both textile and paper bags may be closed by machine sewing. There are many ingenious installations of the sewing machine which have been devised to meet a great variety of production conditions, varying from a conveyorized production line (taking the output of one or more automatic scales) to the small volume operations where one operator fills, check weighs on a platform scale, and closes the bags.

Large textile bags are sewed with either the single-thread chain stitch or the two-thread, double-locked stitch. Either stitch can be used on paper bag , but the insertion of a filter cord on the top side of the stitch is recommended to prevent sifting through the needle punctures. This feature does not complicate the operation of the sewing head.

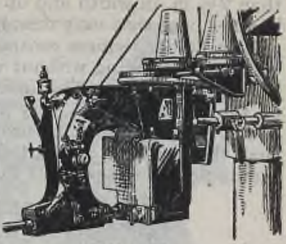


FIG. 88.

The illustrations Figs. 88, 90, and 91 show the two different heavy-duty sewing heads which are designed to close heavy textile and multi-wall paper bags.

A Union Special sewing head, shown in Fig. 88, which is typical of the heavy-duty class 80600, is used for sewing textile and paper bags. The uses for special styles are as follows:

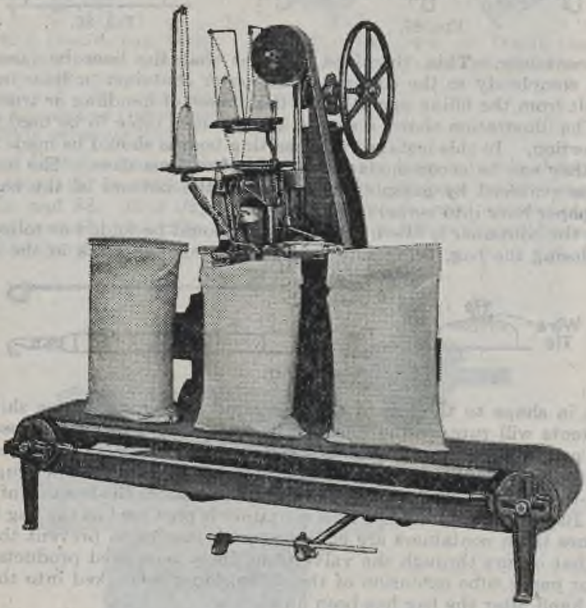


FIG. 89.—This illustrates the Union Special head mounted as described under style 80600-H.

80600 E— For closing paper bags using a filter cord guided under the presser foot to seal needle punctures. Produces double-locked stitch. Stitch range 4 to 3 per in.

80600 F— Produces single-thread chain stitch—otherwise same as style 80600 E.

80600 H— For closing multi-wall paper bags, using a crepe-paper binding and simultaneously inserting a filter cord. Machine is equipped with an automatic mechanical tape clipping device; it produces the double-locked stitch.

80600 AC—This machine is recommended for closing bags made from heavy-weight fabric. It produces the double-locked stitch.

80600 AD—Same as style 80600 AC except it makes the single-thread chain stitch.

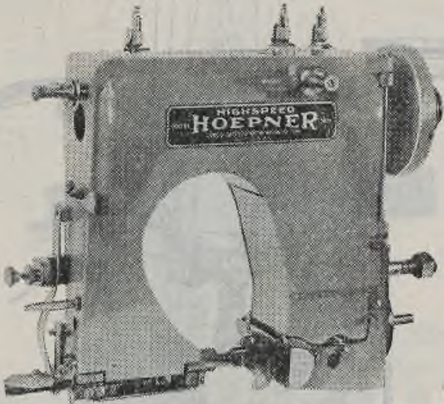


FIG. 90.—Hoepner sewing head.

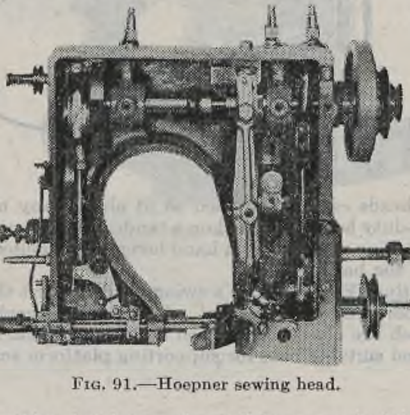


FIG. 91.—Hoepner sewing head.

The Hoepner sewing head uses the well-known principle of two threads forming a chain stitch. These heads have been particularly designed to meet the rough requirements usually found in plants closing bags as large as 100 lb. or greater. The mechanism of the head is practically entirely enclosed to protect it from the dust and dirt which are invariably found in operations of closing bags of this kind.

The head is mounted on either a pedestal alone or on a pedestal over a conveyor, and it can be arranged to run continuously or can be started and stopped as the operator requires by merely depressing a treadle. Some of these heads are operated as high as 3200 r.p.m.

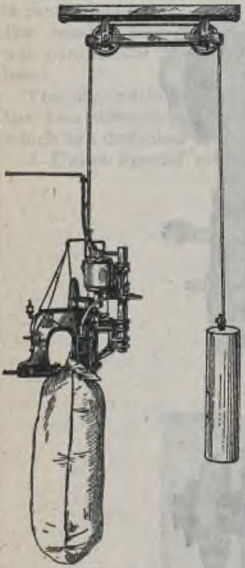


Fig. 92.

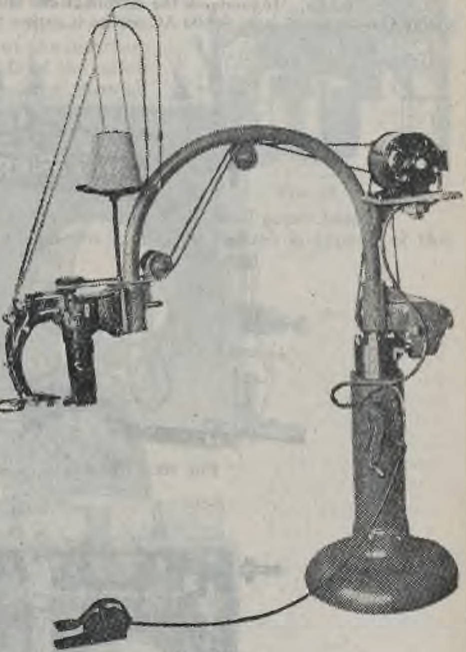


Fig. 93.

These sewing heads can be mounted to fit almost any need. In Fig. 92 is shown a heavy-duty head, mounted on a tandem pulley, cable, and counterweight. The machine operates by a hand lever and is pulled by the operator across the top of the bag.

Another mounting (Fig. 93) is on a swing-head support that travels across the bag and is operated by foot control. There is a convenient mounting for closing bags which are check-weighed on a platform scale. Figure 94 has a traveling head and suitable base for supporting platform scale.

For larger production and more efficient sewing, the head is mounted to synchronize with a belt or slat conveyor which carries the bag mouth through the stationary sewing head (Figs. 95, 96, and 97). If bags are to be hand-trucked, it is desirable to run bags upon inclined conveyor for delivery to handler at waist height.

FIG. 94.

The bag-closing machinery in Figs. 98 and 99 has been specially provided with special taping devices for closing multi-wall paper bags by several equipment manufacturers. These devices vary somewhat in detail, such as sewing through the creped tape or taping over the sewing, and some of these machines

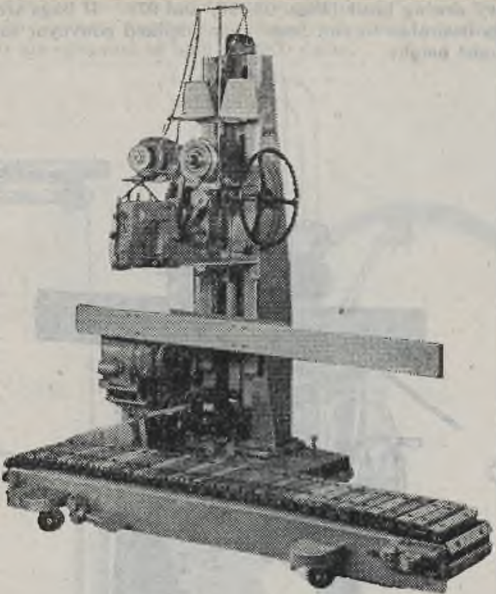


Fig. 95.

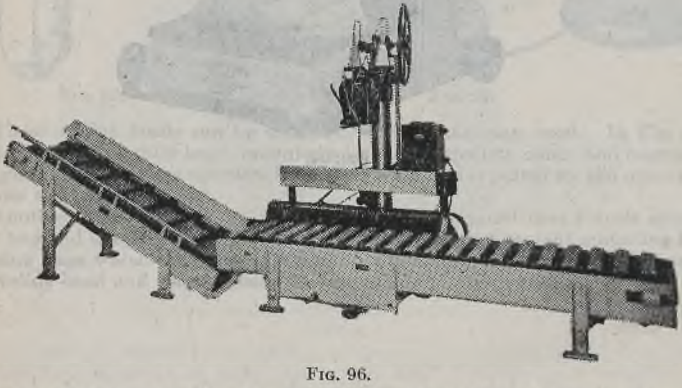


Fig. 96.

automatically cut the tape. In every instance but one the bag is guided by hand through the machine. In this latter case the bag is placed by hand

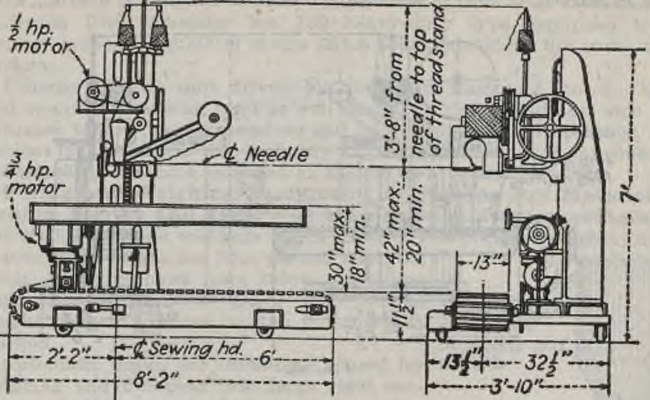


FIG. 97.

on the conveyor, a chain is then lowered to hold the bag mouth, and the operator steps on the treadle. The machine moves only far enough to clear the space where the bags are loaded on the conveyor.

The D. A. Bagpaker, which will serve to illustrate this class of paper bag-closing equipment, sews or sews and tapes over the sewing as desired (see Figs. 98 and 99).

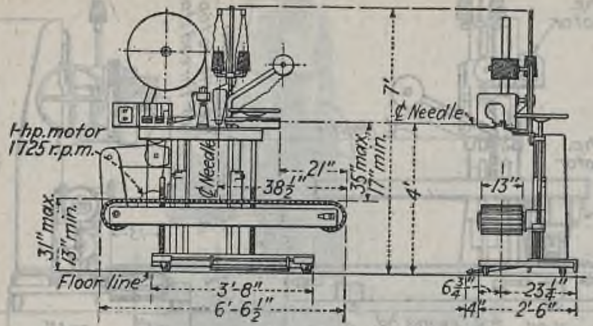


FIG. 98.

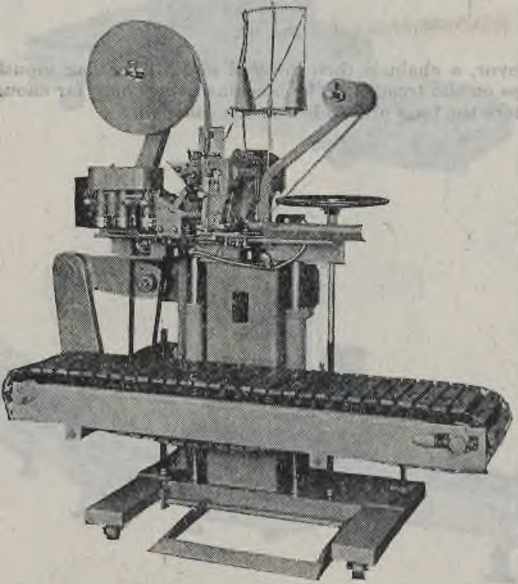


FIG. 99.

This machine consists of a sewing unit and a tape applicator, mounted in a fixed position over a bag conveyor which is adjustable vertically for various bag lengths, and is designed for use in plants that already have weighing and filling facilities and where only the bag-closing equipment is desired. This machine is designed to handle paper bags of from 25 to 100 lb. capacity.

Sewing Unit: Hoepner No. 150 heavy-duty type, equipped to apply various types of double or single stitch seam, including the cushion stitch closure.

Conveyor: Entire unit driven by one 1-h.p. dustproof motor. Starting and stopping of sewing head as well as operator of thread and tape cutters actuated by the bags themselves and fully automatic. Depression of convenient foot pedal will stop instantly all parts of machine for any emergency.

Portability: Machine mounted on casters and readily portable.

Filling and Weighing Equipment for Packing Dry Materials into Drums, Barrels, and Kegs. Container economy often requires agitation of barrels, large drums, and kegs during filling. This prevents weighing until the approximate weight has been packed when the container is removed from below the filling hopper to a platform scale. The correct weight is then adjusted by the put-and-take method.

Automatic scales are occasionally used for weighing and dumping into large rigid containers, but the speed of these devices often makes it difficult to pack, as insufficient time is allowed for settling. The best plan in using automatic scales for barrel filling is to divide the weight to be packed into two or more dumps of the scale and have the container agitated throughout the operation. The layout of Fig. 100 shows an agitator settling the load at the filling spout. After the approximate weight has been packed, the container is moved by roller conveyor to check-weigh scale and then removed to warehouse.

Liner Insertion. The same general principles for liner insertion described under Bags apply for the proper insertion of liners into kegs, barrels, drums, and boxes, but the methods of application are different because in this instance we are dealing with rigid containers. The best and most commonly used method is to draw the liner over a form which is of the proper size for the container to be lined and then insert the form in the container. The liner is pulled down over the sides of the rigid container and the form removed. Figure 101 shows proper lining of a barrel.

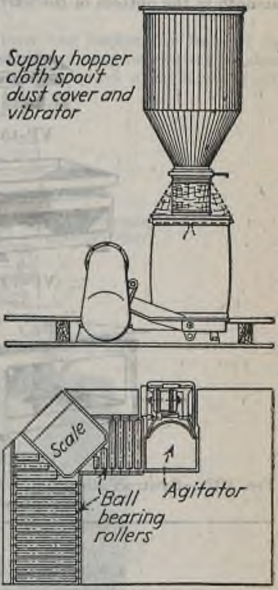


FIG. 100.

After filling, the liner is folded down in the top of the rigid container with sufficient slack to conform to the shape of the closed container.

Agitators. Vibrators are used during the packing of bags, barrels, and drums for quick settling of the products, thereby effecting economies by reduction in container size. Most vibrators are adaptable for use with barrels, drums, or bags.

The Syntron vibrator, shown in Fig. 102, consists of a vibrating platform with the vibrator mounted under the cushioned top plate. The motion is

FIG. 101.—Method for inserting creped paper liners into barrels. (1) Insert the device into the lining as far as it will go. (2) Fold inwardly the ears formed by this operation and insert the lining and device into the barrel. (3) Fold the lining down over the chime of the barrel and remove the device. (4) The bottom of the lining lies smooth in the bottom of the barrel.

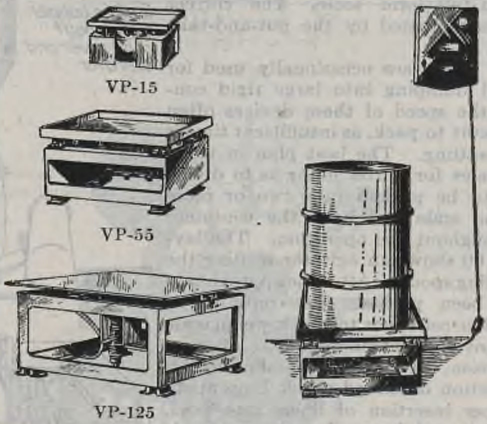


FIG. 102.—Syntron vibrators. At the right is shown a vibrator for drums. Specifications below are for the vibrators at the left.

Specifications	Models		
	VP-15	VP-55	VP-125
Capacity	50 lb.	300 lb.	750 lb.
Deck demensions	11 by 11 in.	20 by 20 in.	30 by 30 in.
Over-all height	6 in.	11½ in.	15 in.
Net weight (packing machine)	30 lb.	135 lb.	275 lb.
Shipping weight (machine and controller)	50 lb.	200 lb.	350 lb.

horizontal in direction at the rate of 3600 vibrations per minute. This is accomplished through the use of a thermionic valve that changes alternating current to pulsating waves with a time interval between.

The Vibrox packer (Figs. 103 and 104) has an oscillating and vibrating motion somewhat similar to the figure 8. It comes equipped with a 1-h.p. motor and should be installed at floor level to prevent lifting of containers.

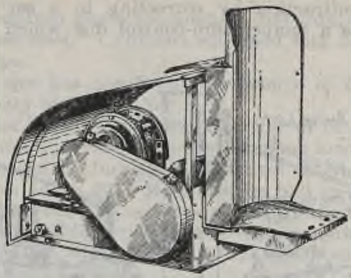
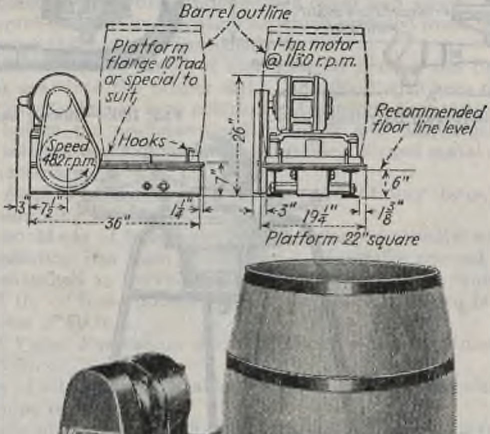


FIG. 103.—Motor-driven single Vibrox. The Vibrox bag packer can be used for packing chemical fertilizers, dry chemicals, dairy feeds, dairy mashes, poultry mashes, linseed meal, cottonseed meal, molasses feed, alfalfa meal, and all other kinds of soft feeds and similar materials that are packed in bags.



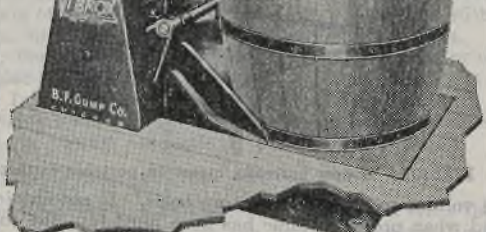


FIG. 104.—Vibrox barrel agitator.

Filling and Weighing Equipment for Packing Liquids into Drums or Tight Barrels. Liquids are measured either by weight or by volume.

Volume Method. When using the volume method, it is necessary to adjust for expansion or contraction by correcting to a set temperature. The illustrated meter has a temperature-control dial which can be quickly set and corrects volume to a set temperature basis. It automatically stops the flow of liquid when proper volume has been filled into the container. One man can perform all duties: rolling empty barrel into position, filling, marking, and rolling the filled barrel away. See Figs. 105, 106, and 107.

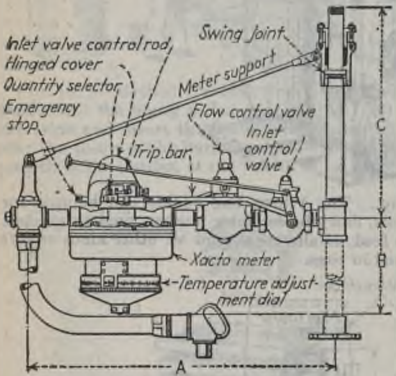


FIG. 105.—Bowser barreling meter.

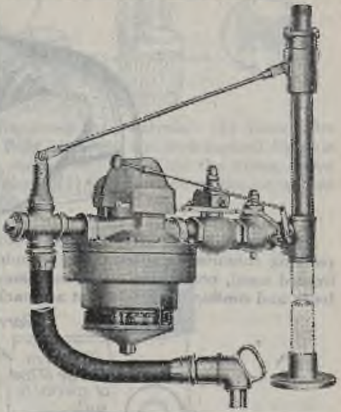


FIG. 106.—Bowser barreling meter.

FIG. 107.—Bowser barreling meter on portable carriage.

Sizes: Available with either a 1½- or with 2-in. meter. Both sizes are equipped with the temperature-control dial for compensating measurement to a 60° basis.

Construction: All exterior parts are built of cast iron; interior parts made of bronze and stainless steel. Cylinders equipped with bronze liners; pistons are leather-sealed type.

Capacity: 1½-in. size has a maximum capacity of 50 gal. per minute. 2-in. size has a capacity of 110 gal. per min.

Pressure: Both sizes have a maximum capacity of 50 lb. In most cases, however, the maximum flow can be secured at much lower pressure.

Range: Either size can be furnished to deliver predetermined quantities in single gallons from 14 to 60. May also be had in the same range for imperial gallons. However, the meter can be furnished to deliver only *all* United States gallons or *all* imperial gallons—not a combination.

Continuous Counter: Records in gallons to 1,000,000, and repeats. Cannot be set back.

Temperature-control Dial: All models are equipped with temperature-control dial for compensating measurement to a 60° basis. Glass covered, metal holder furnished for temperature adjustment chart. Holder attached and held in place by inserting under meter cover cap screw. The specific gravities of liquid and the temperature at which barreled must be specified in order that proper chart can be furnished.

Quantity Selector: Gears made of steel, set by sliding the bar connected to the selector gear into the proper slot. Equipped with zero positioner for use in setting mechanism in zero position in the event that previous delivery was stopped before the predetermined quantity was discharged. Counter and quantity selector mechanism enclosed under a hinged metal cover, easily raised when necessary.

Emergency Stop: Permits flow to be stopped at any time. Especially desirable for sampling, etc.

Control Valve: Made of brass; shuts off flow automatically when the predetermined quantity has been discharged. Manually opened by pushing knob on rod attached to valve. Single-stage type for use where inlet line pressure is 25 lb. or less—double-stage type where pressure is over 25 lb. but not in excess of 50 lb.

Flow-control Valve: For use on all installations where the inlet pressure is in excess of 25 lb.

Hose: Made of oil-resisting, pliable synthetic rubber. Furnished in same size as meter pipe connections.

Nozzles: Made of bronze. Equipped with built-in check valve to prevent dripping. Shuts off instantly when flow stops. Furnished with two tips—1⅝ and 2⅛ in. for filling barrels with small or large openings.

Swing Joint: Permits unit to be turned in a 360-deg. arc for convenience. Connects to inlet pipe line.

Connections: Inlet: Furnished in 1½- or 2-in. pipe sizes.

Discharge: Same size as inlet. An L-type connection permits hose to be attached to end or bottom connection. Pipe plug included.

Additional Equipment Obtainable.

Strainer: For removing dirt and other foreign matter from the liquid.

Air Release: To remove air from the liquid, preventing inaccuracies in measurement. Assists in absorbing impact pressures. Necessary when liquid is discharged by pump pressure.

Weight Method. Where random weights are used, the container is filled to the desired level and then check-weighed. The container is either filled on a scale or is placed on a scale after filling. Where predetermined weights are used the container must be filled on a scale.

FIG. 108.—Vol-U-Meter valve.

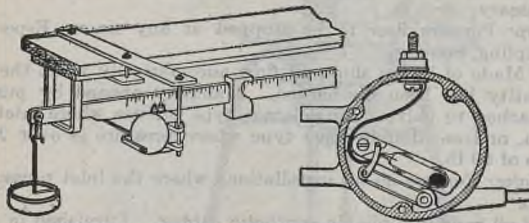


FIG. 109.—Vol-U-Meter mercury switch. A mercury-tube cut-off switch placed at the extreme end of the scale beam automatically closes the valve the instant the beam rises. This switch does not interfere with the sliding beam weight and can be moved vertically to vary the closing time of the valve. At the right is shown an interior view of mercury-tube cut-off switch in "Off" position. Line is opened or closed by action of mercury bath flowing over terminals sealed into end of glass tube. All arcing takes place inside a sealed glass tube.

The Vol-U-Meter is an electrically controlled valve (Fig. 108) for filling containers over 5 gal. capacity which can be operated from a light socket. This equipment operates by means of a mercury-tube, cut-off switch (Fig. 109), placed at the extreme end of the scale beam which automatically closes the valve the instant the beam rises.

The complete Vol-U-Meter unit includes:

1 valve.
1 telescopic spout, or

1 no-drip spout, or
1 flexible hose connection.
1 mercury-tube cut-off switch.
1 transformer 100 or 220 volt if alternating current is used, or
1 resistance coil if direct current is used, or
1 hot shot battery if no other current is available.
1 yoke for attaching cut-off switch to scale.
1 snap switch to cut off current when unit is not in use.
1 coil of rubber-covered wire (15 ft.).

Valve Specifications:

Intake 2 in., discharge 1½ in.
Spout plain or no-drip.
Valve dimensions 14 in. high, 5 in. wide, 8 in. deep.

Metal:

Bronze, cast-iron, or aluminum valves are standard.
Special alloys can be obtained.

There are several efficient installations of this equipment; those illustrated in Figs. 110 to 112 are the most important.

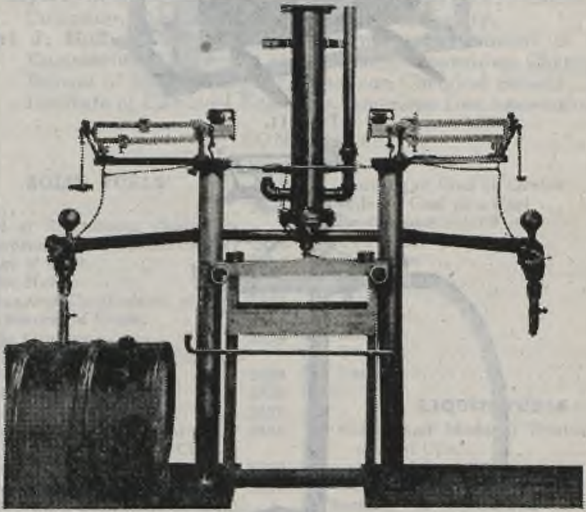


FIG. 110.

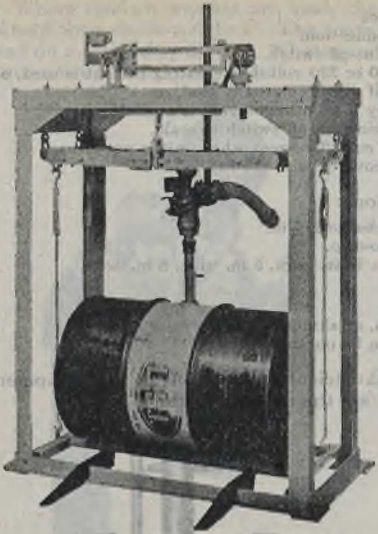


FIG. 111.

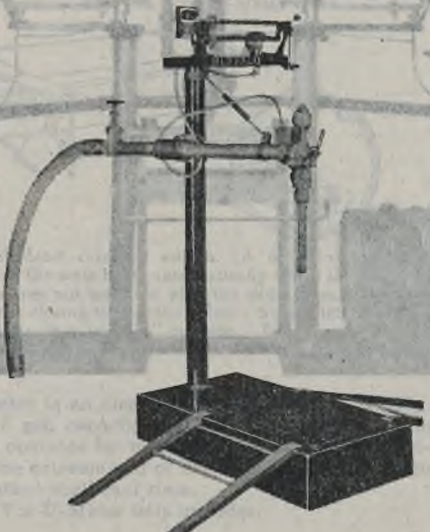


FIG. 112.

SECTION 21

FUELS

BY

Harry A. Curtis, Ph. D., Dean, College of Engineering, University of Missouri; Member, American Chemical Society, American Institute of Chemical Engineers, Society for Promotion of Engineering Education, American Association for Advancement of Science.

H. M. Weir, B. Ch. E., Ph. D., Manager, Chemical Engineering Division, United Engineers & Constructors, Inc.; Member, American Chemical Society, American Institute of Chemical Engineers.

W. A. Myers, B. S. E., M. S. E., Engineering Supervisor, Atlantic Refining Company; Member, American Chemical Society.

Wilbert J. Huff, Ph. D., D. Sc., Chairman, Department of Chemical Engineering, University of Maryland; Consulting Chemist, U. S. Bureau of Mines; Member, American Chemical Society, American Institute of Chemical Engineers, American Gas Association.

CONTENTS

SOLID FUELS

LIQUID FUELS

GASEOUS FUELS

SOLID FUELS

BY HARRY A. CURTIS

REFERENCES: Wheeler, The Chemistry of Coal, *Engineering*, **124**, 344 (1927). Berry, Tar-forming Temperatures of American Coals, *Univ. Wisconsin*, *Bull.* 635, 1914. King and Willgren, The Primary Decomposition of Coal, *British Fuel Research Board*, *Tech. Paper* 16, 1927. Parr, The Classification of Coal, *Ind. Eng. Chem.*, **14**, 919 (1922). Seyler, The Chemical Classification of Coal, *Fuel*, **3**, 15, 41, 79 (1929). Foxwell, The Plastic State of Coal, *Fuel*, **3**, 122 (1924). Layng and Hawthorne, Determination of Temperatures of Plasticity, Maximum Pressures, and Solidification of Coking Coals, *Ind. Eng. Chem.*, **17**, 165 (1925). Ball and Curtis, A Study of Certain American Coals at Temperatures near Their Softening Points, *Ind. Eng. Chem.*, **22**, 137 (1930). Atkinson, Brewer, and Davis, Characteristics of Coal Plasticity, *Ind. Eng. Chem.*, **29**, 840 (1937). Brewer and Atkinson, Plasticity of Coals, *Ind. Eng. Chem.*, *anal. ed.*, **8**, 443 (1936). Auvil, Davis, and McCartney, Expansion of Coal during Coking, *U. S. Bur. Mines*, *Rept. Investigations* 3451, 1939. Richters, Simple Methods for Determining the Relative Caking Power of Coals, *Dinglers polytech. J.*, **195**, 71 (1870). *Brit. Fuel Research Board*, *Tech. Paper* 18, 1927. Marshall and Bird, Test for Measuring the Agglutinating Power of Coal, *Am. Inst. Mining Eng.*, *Tech. Pub.* 216, 1929. *U. S. Bur. Mines*, *Rept. Investigations* 3011, 1930. Parr and Staley, The Reactivity of Coke, *Ind. Eng. Chem.*, **19**, 820 (1927). See also the numerous *Rept. Investigations*, *Tech. Papers*, and journal articles on coal published by the *U. S. Bur. Mines*.

COAL

Of the solid fuels, only coal and the coke derived from it are of prime importance industrially. Within limited areas, a few other solid fuels, such as cordwood, sawmill wastes, bagasse, and bark, are used in considerable quantities.

Trend of Bituminous Coal Consumption in the United States. Table 1, prepared by Otero and Tryon for the National Bituminous Coal Commission, indicates the general trend of bituminous coal consumption in this country since 1918.

Storage of Coal. Underwater storage avoids weathering and danger of spontaneous combustion and is now in common use. For out-of-door storage, the danger of spontaneous combustion and inconvenience therefrom are lessened by observing the following rules:

1. Do not pile in such a way as to permit segregation of lump and fine coal, for this increases the likelihood of fires starting in the segregated fine coal.
2. Do not store coal that has been out of the mine less than a month. When freshly mined, coal goes through an initial weathering period during which it is particularly prone to spontaneous ignition.
3. Do not pile too high, for this tends to prevent escape of heat from regions of local heat development in the pile.
4. Take the temperature of the pile regularly. A temperature above 50°C. at any point is a danger signal.

The storage space necessary for coal varies somewhat with the distribution of sizes. For run-of-mine coal, it is safe to allow 42 cu. ft. per net ton for soft coals and 37 cu. ft. per net ton for anthracite. For prepared sizes, allow about 10 per cent higher space in both cases.

Specific Heats. Coals differ considerably in specific heats, depending on the kind of coal, its ash content, etc. The range is from about 0.25 to 0.38. For metallurgical coke of 5 per cent ash, the specific heats shown in Table 2 have been determined.

Spontaneous Combustion of Coal. The primary cause of spontaneous combustion of coal is the oxidation of the coal substance itself. Moisture and

Table 1. Changes in the United States Consumption of Bituminous Coal by Such Classes of Consumers as Report Currently and by All Other Consumers, 1918–1938

In thousand net tons

Year	Consumed in the United States:								Exported:		Total consumption and exports[g]
	Colliery fuel	Electric power utilities[a]	Bunkers, foreign trade[b]	Locomotive fuel, class I roads[c]	Coke[d]		All other uses[e]	Total consumption[f]	To Canada and Mexico	To other countries (seaborne)	
					Beehive ovens	By-product ovens					
1918	12,521	34,500	6,189	134,214	48,160	36,868	258,141	530,593	18,316	4,034	522,943
1919	11,062	35,100	8,224	119,692	29,730	35,857	241,993	481,658	12,064	8,050	501,772
1920	11,896	37,124	10,486	135,414	31,986	44,205	237,484	508,595	16,458	22,059	547,112
1921	9,123	31,585	8,453	107,910	8,475	28,713	197,590	391,849	13,590	9,541	414,980
1922	7,831	34,179	4,615	113,163	13,286	41,053	212,788	426,915	10,938	1,475	439,328
1923	8,765	38,966	5,093	131,492	30,084	54,276	250,317	518,993	16,960	4,494	540,447
1924	6,618	37,556	4,460	117,247	15,914	49,061	253,148	484,004	12,746	4,354	501,104
1925	5,776	40,222	4,866	117,714	17,423	57,110	256,082	499,193	13,547	3,915	516,655
1926	5,728	41,311	7,736	122,823	19,225	63,647	272,111	532,581	13,762	21,510	567,853
1927	4,930	41,888	4,565	115,883	11,208	63,240	258,087	499,801	14,724	3,288	517,813
1928	4,602	41,350	4,294	112,382	7,018	70,166	259,016	498,828	14,050	2,114	514,992
1929	4,663	44,937	4,287	113,894	10,028	76,759	264,987	519,555	14,727	2,702	536,984
1930	3,993	42,898	3,497	98,400	4,284	65,521	236,397	454,990	13,667	2,210	470,867
1931	3,205	38,735	2,195	81,725	1,767	46,846	197,396	371,869	10,647	1,479	383,995
1932	2,781	30,290	1,350	66,498	1,030	30,887	174,081	306,917	8,429	385	315,731
1933	2,858	30,575	1,316	66,198	1,461	38,681	180,659	321,748	8,600	437	330,785
1934	3,175	33,555	1,321	70,496	1,635	44,343	192,518	347,043	10,213	656	357,912
1935	3,103	34,807	1,576	71,335	1,469	49,046	198,956	360,292	9,044	698	370,034
1936	3,227	42,025	1,622	81,130	2,698	63,244	228,850	422,796	9,912	743	433,451
1937	3,250	44,766	1,832	82,667	4,927	69,575	218,403	425,420	12,052	1,093	438,565
1938[h]	2,518	40,094	1,352	69,675	1,365	44,993	178,736	338,733	9,561	929	349,223

[a] U. S. Geological Survey and Federal Power Commission. Represents all coal consumed by public-utility power plants in power generation, including a small amount of anthracite.

[b] Bureau of Foreign and Domestic Commerce.

[c] Interstate Commerce Commission. Represent bituminous coal consumed as locomotive fuel by class I steam railways, excluding switching and terminal companies. Does not include powerhouse, shop, and station coal.

[d] U. S. Bureau of Mines.

[e] Obtained by subtracting the known items from the calculated total consumption. Includes general manufacturing, domestic, and many miscellaneous uses.

[f] Production plus imports minus exports, plus or minus changes in consumers' stocks.

[g] Note that consumption includes the small amount imported.

[h] Preliminary. Consumption in 1938 was reduced by the business recession and by abnormally mild weather, the year being one of the warmest on record.

pyrites in the coal are contributing factors, as are many other items, such as size of the coal and particularly the segregation of fines in the coal pile, exposure to the sun, steam pipes, or any external source of heat.

Table 2. Specific Heat of Coke

Temperature Range, °C.	Average Specific Heat
20– 260	0.240
20– 538	0.303
20– 815	0.338
20–1093	0.363

The rate of oxidation in most coals increases only very slowly with the temperature up to about 50°C. If conditions for oxidation are particularly favorable and conditions for heat dissipation are poor, the temperature rises above this point, more rapid oxidation occurs with still further increase in temperature until finally the kindling point of the coal is reached. For most bituminous coals, the kindling temperature is above 150°C.

Classification of Coals. No entirely satisfactory scheme of classifying coals has been devised, although many have been proposed using various items and ratios selected from the ultimate and the proximate analyses. The U. S. Geological Survey and the U. S. Bureau of Mines use a classification, shown

Table 3. Classification of Coals by Rank[a]

Class	Group	Limits of fixed carbon or B.t.u., mineral-matter-free basis	Requisite physical properties
I. Anthracite	1. Meta-anthracite	Dry F.C., 98% or more (dry V.M., 2% or less)	Non-agglomerating[b]
	2. Anthracite	Dry F.C., 92% or more and less than 98% (dry V.M. 8% or less and more than 2%)	
	3. Semianthracite	Dry F.C., 86% or more and less than 92% (dry V.M., 14% or less and more than 8%)	
II. Bituminous[c]	1. Low volatile bituminous coal	Dry F.C., 78% or more and less than 86% (dry V.M., 22% or less and more than 14%)	
	2. Medium volatile bituminous coal	Dry F.C., 69% or more and less than 78% (dry V.N., 31% or less and more than 22%)	
	3. High volatile A bituminous coal	Dry F.C., less than 69% (dry V.M., more than 31%); and moist[d] B.t.u. 14,000[e] or more	
	4. High volatile B bituminous coal	Moist[d] B.t.u., 13,000 or more and less than 14,000[e]	
	5. High volatile C bituminous coal	Moist B.t.u., 11,000 or more and less than 13,000[e]	Either agglomerating or non-weathering[f]
III. Subbituminous	1. Subbituminous A coal	Moist B.t.u., 11,000 or more and less than 13,000[e]	Both weathering and non-agglomerating
	2. Subbituminous B coal	Moist B.t.u., 9500 or more and less than 11,000[e]	
	3. Subbituminous C coal	Moist B.t.u., 8300 or more and less than 9500[e]	
IV. Lignitic	1. Lignite	Moist B.t.u., less than 8300	Consolidated
	2. Brown coal	Moist B.t.u., less than 8300	Unconsolidated

F.C. = fixed carbon.
V.M. = volatile matter.

[a] This classification does not include a few coals which have unusual physical and chemical properties and which come within the limits of fixed carbon or B.t.u. of the high-volatile bituminous and subbituminous ranks. All these coals either contain less than 48% dry, mineral-matter-free fixed carbon or have more than 15,500 moist, mineral-matter-free B.t.u.

[b] If agglomerating, classify in low-volatile group of the bituminous class.

[c] It is recognized that there may be non-caking varieties in each group of the bituminous class.

[d] Moist B.t.u. refers to coal containing its natural bed moisture but not including visible water on the surface of the coal.

[e] Coals having 69% or more fixed carbon on the dry, mineral-matter-free basis shall be classified according to fixed carbon, regardless of B.t.u.

[f] There are three varieties of coal in the high-volatile C bituminous coal group, namely, variety 1, agglomerating and non-weathering; variety 2, agglomerating and weathering; variety 3, non-agglomerating and non-weathering.

Table 4. Seyler's Classification of Coal*

Carbon	Anthracite; carbon over 93.3%	Carbonaceous; carbon 93.3–91.2%	Bituminous			Lignitous	
			Meta-, carbon 91.2–89.0%	Ortho-, carbon 89.0–87.0%	Para-, carbon 87.0–84.0%	Meta-, carbon 84–80%	Ortho-, carbon 80–75%
Per-bituminous genus; hydrogen over 5.8%	....	....	Per-bituminous (per-meta-bituminous)	Per-bituminous (per-ortho-bituminous)	Per-bituminous (per-para-bituminous)	Per-lignitous	
Bituminous genus; hydrogen 5.0–5.8%	....	(Pseudo-bituminous species)	Meta-bituminous	Ortho-bituminous	Para-bituminous	Lignitous (meta-ortho-)	
Semibituminous genus; hydrogen 4.5–5.8%	....	Semibituminous species (ortho-semibituminous)	Subbituminous (sub-meta-bituminous)	Subbituminous (sub-ortho-bituminous)	Subbituminous (sub-para-bituminous)	Sublignitous (meta-ortho-)	
Carbonaceous genus; hydrogen 4.0–4.5%	Semianthracitic species	Carbonaceous species (ortho-carbonaceous)	Pseudo-carbonaceous (sub-meta-bituminous)	Pseudo-carbonaceous (sub-ortho-bituminous)	Pseudo-carbonaceous (sub-para-bituminous)		
Anthracite genus; hydrogen under 4%	Ortho-anthracite	Pseudo-anthracite (subcarbonaceous)	Pseudo-anthracite (sub-meta-bituminous)	Pseudo-anthracite (sub-ortho-bituminous)	Pseudo-anthracite (sub-para-bituminous)		

* The various genera are arranged in column 1 vertically, according to the hydrogen. The species in each genus are arranged horizontally according to the carbon.

in Table 3, which has been adopted as standard by the A.S.A. and the A.S.T.M. (D388–38).

The classification of Seyler has been adopted by the publication *Fuels*. In this scheme, the percentages of carbon, hydrogen, and oxygen plus nitrogen as revealed by the ultimate analysis are adjusted so as to add to 100 per cent. The carbon and hydrogen are then plotted on rectangular coordinates, certain areas of the graph being assigned to what Seyler terms the species and genera of coal. Seyler's classification is represented in Table 4.

The Parr scheme of classifying coals is given in the "International Critical Tables."

Specific Gravity. It is necessary to distinguish between the apparent specific gravity of a lump of porous material, such as coke, and the true specific gravity of the substance forming the lump. The A.S.T.M. has adopted standard methods for determining the **true** and the **apparent** specific gravities of coal and coke.

Table 5. Typical Specific Gravities

Fuel	Specific gravity		Pores, per cent
	True	Apparent	
Bituminous coal	1.25–1.45		
By-product coke	1.75–2.00	0.75–1.1	40–60
Low-temperature coke	1.50–1.75	0.5 –1.1*	30–70
Charcoal	1.4–1.7	0.3 –0.6	65–80
Anthracite	1.45–1.7		
Wood	0.5–1.1		

* By unusual procedures in preparing the coal previous to carbonization, and control of carbonization conditions, an apparent specific gravity as high as 1.4 may be obtained.

The Dulong Formula for Calorific Value. For most high-rank coals, the calorific value can be estimated from the ultimate analysis with an accuracy within 2 or 3 per cent by means of the Dulong formula:

$$\text{B.t.u. per lb.} = 14{,}544 \times \text{fraction } C + 62{,}100 \times (\text{fraction } H - \tfrac{1}{8} \text{ fraction } O) + 4050 \times \text{fraction } S$$

Sampling of Coal. It is of utmost importance that coal and coke sampling be carried out rigorously in accordance with a standard method such as that adopted by the A.S.T.M. (D 21-16, D 271-37, D 346-35).

Ultimate Analysis of Coal. The percentages of carbon, hydrogen, nitrogen, and sulfur in the dry coal sample are determined by direct analytical methods. The percentage of ash is determined by the method used in making the proximate analysis. The sum of these components is subtracted from 100 and the difference called the "per cent oxygen" in coal. This procedure does not give quite the true composition of the coal, for the sulfur may be largely in the form of FeS_2 or mineral sulfates which do not appear as such in the ash. Furthermore, the carbonates present in the ash are reduced to oxides and the water of hydration in the minerals driven off. The ash found is, therefore, not the true mineral content of the coal, and the undetermined error in the ash is reflected as an error in the oxygen (see typical analyses, Table 6).

Proximate Analysis of Coal. The proximate analysis is a set of numerical values obtained when a coal is subjected to a series of purely arbitrary procedures which have been standardized by the A.S.T.M. The **moisture** is determined by drying the coal at 105°C. for 1 hr. The **ash** is obtained as a residue on burning the coal. The loss of weight on heating the coal 7 min. at 950°C., minus the moisture, is called the **volatile combustible matter,** and the coke residue, minus the ash, is called the **fixed carbon.** The total sulfur is usually determined in connection with the proximate analysis. The results obtained in each procedure depend to some extent on the details of the methods used, and the arbitrary standards adopted by the A.S.T.M. must, therefore, be followed carefully.

The proximate analysis is the most widely used procedure for evaluating coal, particularly when the general characteristics of other coals from the same district are known. The proximate analysis, however, falls far short of being a complete evaluation of the coal for any one of the uses to which coal is put.

In Table 6 are given the results of proximate and ultimate analyses of typical American coals of various ranks.

Table 6. Proximate and Ultimate Analyses of Typical American Coals*

All data on the ash-free and moisture-free basis

Rank	Volatile combustible matter	Fixed carbon	Sulfur	Hydrogen	Carbon	Nitrogen	Oxygen	B.t.u. per lb.
Montana lignite..................	52.9	47.1	1.5	5.1	70.5	1.5	21.4	12,160
Wyoming subbituminous.........	40.5	59.5	0.5	4.8	76.6	1.1	17.0	13,030
Pennsylvania bituminous.........	30.2	69.8	1.1	5.9	87.1	1.4	4.5	15,560
West Virginia semibituminous ...	19.5	80.5	0.8	4.7	90.7	1.6	2.2	15,670
Pennsylvania semianthracite......	9.9	90.1	0.8	3.7	91.6	1.3	2.6	15,410
Pennsylvania anthracite..........	6.2	93.8	0.6	2.7	94.0	1.0	1.7	14,070

* Analyses selected from data of U. S. Bureau of Mines.

Ash Fusing Point and Clinker Formation. A standard procedure for determining the softening and fusing points of a coal ash has been adopted by the A.S.T.M. Ashes which fuse in the range 1900° to 2200°F. are considered *low fusing;* those in the range 2200° to 2600°F., *medium fusing;* and those from 2600° to 3100°F. or over, are considered as *non-fusing.* In general, low-fusing-point ashes cause trouble through clinkering, but several factors in addition to the ash fusing point affect the clinker formation.

Sufficient data are not yet available for predicting the fusing point of a coal ash from the chemical analysis of the ash except within wide limits.

Briquetting. Roll presses are used almost exclusively in American practice. Coal-tar pitch and petroleum asphalt are the two binders used most extensively. Pitch of about 170°F. melting point is usually specified, or asphalt of 50 to 60 penetration test. It requires 8 to 10 per cent pitch or 6 to 8 per cent asphalt to give a satisfactory briquet. "Sulfite pitch," obtained by evaporating the black liquor from paper mills using the sulfite process, is used to a small extent as a binder, usually in admixture with asphalt or other binders. Starch is used as a binder for charcoal briquets. An emulsion of asphalt in starch paste (Hite binder) has been used successfully in two plants making anthracite culm briquets. The asphalt renders the briquet water resistant, and by replacing a part of the normally required asphalt with starch, a less smoky briquet is obtained.

Coals differ considerably in the ease with which they can be briquetted, and this is particularly true of bituminous coals.

The A.S.T.M. Standards on Coal and Coke. Inasmuch as nearly all the methods of testing coal are empiric, it is necessary to use specifically defined procedures. A compilation of standards relating to coal and coke has been published by the A.S.T.M. covering the following:

D 17-16	Specifications for Foundry Coke.
D 21-16	Method of Sampling Coal.
D 121-30	Definitions of Terms Relating to Coal and Coke.
D 141-23	Method of Shatter Test for Coke.
D 166-24	Specifications for Gas and Coking Coals.
D 167-24	Method of Test for Volume of Cell Space of Lump Coke.
D 197-30	Method of Test for Fineness of Powdered Coal.
D 271-37	Methods of Laboratory Sampling and Analysis of Coal and Coke.
D 291-29	Method of Test for Cubic Foot Weight of Crushed Bituminous Coal.
D 292-29	Method of Test for Cubic Foot Weight of Coke.
D 293-29	Method of Test for Sieve Analysis of Coke.
D 294-29	Method of Tumbler Test for Coke.
D 310-34	Method of Test for Size of Anthracite.
D 311-30	Method of Test for Sieve Analysis of Crushed Bituminous Coal.
D 346-35	Method of Sampling Coke for Analysis.
D 388-38	Specifications for Classification of Coals by Rank.
D 389-37	Specifications for Classification of Coals by Grade.
D 407-35 T	Definition of the Terms Gross Calorific Value and Net Calorific Value of Fuels (Tentative).
D 408-37 T	Method of Test for Grindability of Coal by the Ball-Mill Method (Tentative).
D 409-37 T	Method of Test for Grindability of Coal by the Hardgrove-Machine Method (Tentative).
D 410-38	Method of Test for Screen Analysis of Coal.
D 431-38	Method for Designating the Size of Coal from Its Screen Analysis.
D 440-37 T	Method of Drop Shatter Test for Coal (Tentative).
D 441-37 T	Method of Tumbler Test for Coal (Tentative).
D 492-38 T	Method of Sampling Coals Classed According to Ash Content (Tentative).
D 493-38 T	Definitions for Varieties of Bituminous and Subbituminous Coals (Tentative).
E 11-38 T	Specifications for Sieves for Testing Purposes (Wire-Cloth Sieves, Round-Hole and Square-Hole Screens or Sieves) (Tentative).

Formation of Coal. The initial changes by which vegetable matter is converted into coal are presumably those occurring in every peat bog, the chief of which is the decomposition of the cellulosic and lignitic tissue of plants and their transformation into a class of amorphous substances known as **ulmins.** The ulmins in peat are soluble in caustic alkali, but after the peat has been buried and the subsequent coalification process proceeds, the ulmins become less and less soluble and in high-rank coals are quite insoluble in caustic. When a coal is subjected to mild but prolonged oxidation, the ulmins again become soluble in caustic alkali, the amount of oxidation necessary being greater the higher the rank of the coal.

Those parts of plants which are particularly resistant to decay, such as spore exines, cuticle, and cell walls, are not altered sufficiently to lose their structural identity, either by decay in the peat bog or in the subsequent transformation of peat to coal. Microscopic examination of coal reveals a great variety of such plant structures imbedded in a matrix of amorphous material. Coal also contains resins and other by-products of plant cells which have evidently passed through the coalification process unchanged.

The Macrostructure of Coal. If a small lump of bituminous coal is examined, it will usually be found to consist of laminae. Across the faces of the lump at right angles to the original bedding plane of the coal, the cross sections of these laminae appear as bands. The four ingredients of coal giving rise to these bands have been given distinctive names, although much confusion arose from attempts to correlate the terms used by different investigators, particularly those in England and the United States. The brightest of the ingredients has been called **vitrain** by British investigators, and is to be correlated with the **anthraxylon** discussed by American writers. It is derived from single pieces of woody material. The **clairain** of British writers is also a bright coal, although not so bright as vitrain, and it forms the slightly duller bands noted in American coals. It is an attritus of plant debris, and has been called **transparent attritus** in some recent American writings. The dullest ingredient in British banded coals has been called **durain.** It is also an attritus but is characterized by its opacity. It does not ordinarily occur in typical American banded coals but is the main ingredient of American splint coal. The fourth macroscopic ingredient of coals is mineral charcoal or **fusain.**

Rational Analysis of Coal. The fact that the ulmins in coal of any rank may be rendered soluble in caustic alkali by mild oxidation permits a resolution of the coal into its natural components. Wheeler [*Engineering*, **124**, 344 (1927)] and his coworkers have thus analyzed separately the bright, dull, and charcoal-like parts of a British banded coal with the following results.

Table 7. Rational Analysis of Component Bands of Hamstead Coal

Band	Ulmin compound	Organized plant entities	Hydrocarbons and resins	General character of the plant entities
Vitrain	96	Nil*	4	
Clarain	92	5	3	Cuticle and spore exines
Durain	83	15	2	Cuticle, spore exines, and woody tissues
Fusain	20	80	Nil	Woody tissues

* Vitrain does, however, show plant structure. Curtis.

Carbonization

When the temperature throughout a small mass of crushed bituminous coal is raised slowly, say at 2°C. per min., the following typical events occur. Gases are evolved almost from the beginning, the rate of evolution being very slow at first, with carbon dioxide, carbon monoxide, and water vapor as the principal components. As the temperature rises, the composition of the evolved gases changes continually. Traces of gaseous hydrocarbons, particularly methane, appear early in the process. As the temperature exceeds 100°C., there may be a more or less marked increase in the rate of water-vapor evolution. As the temperature continues to rise, liquefiable hydrocarbons appear in the volatile products, but the rate of evolution of gases and vapors continues low until the softening temperature of the particular coal is reached. A coal does not fuse at a well-defined temperature, but for each coal there is a short temperature range, characteristic of the coal, within which enough liquid products form to cause the whole mass of coal particles to coalesce more or less completely. The degree of fusion varies widely for different coals, from only a slight sintering together of the individual particles to a fusion so complete that the coal liquefies and all trace of individual particles disappears.

The rate of evolution of volatile products formed by decomposition of the coal rapidly increases as the temperature rises above the softening point. If the coal is one which fuses sufficiently to become plastic, the evolved gases form bubbles which work their way out of the plastic mass, causing it to become spongy. As decomposition of the coal substances progresses, with continued evolution of volatile products, the plasticity of the coking mass grows less and less, until finally the bubbles are trapped and the mass takes on a fairly rigid cellular structure. Further evolution of gas can then occur only by diffusion through the cell walls or through cracks in the structure.

The temperature range during which fusing coals are in a plastic condition is relatively small, seldom more than 100°C. and often less than 50°C. Since most fusing coals soften below 410°C., it follows that in most cases the end of the plastic range lies below 500°C. The degree of fluidity, the rate of gas evolution, the rate of temperature increase, and probably several other factors combine to determine the lump or apparent specific gravity of cellular coke, which forms at the end of the plastic range. The true specific gravity of the coke substance remaining at 500°C. is always in the neighborhood of 1.5.

Table 8. Softening Temperatures of Typical Coals*

Trade name	Seam	Locality	Volatile combustible matter, per cent	Fixed carbon, per cent	Ash, per cent	Volatile combustible matter (ash-free basis), per cent	Softening temp. °C.
Island Creek........	Island Creek	Logan Co., W. Va.	39.4	55.3	5.3	41.6	390
Brilliant............	Black Creek	Marion Co., Ala	38.3	59.0	2.7	39.4	†
Black Band.........	Winifred	Kanawha Co., W. Va.	38.1	57.8	4.1	39.8	340
Benham............	Keokee	Harlan Co., Ky.	37.2	60.4	2.4	38.1	390
Pond Creek.........	Pond Creek	Mingo Co., W. Va.	35.9	62.2	1.9	36.6	380
Hazard.............	Hazard No. 6	Perry Co., Ky.	35.8	59.0	5.2	37.8	†
Big Moshannon.....	Moshannon	Clearfield Co., Pa.	23.8	67.7	8.5	26.0	410
Davis..............	Pool 71	Tucker Co., W. Va.	23.1	70.0	6.9	24.8	395
Beckley............	Beckley	Raleigh Co., W. Va.	17.3	81.0	1.8	17.6	430
Stineman...........	Moshannon or Lower Freeport	Cambria Co., Pa.	17.1	76.1	6.7	18.4	425
Manor..............	Kittanning	Garrett Co., Md.	16.3	75.1	8.6	17.9	435
Rockhill............	Fulton	Broadtop field in eastern Pa.	15.9	77.6	6.5	17.0	440
Georges Creek......	Big Vein	Allegany Co., Md	15.0	74.0	11.0	16.8	440

* From Ball and Curtis, *Ind. Eng., Chem.*, **22**, 137 (1930).

† Non-fusing coal.

As the temperature of coke is carried still higher, several processes occur simultaneously. Volatile products continue to be evolved and the coke pieces shrink in volume, these two processes being opposite in their effect so far as the apparent specific gravity is concerned. Depending on the nature of the original coal, either may predominate, *i.e.*, the apparent specific gravity may either increase or decrease. The true specific gravity of the coke substance, however, always increases, going from about 1.5 at 500°C. to 1.9 or higher at 1000°C. The reactivity of the coke decreases steadily during the devolatilization process, and for most cokes the hardness and the strength of the coke likewise increase with the devolatilization.

The Plastic Stage in Coal Carbonization. Most coals below the rank of anthracite and above that of lignite undergo some degree of fusion when heated. Such coals do not have sharply defined fusing points, but for each coal there is a characteristic temperature within a few degrees of which the coal will soften. The method of Foxwell [*Fuel*, **3**, 122 (1924)], as modified by Layng [*Ind. Eng. Chem.*, **17**, 165 (1925)] and later by Ball and Curtis [*Ind. Eng. Chem.*, **22**, 137 (1930)] for determining the plastic range of temperature, is based on the fact that, if a stream of inert gas is passed at a constant rate through a column of crushed coal while the temperature of the coal is slowly raised, a marked increase of the resistance to gas flow through the column may be observed when the coal softens.

Dilatometer methods of studying the plastic condition of coal have been used by Agde (*Brennstoff-Chem.*, **10**, 1929) and by Damm [*Fuel*, **8**, 163 (1928)]. J. D. Davis of the U. S. Bureau of Mines has devised a successful plastometer and Giesler (*Glückauf*, **70**, 178, 1939) has also described a plastometric method. The A.S.T.M. has not as yet adopted a standard method of characterizing the plastic condition of coal.

Products of Coal Carbonization. When coal is carbonized, a complex mixture of compounds passes off in the volatile fraction leaving coke as a residue. In a by-product coke plant, the volatile fraction is subjected to cooling and scrubbing operations which finally yield from a ton of coal:

1. About 12,000 cu. ft. fuel gas running approximately 550 B.t.u. per cu. ft.
2. Two to four gallons of crude light oil removed from the fuel gas by scrubbing with oil.
3. An aqueous liquor containing ammonia, ammonium salts, and various other water-soluble compounds. This liquor is usually heated with a slurry lime to remove all the ammonia and is then discarded.
4. Ammonia, recovered as indicated above and also scrubbed out of the fuel gas, the yield being about 5 lb. In most coke plants all the ammonia is converted to ammonium sulfate and sold as such.
5. Tar, amounting to about 12 gal., is collected from various points in the by-product train of equipment. The tar is composed of a large number of liquids and of crystalline substances, such as naphthalene, anthracene, and others, dissolved in the complex liquid mixture.

In the ordinary coke oven and gas retort, the volatile products liberated from the coal are subjected to subsequent **cracking** by the hot coke and hot walls, and the volatile products listed above are therefore not those initially liberated by the coal. In the various low-temperature carbonization processes, there is much less cracking of the initially liberated products, and the volatile products finally collected are therefore different both in character and in relative amounts. Typical yields for a low-temperature carbonization process operating on a high-rank fusing coal of 35 per cent volatile are:

Semicoke (12% volatile combustible matter)	1500 lb.
Tar	30 gal.
Light oil	2 gal.
Ammonia	2 lb.
Gas, 950 B.t.u. per cu. ft.	3000 cu. ft.

The Agglutinating Power of Coal. Committee D-5 of the A.S.T.M. has proposed a method of determining the agglutinating value of coal. This method involves measurement of the crushing strength of buttons obtained through carbonization of mixtures of coal and silicon carbide under specified conditions (see "A.S.T.M. Standards on Coal and Coke," prepared by Committee D-5, October, 1938).

Tar-forming Temperature of Coal. When the temperature of a mass of coal is slowly increased, say at the rate of 1° or 2°C. per min., water vapor, carbon dioxide, and gaseous hydrocarbons are evolved almost from the beginning of heating, the rate of evolution being very slow at first but gradually increasing. Presently hydrocarbons or related compounds liquefiable at room temperature begin to distill. The temperature at which this "tar" is first observed will obviously depend somewhat on the method used to make the observation. As the coal passes into the plastic condition, the rate of tar evolution increases rapidly and presently passes through a maximum, after which it decreases rather rapidly with rising temperature and presently becomes zero. Berry (*Univ. Wisconsin, Bull.* 35, 1914) studied tar formation from 11 American coals of carbon-hydrogen ratio from 19.1 to 10.6. He records initial tar-forming temperatures ranging from 260° to 375°C., maximum rate of tar formation for all but two of the coals lying somewhere in the temperature range 350° to 500°C., and the rate of tar evolution decreasing to zero before reaching a temperature of 600°C. in all cases except one.

Using a different apparatus, and English coals, King and Willgren (*British Fuel Research Board, Tech. Paper* 16, 1927) observed liquefiable oils liberated from bituminous coals at temperatures in the range 215° to 245°C.

The Reactivity of a Solid Fuel. No generally accepted method of evaluating that property of a solid fuel called "reactivity" exists today. The British Fuel Research Board (*Tech. Paper* 18, 1927) has adopted a method which consists in heating a screened sample (10- to 20-mesh I.M.M.) to 950°C. in a stream of nitrogen under standardized conditions and then measuring, at various intervals of time, the carbon monoxide formed when 100 cc. carbon dioxide are passed over the sample at a rate of 5 cc. per min. Since the reactivity of most solid fuels decreases slowly at 950°C., the British Fuel Research Board defines three activities:

Reactivity I. The sample is heated to 950°C. in a stream of pure nitrogen during the course of an hour, nitrogen is then passed through the sample for another hour, followed by 100 cc. carbon dioxide and the determination of reactivity then made with a further 100 cc. pure carbon monoxide.

Reactivity II. The reactivity is determined in the same manner after nitrogen has been passed over the sample for 5 hr. at 950°C.

Reactivity III. If, after determining **Reactivity I**, the passage of carbon dioxide be continued at 950°C., and new measurements of reactivity be made from time to time, lower and lower values will be obtained until finally the reactivity becomes practically constant at a minimum value. This is called **Reactivity III.**

At 950°C., equilibrium in the system $CO_2:CO:C$ calls for 98.7 per cent CO (the equilibrium value varies somewhat for different kinds of carbon). The number of cubic centimeters of carbon monoxide formed from 100 cc. carbon dioxide, *i.e.*, the reactivity value, is limited to a maximum of 197.4

Typical values for **Reactivity I,** as determined by the British Fuel Research Board's method (*Tech. Paper* 18, 1927) are:

Active carbon	192
Low-temperature briquet	178
By-product coke	100
Beehive coke	43

It must be emphasized that the numerical values obtained depend on the method, and the procedure laid down by the British Fuel Research Board must be followed in detail if results comparable to its values are desired.

A number of methods other than that described above have been proposed for determining the reactivity of a solid fuel, particularly of coke. Most of these depend on the rate of reduction of carbon dioxide to carbon monoxide, but the rate of reaction between the hot coke and steam has also been used. Parr [*Ind. Eng. Chem.*, **19**, 820 (1927)] has taken the ignition temperature in a stream of oxygen as a measure of the reactivity of a coke. Parr found that the ignition temperature of coke from a Franklin County, Ill., coal increased with the temperature at which the coke had been made in the following manner:

Table 9. Effect of Coking Temperature on Reactivity

Coking Temp., °C.	Reactivity Temp., °C.
500	144
600	178
700	218
800	314
900	456

There is a rough correspondence between the ignition temperature of a coke, its reactivity toward carbon dioxide, and its reactivity toward steam.

Burning of Coal on Grates. Most coal used as a fuel industrially is burned either on mechanically operated grates and stokers or in powdered

Table 10. Summary of Information on Stokers

	Chain grates	Overfeed stokers	Underfeed stokers
Description	Fuel from hopper is carried horizontally into furnace on a continuous web. The web is cooled coming out through the ash pit and returns to coal hopper.	Grate extends from hopper into furnace at a rather steep incline, although coal only flows down if a rocking or plunger movement is imparted to grate bars.	Raw coal is pushed up through the fuel bed and cinder falls off onto cinder plate. Gases are distilled in an oxidizing atmosphere. Capable of high overloads.
Fuel used	1. Coke breeze, steam sizes of anthracite, or 2. High-volatile mid-western coals. In general, any non-coking, clinkering coal may be used, but preferably no mixtures.	All coals can be used. Good all-round stoker. Is mainly used on mid-west fuels. Coking coals can be used. Will burn refuse fuels.	High-volatile coals, coking coals, and slack or fines may be burned. Ash must not be easily fusible.
Draft	Natural: 0.25 to 0.60 in. water. Forced: 1- to 2-in. water pressure with coke. 1- to 4-in. water pressure with Illinois and similar coals.	Natural: 0.25 to 0.6 in. water. Forced: 1- to 3-in. water pressure.	All forced draft. Normal: 2- to 4-in. water pressure in wind box. Maximum: 5- to 7-in. water pressure in wind box.
Rate of combustion, lb. per sq. ft. per hr.	Average: 30–35. Maximum: 45–60. 10 lb. per 0.1 in. water, for natural drafts.	Average: 25–35. Maximum: 40–50.	Average: 30–40. Maximum: 60–80. 10 lb. per 1-in. water pressure.
Means of regulation.	1. Height of coal grate. 2. Speed of grate. 3. Amount and distribution of air.	1. Rate of plunger feeder. 2. Rate of ash removal. 3. Amount and distribution of air.	1. Rate of feed. 2. Amount and distribution of blast.
Miscellaneous	Watch for live coals going over end of grate.	Fireman needed for ash removal at times and fire must be cleaned.	Air is admitted over clinker plate to burn out cinder. Fuel bed from 12 to 24 in. deep.

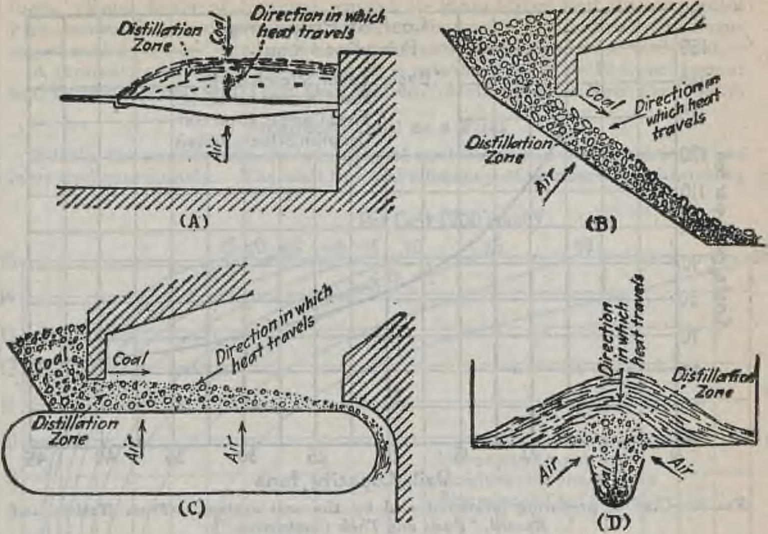


FIG. 1.—Principal methods of feeding coal in industrial furnaces. (*A*) hand-fired; (*B*) over-feed stoker; (*C*) chain-grate stoker; (*D*) under-feed stoker. (*From Haslam and Russell, "Fuels and Their Combustion."*)

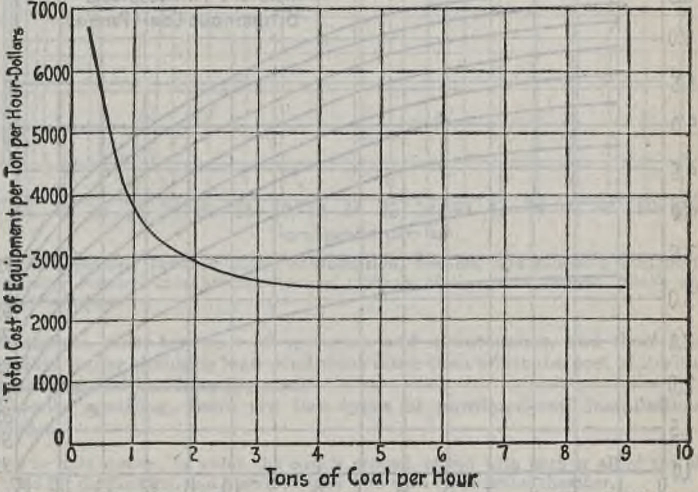


FIG. 2.—Cost of unit pulverizers vs. capacity. (*From Haslam and Russell, "Fuels and Their Combustion."*)

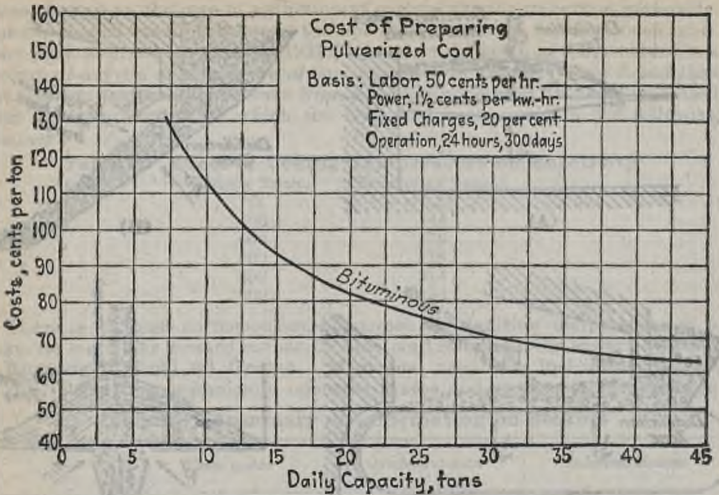


FIG. 3.—Cost of preparing powdered coal by the unit system. (*From Haslam and Russell, "Fuels and Their Combustion."*)

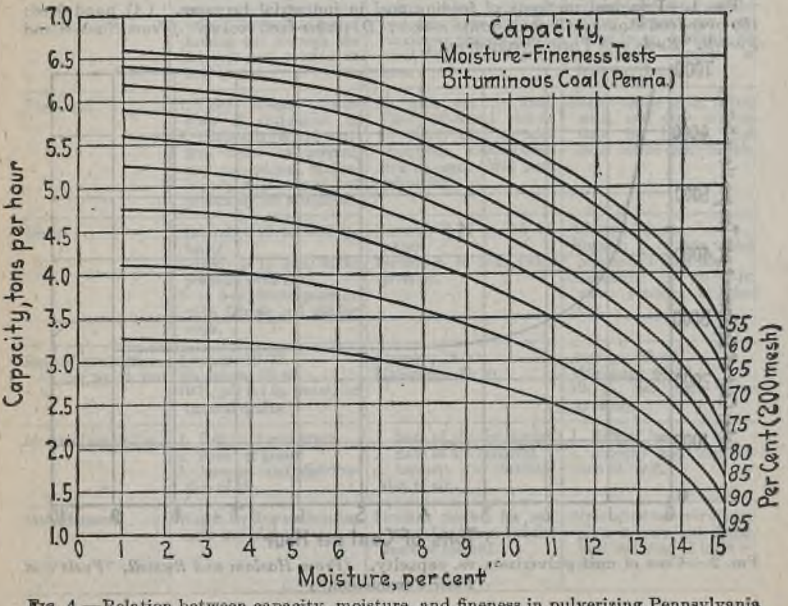


FIG. 4.—Relation between capacity, moisture, and fineness in pulverizing Pennsylvania bituminous coal. (*From Haslam and Russell, "Fuels and Their Combustion."*)

form. Hand firing of furnaces, except in locomotives and in very small installations, is obsolete. In Fig. 1 there is shown diagrammatically the burning of coal on a hand-fired grate and on grates mechanically fed with coal.

A summary of information on stokers is presented in Table 10 from Haslam and Russell, "Fuels and Their Combustion," McGraw-Hill, New York, 1926.

Powdered Coal as a Fuel

Within the past decade the quantity of coal burned in powdered form has increased enormously. The high thermal efficiency of powdered-coal-burning installations, their low cost of operation and maintenance, and their high flexibility under changing load conditions more than offset the cost of drying, pulverizing, and handling the coal.

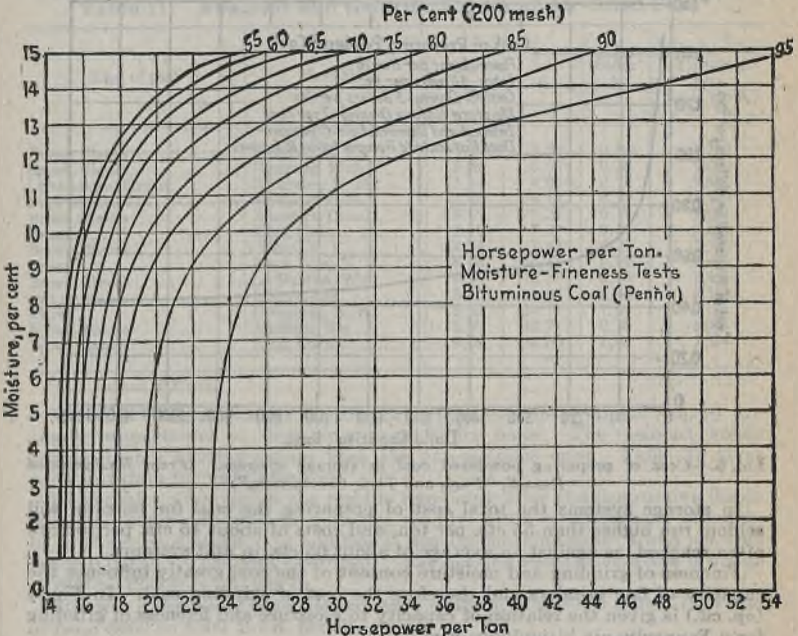


FIG. 5.—Relation between power consumption, fineness, and moisture content in pulverizing Pennsylvania bituminous coal. (*From Haslam and Russell, "Fuels and Their Combustion."*)

Broadly speaking, there are two types of powdered-coal installations, namely:

1. The unit system, in which the coal is ground, mixed with part or all of the air required for combustion, and blown directly into each combustion chamber.
2. The storage system, where the coal is ground and sent to storage, being subsequently drawn from storage, mixed with air, and blown into the combustion chambers.

In general, the unit system has been used for small furnaces burning only a few tons per hour, the storage system being used in large installations. The unit system enjoys the advantages of simplicity, low first cost, and small floor space, as compared with the storage system. The major item of cost in a powdered-coal system is in power consumption for grinding, and here the advantage lies with the large installation.

The cost of unit pulverizers *vs.* capacity is shown in Fig. 2 (from Haslam and Russell, *loc. cit.*) and the total cost of preparing coal in a unit pulverizer is given for various capacities in Fig. 3 (*op. cit.*).

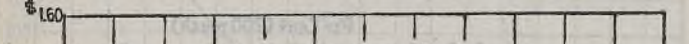


FIG. 6.—Cost of preparing powdered coal in storage systems. (*From Haslam and Russell, "Fuels and Their Combustion."*)

In storage systems the total cost of preparing the coal for burning will seldom run higher than 55 cts. per ton, and costs of about 45 cts. per ton are often reached, as against an average of about 65 cts. in unit systems.

Fineness of grinding and moisture content of the coal greatly influence the capacity of fine grinders and therefore the cost of grinding coal. In Fig. 4 (*op. cit.*) is given the relation of capacity to moisture and fineness of grinding for a Pennsylvania bituminous coal.

In Fig. 5 (*op. cit.*) is shown how power consumption in grinding is affected by the moisture and the fineness of the coal.

An approximation to the total cost of preparing coal in a storage system of pulverizing coal is shown in Fig. 6 (*loc. cit.*).

MISCELLANEOUS SOLID FUELS

Petroleum Coke. In several of the commonly used operations in petroleum refineries, a coke residue is obtained. Typical analyses of petroleum coke are as follows:

	Per Cent
Volatile matter	5–20
Fixed carbon	80–95
Ash	Trace to 1.5
Sulfur	0.5–3

Petroleum coke is used largely in the manufacture of dry cells, carbon electrodes, and electric-furnace resistance elements.

Gas Coke. Gas coke is the residue remaining from the manufacture of coal gas. It is very soft and easily friable and is not suitable for use in metal foundries or furnaces. It is used mainly in the manufacture of water gas.

Peat. Peat is commercially unimportant as a fuel in the United States; no sales of peat for use as fuel were reported to the U. S. Bureau of Mines in 1939.

Typical analyses of peat are shown in Table 11.

Table 11. Analyses and Calorific Values of Air-dried Peat*

Kind of peat	Locality	Water %	Ash %	Sulfur %	Calorific value of air-dried peat, B.t.u. per lb.
Brown, fibrous	Fremont, N. H.	6.34	7.93	0.69	9,290
Brown, fibrous	Hamburg, Mich.	7.50	6.55	0.28	9,090
Light brown, fibrous	Rochester, N. H.	11.64	4.06	0.22	9,083
Dark brown	Westport, Conn.	12.70	4.12	0.24	8,590
Brown, fibrous	Westport, Conn.	19.69	3.23	0.19	7,691
Brown	Kent, Conn.	12.10	7.22	0.83	7,684
Brown, fibrous	Cicero, N. Y.	14.57	7.42	0.25	7,576
Brown	Black Lake, N. Y.	8.68	16.61	0.99	7,522
Brown, fibrous	La Martine, Wis.	9.95	16.77	0.79	7,468
Salt marsh	Kittery, Me.	13.50	12.04	1.94	7,319
Black	Greenland, N. H.	6.62	24.11	1.01	7,186
Brown, fibrous	Madison, Wis.	6.99	18.77	0.38	6,943
Brown, sandy	Kent, Conn.	9.06	36.06	1.46	5,924

* U. S. Bureau of Mines.

Wood. The fuel values of different woods (except resinous woods) are nearly proportional to their weights on a dry basis. The resinous woods possess higher heating values than the non-resinous ones. Table 12 gives pertinent data for various woods.

Charcoal. Charcoal is the residue remaining after the destructive distillation of wood. It absorbs moisture readily, often containing as much as 10 to 15 per cent water. In addition it usually contains about 2 to 3 per cent ash and 0.5 to 1.0 per cent hydrogen. The heating value of charcoal is about 12,000 to 13,000 B.t.u. per lb.

Straw. Depending upon its moisture content, straw has a heating value of from 5000 to 6500 B.t.u. per lb.

Tanbark. Tanbark is the residue remaining after the bark has been used in tanning operations. It usually contains from 60 to 70 per cent water and has a heating value of 2500 to 3000 B.t.u. per lb. In using tanbark as a fuel a very large combustion space is required.

Bagasse. Bagasse is the solid residue remaining after sugar cane has been crushed by pressure rolls. It usually contains from 40 to 50 per cent water and has a heating value of 8000 to 9000 B.t.u. per lb. The fuel value of green mill bagasse is shown in Table 13 (from Marks, "Mechanical Engineers' Handbook," McGraw-Hill, New York, 1941).

Table 12. Approximate Weights and Heating Values per Cord of Fuel Woods*

Variety of wood	Weight per cord, lb.		Available heat units per cord, million B.t.u.		Equivalent in heat value to tons of coal†	
	Green	Air dry	Green	Air dry	Green	Air dry
Ash, white	4300	3800	19.9	20.5	0.77	0.79
Beech	5000	3900	19.7	20.9	0.76	0.80
Birch, yellow	5100	4000	19.4	20.9	0.75	0.80
Chestnut	4900	2700	12.9	15.6	0.50	0.60
Cottonwood	4200	2500	12.7	15.0	0.49	0.58
Elm, white	4400	3100	15.8	17.7	0.61	0.68
Hickory	5700	4600	23.1	24.8	0.89	0.95
Maple sugar	5000	3900	20.4	21.8	0.78	0.84
Maple, red	4700	3200	17.6	19.1	0.68	0.73
Oak, red	5800	3900	19.6	21.7	0.75	0.83
Oak, white	5600	4200	22.4	23.9	0.86	0.92
Pine, yellow	....	....	21.1	22.0	0.81	0.85
Pine, white	....	....	12.9	14.2	0.50	0.55
Walnut, black	....	....	18.6	20.8	0.72	0.80
Willow	4600	2300	10.9	13.5	0.42	0.52

* *U. S. Dept. Agr., Bull.* 753, The Use of Wood for Fuel.

† Short ton (2000 lb.) of coal having a heating value of 13,000 B.t.u.

Table 13. Fuel Value of Green Mill Bagasse

Extraction of weight of cane, %	Moisture in bagasse, %		B.t.u. available for steam generation		Lb. bagasse equivalent to 1 lb. of 14,000 B.t.u. coal	
	(*a*)	(*b*)	(*a*)	(*b*)	(*a*)	(*b*)
75	42.64	51.00	4139	3294	3.38	4.25
77	39.22	48.07	4475	3630	3.13	3.86
79	35.15	44.52	4874	3976	2.87	3.52
81	30.21	40.18	5359	4392	2.61	3.19
83	24.12	35.00	5958	5005	2.35	2.80
85	16.20	28.33	6716	5558	2.08	2.52

(*a*) Based upon tropical cane of 12 per cent fiber and juice containing 18 per cent solid matter. (*b*) Based upon Louisiana cane of 10 per cent fiber and juice containing 15 per cent solid matter.

LIQUID FUELS

BY H. M. WEIR AND W. A. MYERS

Source and Mode of Production of Fuel Oils. Except in unusual and relatively unimportant circumstances, the only commercial liquid fuels sufficiently cheap for power generation are certain fractions of petroleum oil. The term **fuel oil**, as used hereafter, will refer to these materials only. Since most of the fuel oil purchased on the market today has passed through a manufacturing process in a refinery, a general knowledge of the methods used is helpful, if not necessary, to intelligent purchase.

The petroleum refiner is engaged in the production of several major classes of materials from the various types of crude oils which are tapped from nature's underground reservoirs. Products commonly manufactured from so-called "Mid-continent crudes" may be listed roughly in the order of increasing boiling range as: gasoline, kerosene or lamp oil, gas oil, fuel oil, lubricating oil, petroleum wax or paraffin wax, asphalt, and coke. Crude oils obtained in Pennsylvania, West Virginia, etc., yield practically no asphalt and relatively large percentages of paraffin wax. California, Mexico, Texas, or Louisiana Gulf coast crude oils contain negligible amounts of paraffin wax and larger proportions of asphalt than the intermediate type (Mid-continent). The first step in the manufacture of all of these products is a fractional distillation of the crude oil. The conditions maintained in this initial step suffice for separation but do not greatly alter the physical and chemical characteristics of any given fraction.

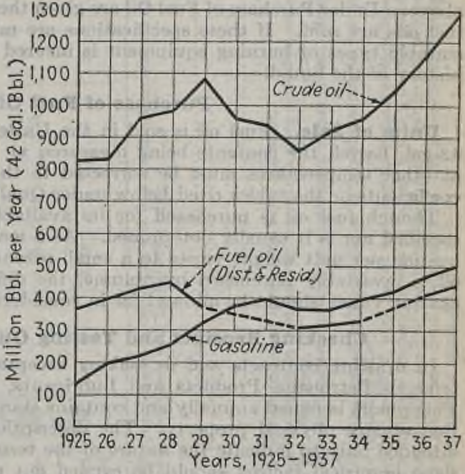


FIG. 7.—Oil and gasoline consumption in the United States. Data from *Mineral Resources* or *Mineral Yearbooks*. Dotted lines indicate data not tabulated.

Figure 7 shows the indicated daily domestic consumption of all types of crude oil, most of which was "consumed" by refineries to make the products listed above. Figure 7 also shows the consumption (by ultimate consumer) of the two major refinery products: gasoline, and distillate and residual fuel oil listed together. It is evident from the graph (Fig. 7) that the demand for gasoline has advanced out of proportion to that for all other petroleum products. On the other hand, the natural composition of crude oils as a class has not changed during the last two decades. The refiner has met this situation by supplementing and, to a significant extent, replacing non-destructive fractional-distillation processes by the installation of "cracking" processes, which convert a portion of the other naturally occurring fractions into the lower boiling gasoline. Liquid residues from these cracking processes are usually sold as fuel oil, because of their quality and because no other product offers the necessary large-volume outlet. As a result of this economic situation, originating in and dominated by the demand for gasoline, very little American crude oil is now burned for fuel *as such*. Fuel oils of today are usually mixtures of residues from cracking processes, together with larger or smaller proportions of corresponding boiling-point fractions, separated in non-destructive fractional distillation of crude oil. The higher boiling and more viscous the oil, the less suitable it is as a source material for refinery cracking units and therefore these "heavier" fuel oils are usually cheaper than the "lighter" (lower boiling) materials.

From the refiner's standpoint, then, a fuel oil is that portion of crude oil and residues which cannot be economically converted to, and marketed as, a higher priced product, either gasoline or one of the other materials mentioned above. Under Purchase of Fuel Oil are given the specifications by which such fuel oils are sold. If these specifications are met, satisfactory operation of suitable types of burning equipment is insured irrespective of the previous history of the liquid.

Purchase of Fuel Oils

Units of Sale. Fuel oil is sold in the United States in multiples of the 42-gal. barrel, the contents being measured at 60°F. Measurements made at other temperatures must be corrected to the standard, using expansion coefficients or the tables cited below under Quality of Material.

Though fuel oil is purchased for its available heat, this factor is never specified nor is it usually determined. As a matter of fact, the heat of combustion per unit weight varies to a small extent only. However, since fuel oil is invariably purchased by volume, the differences in weight, hence in heating value, should be allowed for in considering price.

Checking Receipts and Testing Quality of Fuel Oils

In drawing contracts and in making acceptance tests, it is advisable to refer to Petroleum Products and Lubricants, *A.S.T.M. Rept. Comm. D2.* This report is issued annually and contains standard methods for determination of any physical property. The descriptions of tests given below are intended only to indicate the nature of the tests. Actual determinations of these empirical factors should be carried out precisely as described in the above publication.

Sampling the Oil and Determining Receipts. On receipt in consumers' tanks, the volume of oil should be measured and its average temperature (°F.) obtained immediately. Any recognized method of sampling for laboratory inspection may be used under conditions dictated by the fact that heavy fuel oil, especially, may not be homogeneous but may contain considerable water and salts in suspension or at the bottom of tanks.

Specific Gravity. This determination complements that of temperature of shipments in checking the volume of receipts. Determination can be made by a hydrometer graduated in terms of specific gravity, but it is preferably made with a hydrometer carrying an arbitrary scale termed **Degrees A.P.I.** The latter is defined by

$$\text{Degrees A.P.I.} = \frac{141.5}{\text{specific gravity } 60°/60°\text{F.}} - 131.5$$

Table 14. Detailed Requirements for Fuel Oils[a]

Grade[b]	Flash point, °F.		Pour point, °F.	Water and sediment, %	Carbon residue, %	Ash, %	Distillation temp. °F.				Viscosity, sec.			
							10% point	90% point		End point	Saybolt Universal at 100°F.		Saybolt Furol at 122°F.	
	Min.	Max.	Max.	Max.	Max.	Max.	Max.	Max.	Min.	Max.	Max.	Min.	Max.	Min.
No. 1. fuel oil—a distillate oil for use in burners requiring a volatile fuel	100 or legal	165	15[c]	Trace	0.05 on 10% residuum[d]	......	410	......	......	560[e]				
No. 2. fuel oil—a distillate oil for use in burners requiring a moderately volatile fuel	110 or legal	190	15[c]	0.05	0.25 on 10% residuum[f]	......	440	600						
No. 3. fuel oil—a distillate oil for use in burners requiring a low-viscosity fuel	110 or legal	230	20[c]	0.10	0.15 straight	......	......	675	600[g]	......	55			
No. 5. fuel oil—an oil for use in burners requiring a medium-viscosity fuel	130 or legal	......	......	1.00	......	0.10	......	......	......	......	......	60	40	
No. 6 fuel oil—an oil for use in burners equipped with preheaters permitting a high-viscosity fuel	150	......	......	2.00[h]	......	......	......	......	......	......	......	......	300	45

[a] Recognizing the necessity for low-sulfur fuel oils used in connection with heat treatment, non-ferrous metal, glass and ceramic furnaces, and other special uses, a sulfur requirement may be specified in accordance with the following table:

Grade of Fuel Oil	Sulfur, Max., %
No. 1	0.5
No. 2	0.5
No. 3	0.75
No. 5	No limit
No. 6	No limit

Other sulfur limits may be specified only by mutual agreement between the buyer and seller.

[b] It is the intent of these classifications that failure to meet any requirement of a given grade does not automatically place an oil in the next lower grade unless in fact it meets all requirements of the lower grade.

[c] Lower or higher pour points may be specified whenever required by conditions of storage or use. However, these specifications shall not require a pour point lower than 0°F. under any conditions.

[d] For use in other than sleeve-type, blue-flame burners carbon residue on 10% residuum may be increased to a maximum of 0.12%. This limit may be specified by mutual agreement between the buyer and seller.

[e] The maximum end point may be increased to 590°F. when used in burners other than sleeve-type, blue-flame burners.

[f] To meet certain burner requirements the carbon residue limit may be reduced to 0.15% on 10% residuum.

[g] The minimum distillation temperature of 600°F. for 90% may be waived if A.P.I. gravity is 26 or lower.

[h] Water by distillation, plus sediment by extraction. Sum, maximum 2.0%. The maximum sediment by extraction shall not exceed 0.50%. A deduction in quantity shall be made for all water and sediment in excess of 1.0%.

If the measured temperature of the oil is 60 ± 30°F., the true volume at 60°F. can be determined with sufficient accuracy for most cases from the following coefficients of cubical expansion:

Up to 35°A.P.I. = 0.0004 per degree
35 to 50°A.P.I. = 0.0005 per degree

Abridged volume-correction tables of greater actual and recognized legal accuracy will be found in the publication named above, the same being based upon a complete table in *Nat. Bur. Standards Circ.* 154.

Specifications for Fuel Oils. The specifications described in Table 14 are those promulgated by the Department of Commerce under the title of Commercial Standards CS12-38, issued May 31, 1938.

Water and Sediment. (For Grades 1 to 5, Inclusive.) Standard Method of Test for Water and Sediment in Petroleum Products by Means of Centrifuge, A.S.T.M. Designation: D 96-35.

Water by Distillation. (For Grade 6.) Standard Method of Test for Water in Petroleum Products and Other Bituminous Materials, A.S.T.M. Designation: D 95-30.

Sediment by Extraction. (For Grade 6.) Sediment in Fuel Oil by Extraction, A.S.T.M. Designation: D 473-38T.

Pour Point. Standard Method of Test for Cloud and Pour Points, A.S.T.M. Designation: D 97-34.

Carbon Residue. Standard Method of Test for Carbon Residue of Petroleum Products (Conradson Carbon Residue), A.S.T.M. Designation: D 189-36.

Ash. Procedure for Determination of Ash as Described in the Standard Methods of Analysis of Grease, A.S.T.M. Designation: D 128-37. Sample shall be thoroughly mixed to ensure that portion for ash determination is representative of the sample.

Distillation. Distillation of Grade 1 oil shall be made in accordance with the Standard Method of Test for Distillation of Gasoline, Naphtha, Kerosene, and Similar Petroleum Products, A.S.T.M. Designation: D 86-35; and of Grades 2 and 3 in accordance with the Standard Methods of Testing Gas Oils, A.S.T.M. Designation: D 158-28.

Viscosity. The standard test method designated as D 88-36 calls for the use of a Saybolt Furol or Saybolt Universal viscosimeter. The reading obtained is the time, in seconds, required for 60 cc. of oil at constant temperature to flow through the Furol or Universal orifice under its own (continually decreasing) head. On the other hand, if relative values of viscosity only are required, the expensive Saybolt instrument may be dispensed with and a pipette-type viscosimeter such as described by Ferris [*Ind. Eng. Chem.*, **20**, 974 (1928)] may be used.

The Saybolt Furol orifice is to be used for viscous oils only and at a single standard temperature of 122°F. If the oil shows less than 25 sec. efflux time at this temperature, its viscosity is to be measured at 100°F. with the Saybolt Universal orifice. The application of viscosity values to the design and operation of plant fuel-burning equipment is discussed on p. 2352.

Flash Point. Special equipment is required for determining the flash point of fuel oil, according to the American standard method D 93-36 using the A.S.T.M. Pensky-Martens tester. The flash point has scarcely any connection with the manner in which an oil behaves in an oil burner. However, it does bear some relation to safety in storage even though it be obscure. In general, the lower the flash point, the greater the possibility of fire in storage tanks due to a spark or flame. The small consumer will not find it necessary to check the flash point of his fuel oils, but its value should be obtained from the refiner and provision made to keep storage tanks well below this temperature.

Fuel Oil vs. Coal

Fuel Costs. Figure 8 will be of assistance in making this basic calculation with any given set of data. The example indicates that 1 ton of 9500 B.t.u. per lb. coal burned in a furnace with 80 per cent efficiency is equivalent to 3.5 bbl. of 148,000 B.t.u. per gal. oil burned in a furnace with 70 per cent efficiency. Whereas, the utilization of 80 per cent of the heating value of coal is quite exceptional, this ratio of heat into steam *vs.* heat in the fuel is not uncommonly obtained with oil-burning equipment. The average coal-burning boiler probably operates at something less than 65 per cent efficiency. On the other hand, an over-all boiler-furnace efficiency of 75 per cent should always be obtainable with fuel oil, providing the equipment is in reasonably good operating condition and firemen are alert and passably well informed.

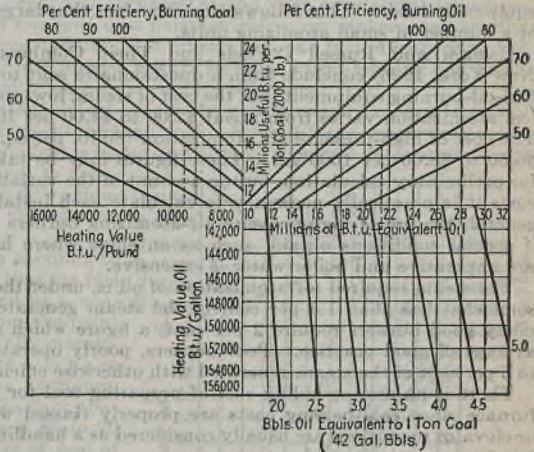


FIG. 8.—Conversion chart. Number of barrels fuel oil equivalent to 1 ton coal.

The higher average efficiency of oil-burning equipment reflects the relatively lower percentages of excess air necessary for combustion and the absence of ashpit losses. Nevertheless, there is an unavoidable loss in fuel-oil firing which may be considered to correspond to ashpit loss in coal firing.

Complete combustion of oil involves the formation of approximately an equal weight of water—since oil is about one-eighth hydrogen. The flue gases thus carry out the stack not only the sensible heat in the other gases but also all the latent-heat equivalent of the water (steam) formed. Depending on the oil used and the stack temperature, this loss may vary from 7 to 9 per cent, whereas the corresponding "hydrogen-steam" loss from anthracite coal is of the order of 2.5 per cent and from bituminous coal up to 4.5 per cent.

Labor. The oil-fired plant shows a great saving in labor cost over the hand-fired coal-burning plant, chiefly because of the elimination of coal passing and ash handling. Even the most modern stoker and ash-handling equipment, or powdered-coal-burning equipment, as installed in larger plants, requires more labor than oil firing. A few firemen can control a large number of oil burners efficiently and the labor connected with cleaning fires is obviated.

If the oil burners are properly controlled, it is not necessary to clean flues so often as with equally well-operated coal-burning equipment. Good-class firing labor can be retained for oil-fired installations due to the pleasant working conditions which obtain. As a matter of fact, only conscientious and intelligent firemen should be employed, for, though fuel-oil firing is capable of exact control, poorly informed or careless operators can easily waste several times their daily wage in the price of oil inexpertly burned.

Operating Cost. The cost of atomizing fuel oil depends more on the primary cost of power, steam, or air than it does on the size of the oil-burning equipment itself. This follows from the fact that large installations consist of a number of small atomizing units.

Haslam and Russell ("Fuels and Their Combustion," McGraw-Hill, New York, 1926) conclude from a questionnaire sent to 24 manufacturers of fuel-oil-burning equipment that the cost of steam, low-pressure air, or mechanical atomization varies from about $2.00 to $4.00 per 1000 gal. oil atomized. The cost of high-pressure air atomization (80 lb. per sq. in.) is given as from $6.50 to $8.00 per 1000 gal. These figures may be taken as good averages for preliminary calculation, but on account of the variation in primary power costs it is practically necessary to calculate each installation for itself. In general, the use of high-pressure air-atomizing burners can only be justified if special conditions obtain, such as on ships where high-capacity burners are imperative and boiler water is expensive.

The steam required for atomization of oil is, under the best circumstances, somewhat less than 1.5 per cent of the steam generated by the boiler, but many good burners require 2 per cent, a figure which may be taken as the average of good practice. Poor burners, poorly operated, may consume up to 5 per cent of the steam generated with otherwise efficient boiler equipment.

There is no corresponding cost of preparing coal for hand- or stoker-fired furnace since coal-passing costs are properly classed with labor, and crane or elevator operations are usually considered as a handling cost to be added to freight.

The cost of preparing powdered coal varies between $0.25 to $0.75 per ton, exclusive of fixed charges. The power used is 1.5 to 2 per cent of the output of the boiler equipment, hence is comparable to that required for oil burners using steam atomization.

Repairs, Maintenance, and Depreciation. The repair and maintenance cost of the oil-burning equipment itself is invariably small since the only moving parts in many installations are the pumps. Five to seven per cent of the total investment is usually ample allowance for repairs and fixed-rate depreciation of 10 per cent per annum is considered good practice.

Oil firing is synonymous with high-temperature combustion zones, hence high-grade refractories must be used if the repair cost of furnace linings is to be kept within reason. However, the same applies to powdered-coal- or stoker-fired-coal-burning furnaces. The expansion and contraction of furnace linings, due to opening firing doors in hand firing of coal, are obviated by any of the above alternatives. Proportional decreases in maintenance cost partly, if not wholly, offset the effect of inherently high-temperature operations.

Investment Costs. The investment required for oil-burning equipment is influenced largely by items of tankage and piping. Since the kind of tankage and its location are frequently prescribed by local ordinance, these factors are not entirely under the control of the engineer. The oil-storage volume advisable per unit of oil-burning equipment is also contingent on regularity of delivery, which in turn usually reflects the distance from source of oil supply.

As a rough general figure, the investment may be assumed to vary from about $3000 per 1000 gal. of oil burned per day for small installations (150 to 200 boiler h.p.) to about $1200 per 1000 gal. of oil per day for 5000 boiler-h.p. plants. These costs include tankage, piping, heaters, pumps, meters, etc., set up ready to operate.

The comparative installation costs of coal-fired equipment scarcely admit of any generalization. These costs may vary from almost negligible figures in the case of hand-fired coal furnaces to the very substantial investment entailed by overhead bunkers, coal hoists or elevators, stokers, ash-handling devices, etc. However, two items of investment, the fuel inventory and the ground area required, are almost always in favor of oil-burning equipment, even in comparison with hand-fired coal installations.

The wide distribution of fuel-oil supply points usually makes it feasible to carry a smaller oil inventory than coal. Even if large volume reserves of oil must be carried, the fact is important that oil does not deteriorate in storage and is not subject to spontaneous combustion or slow oxidation which reduces the heating value of stored coal.

Another advantage accruing to the use of oil fuels is that the fuel storage may be placed at a considerable distance from the point of use without materially increasing handling costs. Where ground space is at a premium, this, together with the fact that oil requires only about 60 to 70 per cent of the volume of coal of equal heating value may present real advantages.

Properties of Fuel Oils

Under Purchase of Fuel Oils (p. 2344), those properties which form the subject of buyers' and sellers' contracts are discussed. In this section, properties of fuel oils of interest to the engineers designing equipment or to the users of fuel oil are presented. These include heat content of oils, air required for combustion, flue-gas analysis as conditioned by excess air, and oil viscosity as a function of the temperature.

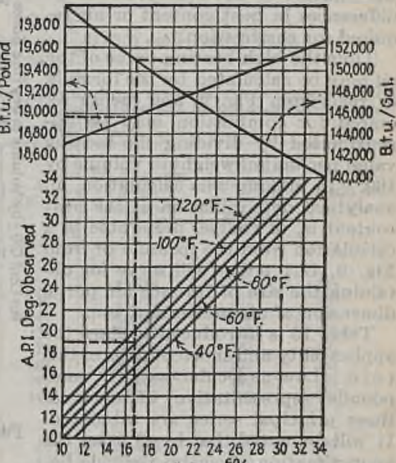


Fig. 9.—Heating value of fuel oil.

The high heating value of an oil (hydrogen burned to liquid water) in terms of both B.t.u. per gallon and B.t.u. per pound is given as a function of the A.P.I. gravity of the oil in Fig. 9. The example, illustrated by dotted lines on Fig. 9, shows that an oil which indicates 19° A.P.I. gravity at 100°F. has a true A.P.I. gravity (measured at 60°F.) of 16.8. Furthermore, its high heating value is 151,000 B.t.u. per gal. or 18,900 B.t.u. per lb.

The air required for the complete combustion of an oil may be accurately calculated from the data of an ultimate organic analysis, using the formula:

$$\text{Pounds of air per pound of oil} = 0.1152\ \text{C} + 0.3456\left(\text{H} - \frac{\text{O}}{8}\right) + 0.04328$$

A fairly good approximation of the air required for combustion can, however, be made without recourse to an elementary analysis. This circumstance is due to the fact that fuel oils do not differ greatly in their elementary composition. As a matter of fact, such oils range from 83 to 87 per cent carbon content, together with 11 to 13 per cent by weight of hydrogen. The remainder consists, for the most part, of sulfur and water dissolved or mechanically mixed with the oil. Despite the similarity in the ultimate composition of such oils, their chemical constitution is very complex and includes rather large molecular weight hydrocarbons of the saturated, unsaturated, and cyclic types. Aside from these hydrocarbons, a few per cent of other organic compounds containing sulfur, together with occasional small amounts of oxygen or nitrogen, are usually present. The varied arrangement of these few elements in the molecules of the mixture is not reflected in large differences in heat content or air required for combustion.

Thus the high heating value of the oil may be calculated by the formula or read from Fig. 9, and the air required for combustion may be approximated by dividing the heating value per unit of weight or volume by 100. In making this calculation, an analytical determination of the heat content is, of course, preferable to a calculation from the formula or from Fig. 9, but either will serve for obtaining the size of furnace air ports, dimension of air preheaters, etc.

Table 15 is introduced to show the applicability and shortcomings of this rule. Low-molecular-weight compounds representative of each of three principal series are tabulated. It will be noted that in all cases the approximation given by the rule becomes better with higher molecular weight. Unfortunately, sufficient data on pure high-molecular-weight compounds do not exist to prove the point with fuel oils, but experience has shown that the rule works quite well for these mixtures.

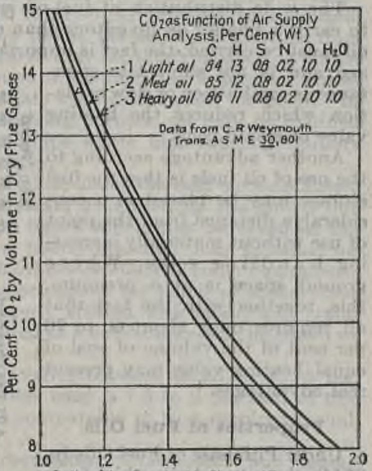


FIG. 10.—Relation of carbon dioxide in flue gas to excess air used.

Figure 10 relates the percentage of carbon dioxide in dry flue gases to the ratio of air theoretically required and to the air actually used. The comparatively small difference between the combustion characteristics of light and heavy oil is illustrated by the chart, which is useful in orienting an observer on the meaning of CO_2 determinations in flue gases from oil-fired equipment. (Note that the usual Orsat gas analysis yields per cent of constituents approximately on the basis of dry gas irrespective of the water content of sample.)

The viscosity of fuel oil at the point of injection to the burner is of considerable importance from the standpoint of efficient operation. This statement is equivalent to stressing the importance of the temperature of the oil since the desired viscosity is obtained with power-plant fuels by raising the temperature. Different oils of the same A.P.I. gravity, or specific gravity, do not necessarily have the same viscosity at any given temperature. This, together with the fact that routine laboratory determinations of viscosity are never carried out at the temperature at which the oil is used, forms a

Table 15. Heats of Combustion and Air Required to Burn Hydrocarbons

1 lb.-mol oxygen is contained in 1820 cu. ft. air (60°F., 29.92 in. Hg).

1 lb. paraffins (C_nH_{2n+2}) require $\dfrac{3+\frac{1}{n}}{28+\frac{4}{n}}(1820)$ cu. ft. air to burn.

1 lb. olefins and cycloparaffins (C_nH_{2n}) require $\frac{3}{28}$ (1820) cu. ft. air to burn.

1 lb. acetylenes and diolefins (C_nH_{2n-2}) require $\dfrac{3-\frac{1}{n}}{28-\frac{4}{n}}(1820)$ cu. ft. air to burn.

1 lb. olefin acetylenes (C_nH_{2n-4}) require $\dfrac{3-\frac{2}{n}}{28-\frac{8}{n}}(1820)$ cu. ft. air to burn.

Hence, as the molecular weight of these hydrocarbons grows larger (n increases), the air required per lb. for complete combustion approaches the constant-quantity characteristic of the olefines, namely, $\frac{3}{28}(1820) = 195$ cu. ft.

Hydrocarbon	Formula	A, Mol. wt.	B*, B.t.u. per lb. on combustion	C, Cu. ft. air to burn 1 lb. completely. Calculated	Ratio $B{:}C$
Paraffins:					
Methane	CH_4 (gas)	16	23,550	228	103
Ethane	C_2H_6 (gas)	30	21,950	212	103
Propane	C_3H_8 (gas)	44	21,390	207	103
Butane (i.)	C_4H_{10} (gas)	58	21,000	203	102+
Pentane (n.)	C_5H_{12} (liquid)	72	20,700	202	102
Hexane (n.)	C_6H_{14} (liquid)	86	20,580	201	102
Octane (n.)	C_8H_{18} (liquid)	114	20,400	200	102
Decane	$C_{10}H_{22}$ (liquid)	142	20,200	199	101+
Olefines:					
Ethylene	C_2H_4 (gas)	28	21,200	195	109−
Propylene	C_3H_6 (gas)	42	20,800	195	107−
Butylene	C_4H_8 (gas)	56	20,600	195	105+
Amylene	C_5H_{10} (gas)	70	20,500	195	105
Hexylene	C_6H_{12} (liquid)	84	20,250	195	104
Diamylene	$C_{10}H_{20}$ (liquid)	140	20,200	195	103.5
Acetylenes:					
Acetylene	C_2H_2 (gas)	26	21,400	151	142
Trimethylene	C_3H_4 (gas)	40	20,400–21,200	176	116–120+
Diallyl	C_6H_{10} (liquid)	82	20,300	182	111
Heptine	C_7H_{12} (liquid)	96	20,300	190−	107

* Data from "International Critical Tables," vol. 5, pp. 163–164, McGraw-Hill, New York, 1929.

problem conveniently solved by reference to the recently adopted A.S.T.M. temperature-viscosity chart for petroleum oils. A straight line drawn on this chart between points representing two viscosity determinations at different temperatures enables the observer to read the viscosity of the particular oil at any other temperature. (The chart is equally applicable to the determination of the viscosity-temperature relationship of lubricating oils.)

If only one viscosity determination is available a rough approximation to the change in viscosity with temperature can be determined by drawing a line through the point parallel to a line drawn on the chart to represent an oil of similar type, *i.e.*, depending on whether the oil is made from a paraffinic, mixed, or naphthenic, base crude. As a general rule the type of the fuel oil is not well known by the user, hence the method is of limited application. In any case, if other than the roughest approximation is desired the actual **determination** of viscosity at two temperatures is necessary.

The minimum temperature to which an oil must be raised for successful burning is conditioned not only by the oil chosen but also by the burner used. To get best results with mechanical burners, the viscosity should not exceed 150 sec. Saybolt Universal. A higher viscosity oil (lower temperature) can usually be handled efficiently by steam atomizing burners. Since the oil must be brought to steam-atomizing burners at pressures from 10 to 70 lb. per sq. in. and to mechanical burners at from 100 up to perhaps 250 lb. per sq. in., the effect of viscosity on pumping costs requires consideration. While it is advantageous to locate the oil-feed pumps near the burners, this is not always possible. In such cases the choice of oil, the viscosity to which it must be reduced, and the kind of burners present a problem of economics involving the laws of fluid flow. The proper-size pumps can be shown by an analysis of the situation in the light of the discussion pp. 2244–2265. No oil burner operates with the regularity inherently possible if it is necessary to heat the oil much above the point where it begins to vaporize. Hence the feed pumps, piping, etc., should be chosen of such dimensions that the required volume and pressure of the oil supply do not demand viscosity reductions corresponding to excessive temperatures.

Details of Fuel-oil Burning Equipment

General. The National Board of Fire Underwriters publishes a pamphlet "Regulations . . . for the Installation of Oil-burning Equipments." In order to make certain that the lowest insurance rates will apply, equipment should be designed to conform to these regulations, which are those recommended by the National Fire Protection Association. While these regulations ensure physically "safe" equipment, it sometimes happens that municipalities or other local districts have certain additional rules governing such installations. It is the part of wisdom, therefore, to obtain the approval of the local fire authorities before actually installing any proposed equipment. The details which are particularly subject to regulation are: location of tanks (with respect to property lines), strength of materials, arrangement of drains or gravity cross connections from one storage tank to another, etc.

The principal units necessary for oil-burning equipment are indicated diagrammatically in Fig. 11. These are the storage tanks, pumps, oil heaters, and burners. Probably the most generally satisfactory procedure to adopt in providing equipment for fuel-oil burners is to purchase a complete "fuel-oil set." A number of manufacturers market these sets consisting of strainers, pump, steam heater, and pressure regulators for oil passing to the burners.

All the equipment is frequently mounted on one base plate. Nevertheless it sometimes happens that a system can best be served by purchase or manufacture of the equipment in parts rather than as a whole; hence, a few notes may be of value.

Storage Tanks.* Division of storage tankage into at least two parts facilitates cleaning and simplifies gaging of receipts. If the latter advantage is to be fully realized, each tank should hold at least one unit of delivery. A 13,000-gal. tank will hold the contents of any tank car; tank trucks ordinarily deliver less than 3000 gal. The maximum size of tanks is a matter of judgment predicated on the regularity of supply and influenced by local regulation. It is usually not absolutely necessary to provide storage for more than 10 days' supply of oil, but three weeks' or more capacity should be provided if at all feasible. Storage tanks for heavy oils should be provided with steam coils to keep the oil fluid for the pump suction. A bottom draw-off or, in the case of underground tanks, a sump with separate suction line should be installed for drawing off water and sediment. If possible, tanks should be arranged for filling by gravity from tank car or truck. A steam hose connection near the inlet to the system will be advantageous, especially for cold-weather deliveries.

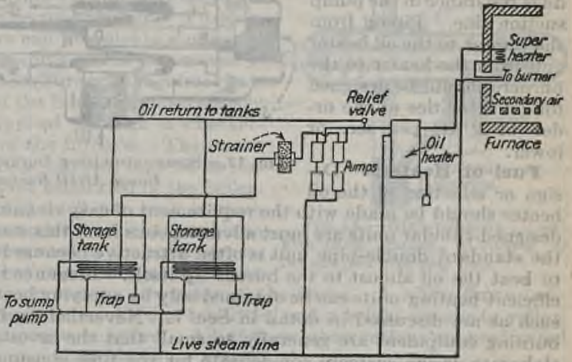


FIG. 11.—Fuel-oil firing system.

A simple gage pole generally meets all measuring requirements for fuel-oil storage. Gage glasses are troublesome with heavy-oil installations and are not to be generally recommended. If a continuous indicator is desired, one of several float devices on the market may be purchased. However, it is doubtful if any of these will serve the purpose better for outside tanks above ground than the following home-made device. Pass a $\frac{1}{10}$-in. steel wire through a gland on the top of the tank and over an old bicycle rim and wheel arranged as a pulley turning on its original ball-bearing axle. Connect one end of the wire to a 2-ft. square wooden float in the tank and the other to a weighted indicator guided by, and registering over, a scale on the outside of the tank. If the gland is lightly packed with lead wool, there will be no danger of accidentally ignited gases striking back through the opening and all insurance regulations in this regard will be met. The device will operate over a period of years without attention.

Pumps—Piping. Almost any type of pump of the proper size and capable of producing the requisite pressure may be used. A horizontal duplex steam pump provided with an air chamber to reduce the pulsation on the outlet oil is the most generally satisfactory, though motor-driven rotary pumps are gaining in favor for this service. Strainers should be inserted before the

---

* See Sec. 20.

pump and a two-compartment-type strainer is advisable, since cleaning can be done without affecting operation. Pumps should be installed in duplicate, particularly where fire protection equipment is dependent on the continuity of fuel-oil (steam) supply. The pumps should be placed as close to the storage tanks and at as low a level as possible. Advantage should be taken of every possible opportunity to reduce resistance in the pump suction line. Piping from the pumps to the oil heater and from the heater to the burners should be designed for oil velocities of the order of 2 ft. per sec. or lower.

FIG. 12.—Steam-atomizing burner. (*Courtesy of National Aeroil Burner Co.*)

Fuel-oil Heaters. Design or selection of the oil heater should be made with the requirement of easy cleaning in mind. Well-designed tubular units are most advantageous from this standpoint alone, but the standard double-pipe unit is often attractive because it makes it possible to heat the oil almost to the burner tip itself. Elegance in design and most efficient heating units can be obtained only by applying heat-transfer formulas such as are discussed in detail in Sec. 7. Nevertheless, heaters for fuel-oil-burning equipment are generally so small that the investment savings from elaborate design scarcely compensate for the time consumed. Such heaters may be designed on the assumption that the heat-transfer coefficient from steam to oil is 80 B.t.u./(sq. ft.) (°F. per hr.), with an oil velocity of 3 ft. per sec.; or 40 B.t.u./(sq. ft.) (hr. per °F.), with an oil velocity of 1 ft. per sec. The specific heat of the oil may be taken as 0.5. Enough surface should be provided to reduce the viscosity of the oil to the proper point for the particular burner when the surfaces are fouled sufficiently to reduce heat-transfer rates to about one-third the above values.

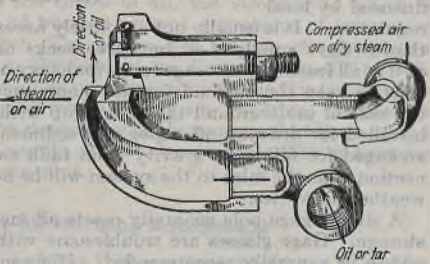


FIG. 13.—Steam-atomizing burner. (*Courtesy of W. N. Best Corp.*)

Burners. There are five types of fuel-oil burners in use at the present time, namely: (1) Steam atomizing; (2) high-pressure air atomizing; (3) low-pressure air atomizing; (4) spray nozzle (also called "mechanical atomizing"); (5) rotary mechanical atomizing.

There are at least 75 companies in the United States manufacturing fuel-oil burners. Furthermore, experience indicates that it is almost impossible for a mechanically minded individual to observe the operation of fuel-oil burners for any length of time without feeling the urge to design his own improvement thereof. As a result of this situation, there are a large number of satisfactory designs in each of the above types of burners as well as a host of very mediocre appliances. The figures which follow illustrate a few of the good burners which are on the market, although the choice of a burner for illustration does not necessarily mean it is considered as the most efficient in its class.

Figures 12 and 13 show two examples of the first class of burner, namely, burners using high-pressure steam for atomizing. Figure 12 shows the inside-mixing type where steam and oil form a mixture inside the burner. Figure 13 shows the outside-mixing type. Both of the burners illustrated present advantages over certain other examples of this type in that there are no small orifices to become clogged. Furthermore both burners can be quickly and easily disassembled for cleaning whenever necessary.

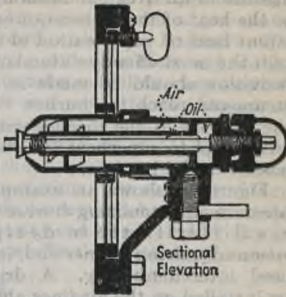


Fig. 14.—High-pressure air-atomizing burner. (*Courtesy of Schutte and Koerting Co.*)

With either of these burners, the primary air is drawn in around the burner, and secondary air is usually supplied through a checkerwork in the base of the furnace. The steam consumption of these burners may be as low as 1 per cent of that generated in the boiler. However, about 1½ to 2 per cent is more usual, and with poor operation it is possible to use as much as 5 per cent of the total steam.

Proper adjustment of the steam quantities can be made by observing the character of the flame. A white flame indicates too much steam, a smoky one too little, and a bright orange to yellow flame indicates the proper mixture. All of these observations are predicated on sufficient air for combustion being present.

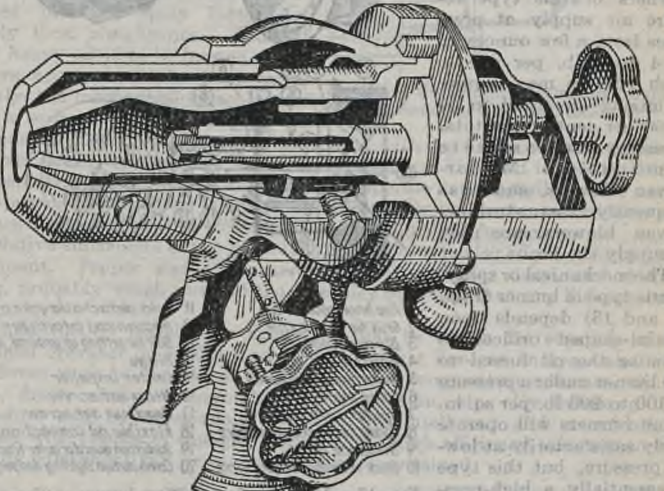


Fig. 15.—Low-pressure air-atomizing burner. (*Courtesy of Hauck Manufacturing Co.*)

It is hardly necessary to state that the only function of the steam is to break up the oil into fine droplets so that it can quickly and thoroughly contact with air for combustion. Having effected this atomization, dry steam has practically no further effect on the combustion. However, if the steam is wet, combustion is slowed down, burners operate in an irregular fashion, and a portion of the heat of the oil is required to supply the latent heat of vaporization of water. To obtain the most effective atomization of the oil, provision should be made so that only dry steam can reach the burner. A small coil in the fire box of the furnace can be used with advantage to superheat the steam before it is passed to the burners.

Figure 14 shows an example of the high-pressure air-atomizing burner. These burners differ little in design from the steam-atomizing burner and, in fact, are often used interchangeably. A drip tank should be installed on the air lines ahead of compressed-air-atomizing burners, since occasional slugs of water may extinguish the flame, particularly before the combustion chamber has attained full operating temperature.

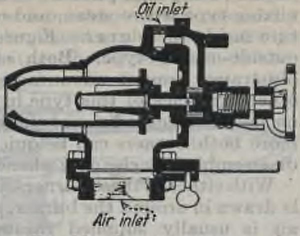
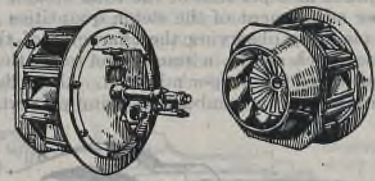
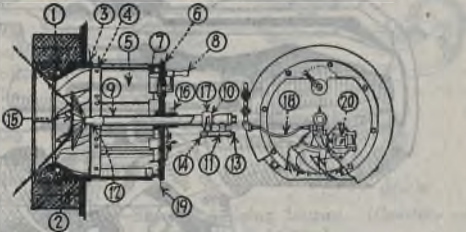


FIG. 16.—Low-pressure air-atomizing burner. (*Courtesy of Schutte and Koerting Co.*)

Figures 15 and 16 show two examples of low-pressure air-atomizing burners. Burners of this type require air supply at pressures from a few ounces up to 4 to 5 lb. per sq. in. Such burners may require atomizing air equal to 50 or 60 per cent of the total necessary for complete combustion. Motor-driven blowers, and less frequently steam-turbine-driven blowers, are used to supply the air.





1 *Fire brick moulded tile*
2 *Grid for holding tile*
3 *Bladed cone*
4 *Register*
5 *Automatic air doors*
6 *Cover plate*
7 *Spider with cams*
8 *Handle for operating air doors*
9 *Distance piece*
10 *Quick detachable coupling*
11 *Quick detachable yoke*
12 *Mechanical atomizer*
13 *Bolt for setting up ground joint*
14 *Hinge*
15 *Center impeller*
16 *Wing set screw*
17 *Headless set screw*
18 *Flexible oil connection*
19 *Radiation guard or outer front plate*
20 *Combination lighting and peep door*

FIG. 17.—Mechanical atomizing burner. (*Courtesy of Babcock and Wilcox.*)

The mechanical or spray-nozzle type of burner (Figs. 17 and 18) depends upon special-shaped orifices to atomize the oil forced to the burner under a pressure of 100 to 200 lb. per sq. in. Some burners will operate fairly satisfactorily at lower pressure, but this type is essentially a high-pressure, low-viscosity oil burner. Heavy fuel oil can of course be burned with these appliances, but the temperature must be raised so that the viscosity of the oil ranges from 100 to 125 sec. Saybolt Universal for best results.

When mechanical spray burners are used, it is practically necessary to arrange to supply all air for combustion through openings around the atomizer. The best results can be secured only if the air is guided in proper channels with respect to the pattern of the oil spray. Air registers such as are shown in the illustrations are, therefore, always supplied with these burners. The size of air ports in these registers should be easily and accurately adjustable.

Mechanical spray burners are very popular for marine boilers when elimination of steam consumption and saving of space and power for air blowers or compressors are of importance. The two principal disadvantages of this type of burner are: (1) the necessity of removing practically all suspended matter from the oil, else the small burner orifices will clog up; (2) interdependence of quantity of oil burned and character of the atomization. As a matter of fact, variation of capacity over wide ranges can only be secured by changing the nozzles in these burners. Small changes in capacity can be effected by changing the oil pressure, though the procedure is less effective with small nozzles than with larger sizes.

The last of the five classes of burners, the rotary mechanical type, is finding extended application for household heating boilers. The invasion of the power-plant field by this type of burner is improbable due to the difficulty of handling heavy fuel oil.

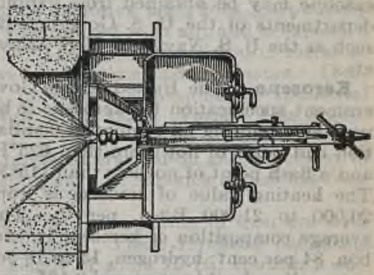


FIG. 18.—Mechanical atomizing burner. (*Courtesy of Bethlehem Steel Co.*)

The selection of the proper type of burner involves many factors of purely local significance. Thus, if very heavy oil is to be burned under boilers upon which the load varies greatly, the mechanical spray burner is at a serious disadvantage and the tendency of steam jet burners to "blow out" with low fires must be considered. If medium- or light-gravity oil is to be used and the load is steady, either mechanical burners or steam burners will operate satisfactorily. The use of high-pressure air burners is becoming obsolete for boiler plants, but they are still used to advantage in some types of metallurgical furnaces.

Relative efficiencies of the various types of burners also admit of no general statement. Proper size of burners for the particular duty, and skill in firing, probably weigh more in the efficiency obtained than any differences of design.

A new fuel-oil installation should at first be under the scrutiny of an informed operator or engineer who can interpret flue-gas analyses and who can direct work to determine the optimum oil temperatures, burner adjustment, draft conditions, secondary air regulation, etc. By following this procedure and by occasionally checking up intelligent and conscientious firemen with results of flue-gas analyses, the high efficiency inherent to oil-burning equipment can be established and maintained.

Miscellaneous Liquid Fuels

Coal Tar. Coal tar is a by-product of the manufacture of coke and of coal gas. This tar is a viscous mixture consisting, for the most part, of aromatic compounds. Its heating value varies from 15,000 to 16,500 B.t.u. per lb. In order to burn coal tar in regular fuel-oil burners, the tar must be filtered and preheated to such a temperature that its viscosity is reduced to that of the oils for which the particular burner is designed.

Tar Oil. Tar oil is obtained by the distillation of coal tar and consists of so-called "creosote oil," "anthracene oil" and other materials. Its heating value is about the same as that of coal tar, *i.e.*, 15,000 to 16,500 B.t.u. per lb.

Gasoline. The composition of an average gasoline is: carbon, 83.5 to 85 per cent; hydrogen, 15.0 to 15.8 per cent; nitrogen plus sulfur plus oxygen, 0 to 1 per cent. The heating value is about 20,000 B.t.u. per lb. and requires for combustion about 15 lb. air (about 200 cu. ft.) per pound of gasoline.

Specifications for different grades of gasoline may be obtained from various departments of the U. S. Government, such as the U. S. Navy, the U. S. Army, etc.

Kerosene. The United States Government specification for kerosene to be used as a burning oil requires a distillation end point of not more than 625°F. and a flash point of not less than 115°F. The heating value of kerosene is from 20,000 to 21,000 B.t.u. per lb. The average composition of kerosene is: carbon, 84 per cent; hydrogen, 15 to 16 per cent; sulfur should not exceed 0.125 per cent (U. S. Government specification).

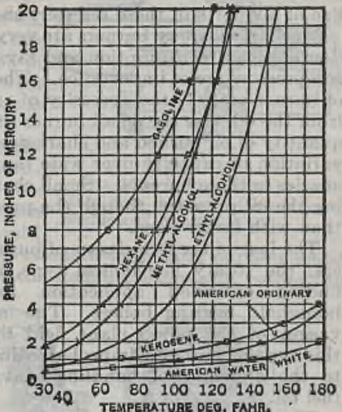


FIG. 19.—Vapor pressures of various substances. (*From Marks, "Mechanical Engineers' Handbook."*)

Alcohol and Benzol. Table 16 gives pertinent combustion data for various alcohol and benzol fuels.

Figure 19 shows the vapor pressures of a number of common liquid fuels: gasoline, hexane, methyl alcohol, ethyl alcohol, kerosene (three grades) (from Marks, "Mechanical Engineers' Handbook," McGraw-Hill, New York, 1941).

Table 16. Combustion Data for Various Alcohol and Benzol Fuels

Fuel	O_2 required for combustion, lb. per lb.	Air required for combustion, lb. per lb.	Products of combustion, lb. per lb.			Approximate higher heating value, B.t.u. per lb.
			CO_2	H_2O	N_2	
Ethyl alcohol (C_2H_6O)	2.08	9.04	1.91	1.17	6.95	12,780
Methyl alcohol (CH_4O)	1.5	6.50	1.38	1.12	5.0	9,550
Benzol (C_6H_6)	3.1	13.32	3.39	0.69	0.24	18,000
Denatured alcohol	1.81	7.83	1.66	1.15	6.02	11,600
50 per cent mixture of alcohol and benzol	2.45	10.60	2.53	0.92	8.16	14,200

GASEOUS FUELS

BY WILBERT J. HUFF

DESCRIPTION OF VARIOUS FUEL GASES*

Acetylene. The use of acetylene as a fuel and illuminant is generally limited to cutting and welding operations requiring high flame temperature, to small isolated lighting plants, and to single "carbide" lights. It is made from calcium carbide and water. To avoid a dangerous rise in temperature, sufficient water (about ½ gal. per lb. carbide) should be present in the generator. The crude gas contains as impurities: ammonia, hydrogen sulfide, and phosphine, which must be removed before the gas can be used for indoor illumination. Acetylene forms explosive acetylides, particularly with copper, has wide explosive limits when mixed with air, and is explosive *per se* at pressures of 2 atm. and at 780°C. Its use as a liquid is therefore prohibited, and it is ordinarily dissolved in acetone under pressure. (Fulweiler, chapter on Industrial Gases, in Rogers, "Manual of Industrial Chemistry," 5th ed., Van Nostrand, New York, 1931. Vogel, "Das Acetylen," Spamer, Leipzig, 1923.)

Blau Gas. Blau gas is a specially prepared cracked-oil gas, processed to permit the use of pressures as high as 100 atm. In America its use has been replaced by liquefied petroleum gases.

Blast-furnace gas is a by-product from the smelting of iron ore with coke and preheated air in the blast furnace. These furnaces, consisting of a cylindrical shaft charged to the height of some 70 ft., contain a column of coke from the hearth to the top of the bosh, and above this are placed coke and iron ore alternately, with an appropriate flux. About one-third of the exit gases from the top of the furnace is used for heating the blast stoves, and the remainder may be burned under boilers or cooled and cleaned for use in gas engines.

The gas is also used alone or in admixture with other gases for underfiring coke ovens. The practice was introduced in the United States from Europe, the first installation occurring at the South Chicago plant of the By-Product Coke Corp. about 1929. The gas is carefully cleaned and washed before use. The low B.t.u. value requires regenerative preheating, as with producer gas. [Camp and Francis, chapter on Blast Furnace Gas, in Bacon and Hamor, "American Fuels," McGraw-Hill, New York, 1922. Wagner, "The Cleaning of Blast Furnace Gases," McGraw-Hill, New York, 1914. *Blast Furnace and Steel Plant*, **17**, 1048–52 (1929).]

Blue water gas (see pp. 2389–2390) is the product obtained by the interaction of steam and a highly heated low-volatile carbonaceous fuel. The gas is non-luminous, hence its name. The fuel, which may be an anthracite coal or a coke, is brought to a high temperature by blasting with air, after which the air supply is cut off and steam is injected. At first, this steam is admitted under the grates. As the lower portion of the bed cools rapidly, the steam injection is split to send a portion down through the bed, followed by a brief up-run purge to clear the ashpit of combustible gas. The blast of air is again admitted to restore the temperature, after which another steam run is made, and the alternate admission of air and steam is continued with brief cessations for the charging of more fuel. Usually the gases from the steam

---

* **Analysis of typical industrial fuel gases will be found on pp. 2363–2368.**

runs only are sent to the gas holder. To balance the thermal conditions within the set, as in making carbureted water gas or to increase the gas volume and density, producer gases from the end of the blow are sometimes sent to the holder. After several hours, in machines not automatically clinkered, it is usually necessary to stop the operation to remove the clinker and to restore the gas-making capacity of the machine. In recent years bituminous coal, preferably of non-caking properties, has come into extensive use as a fuel, and the resulting gases from the generator may contain a small amount of gas obtained from destructive distillation. [Fulweiler, chapter on City Gas, *op. cit.* Meade, "Modern Gas Works Practice," Benn, London, 1921. Travers, *Trans. Inst. Chem. Eng. (London)*, **2**, 65 (1924). Morgan, "American Gas Practice" (privately printed), vol. I, Chap. 15, 1931. Fulweiler, *Proc. 1st Intern. Conf. Bituminous Coal*, p. 472; *U. S. Bur. Mines, Bull.* 203; *Tech. Papers* 246, 274, 284, 335; *Rept. Investigations* 2183. Pettyjohn, *Am. Gas Assoc. Proc.*, 1930, p. 1535.]

Carbureted Water Gas (see pp. 2390–2394). As the thermal content of blue water gas is too low to meet present public requirements, the gas is carbureted with oil gas which is formed by the thermal decomposition (see p. 2392) of the oil usually in supplementary shells connected in series to the generator shell of the water gas machine. These shells, designated (in order of sequence) the carburetor and the superheater, are filled with checkerwork surfaces of firebrick when gas oil is used and are heated by the sensible heat of the blue and the blast gases and by the combustion of the carbon monoxide and other fuel gases contained in the blast gases, using secondary air injected at the top or entrance of the carburetor, which receives the oil during the gasmaking periods. When heavy oil is used, much or substantially all the checkerwork may be removed from the carburetor, and some of the heavy oil may be thrown upon the top of the generator fire during the run, with the simultaneous admission of some air to burn off the carbon.

Coal gas (retort) (see p. 2397) is obtained by the destructive distillation of bituminous coal of suitable characteristics, usually designated as a gas coal, in a closed, highly heated retort of fire clay or silica. The retort may be placed in a horizontal, vertical, or inclined position. Other products, such as tar, ammonia, and water, are driven off at the same time and removed by subsequent cooling and other treatment of the gas stream. (Fulweiler, chapter on City Gas, in Rogers, "Manual of Industrial Chemistry," 5th ed., Van Nostrand, New York, 1931. Meade, *loc. cit.* Morgan, *loc. cit.* Also publications of the American Gas Institute and the American Gas Association, particularly the Carbonization Committee of the latter. See also references under Coke-oven Gas.)

Coke-oven gas is a coal gas derived from the distillation of a bituminous coal generally known as a "coking coal," which is somewhat lower in volatile content than the usual gas coal. The carbonizing chamber is very much larger than the coal-gas retort and is built up from silica forms. The primary object of the process generally is the production of a metallurgical coke with gas as a by-product. Tar, ammonia, and light oils are obtained from the gas. (Sperr, chapter on the Technology of Coke, in Bacon and Hamor, "American Fuels," McGraw-Hill, New York, 1922. See references under Coal Gas; also Haslam and Russell, "Fuels and Their Combustion," chapter on the Carbonization of Coal, with bibliography, McGraw-Hill, New York, 1926. "International Handbook of the By-product Coke Industry" by Gluud, Am. Ed. by Jacobson, Reinhold, 1932.)

Hydrogen. The use of hydrogen as a fuel is limited to certain special industrial purposes, such as certain welding and cutting operations.

Because of its application in the synthesis of ammonia and methanol and in the hydrogenation of oils, it is of considerable importance, however. One of the methods of production at present favored involves the catalytic oxidation of the carbon monoxide in blue water gas to carbon dioxide with steam. Electrolysis and the low-temperature fractionation of coal gas are also used. (Taylor, "Industrial Hydrogen," Reinhold, New York, 1921. Greenwood, "Industrial Gases," Sec. VI, Bailliere, Tindall and Cox, 1920. Pincass, "Die Industrielle Herstellung von Wasserstoff," Steinkopff, Dresden, 1933. See also "Fixed Nitrogen," Reinhold, New York, 1932.)

Natural gas is obtained by drilling through overlying strata to tap porous rocks, generally in sandstone of open texture or broken limestone, known as sands, above which a relatively impervious compact shale or cap rock has been folded in the form of a dome or inverted container. The gas is usually associated with petroleum or coal in carboniferous strata or more recent formations and is often trapped under high rock pressures. (Cross, "Handbook of Petroleum, Asphalt and Natural Gas," 1928 revision, Kansas City Testing Laboratory. Diehl, "Natural Gas Handbook," Metric Metal Works, Erie, Pa., 1927. *U. S. Bur. Mines, Bull.*, particularly 65, 82, 134, 163, 232; and other publications of the same bureau. Bacon and Hamor, "American Fuels," McGraw-Hill, New York, 1922. Dunstan *et al.*, "The Science of Petroleum," Oxford University Press, London, 1938.)

Oil Gas. The oil gas manufactured for public utility distribution on the Pacific coast is made by the gasification of oil with steam in a chamber containing hot checker brick. The heat is obtained by burning oil in the same chamber, and the process is a cyclic one, as in the water gas process. [Pike and West, *Ind. Eng. Chem.*, **21**, 104–109 (1929). Morgan, *op. cit.*]

There are a number of other oil gas processes, none of which are, however, widely used in the United States. The gases from oil refineries, particularly from the operation of various cracking processes for the production of gasoline from higher boiling oils, are utilized to some extent, and this utilization for a time tended to increase. Recent developments in the production of synthetic gasoline by catalytic polymerization of the refinery gases has, however, provided an excellent outlet. (Cross, *op cit.*, pp. 302 *ff.*; *cf.* p. 547. Leslie "Motor Fuels, Their Production and Technology," Reinhold, New York, 1923.)

Petroleum Gases. The fractionation of gasolines, particularly those recovered from natural gas, has made available a large quantity of light hydrocarbon liquids and vapors, chiefly propane and butane. These are available commercially for shipment under pressure for use as fuel gases and possess certain advantages, as in isolated situations where only a small quantity of gas is required, which may be serviced from pressure containers. In noninflammable mixtures with air, the vapors, particularly butane, are distributed by pipe line from a central point for the servicing of small communities. Propane may be distributed directly without admixture. (Thomas and Setrum, *Proc. Am. Gas Assoc.*, 1928, pp. 1284 *ff.* Odell, *U. S. Bur. Mines, Rept. Investigations* 2840, November, 1927. Oberfell, *Gas Age-Record*, Feb. 2, 1929. Odell, *U. S. Bur. Mines, Bull.* 294. *Repts. Am. Gas Assoc., Water-gas Committee.* Dunstan *et al.*, *loc. cit.* Morgan, *op. cit.*, Chap. 20.)

Producer gas (see pp. 2386–2389) is generated by blasting a deep hot bed of coal or coke continuously with a mixture of air and steam. Because of the large percentage of nitrogen in the gas thus obtained, its heating value is low. Solid fuels of widely different characteristics, including wood waste, may be employed in the process, and the variations in the thermal value of the resulting gas with the various volatile contents of the solid fuels are correspondingly large. Gas having as much as 180 B.t.u. per cu. ft. is made with high-volatile coals, while low-volatile coke with poor operations may give 110 B.t.u. per cu. ft. or lower. (Rambush, "Modern Gas Producers," Benn, London, 1923. *U. S. Bur. Mines, Bull.* 7 and 13. Haslam and Russell, chapter on Producer Gas with bibliography in "Fuels and Their Combustion," McGraw-Hill, New York, 1926. Reports, Carbonization Comm. and Subcomm. on Producer Gas, *Am. Gas. Assoc., Proc.*, 1927, 1928, 1929, 1930.)

Pintsch Gas. Oil may be cracked in closed iron retorts, externally heated. When this oil gas is compressed to 10 to 14 atm., it forms Pintsch gas, developed 60 years ago for the lighting of railway coaches. It is not used in America today.

Re-formed Gas. Although applicable to any gas transformed by suitable treatment, the term "re-formed gas" is ordinarily applied to lower thermal value gases obtained by the pyrolysis and steam decomposition of high thermal value gases, such as natural gas or oil refinery gas. The steam minimizes carbon loss and possesses other advantages. (*Cf.* Odell, *U. S. Bur. Mines, Bull.* 301; *Tech. Paper* 483; *Rept. Investigations* 2973 and 2991. Also Schlegel, *Am. Gas Assoc, Proc.*, 1930, p. 1466. Morgan, *op. cit.*, Chap. 19.)

Impurities and By-products

Sulfur Impurities in Gaseous Fuels. The chief sulfur impurity in gaseous fuels is **hydrogen sulfide.** Certain natural and petroleum gases are found free from this undesirable impurity, but its presence may otherwise be expected rather universally in raw fuel gases in amounts which may range from approximately 100 grains per 100 cu. ft. in blue and carbureted water gas to several hundred grains per 100 cu. ft. in coal and coke-oven gases. In refinery gases from sulfur crudes and natural gases from sulfur-bearing regions the concentration may be several thousand grains per 100 cu. ft. For public utility purposes laws generally forbid the distribution of gas containing any hydrogen sulfide. In manufactured gas practice this impurity may be removed by hydrated iron oxide masses or by certain liquid purification processes. The liquid processes ordinarily involve absorption in an aqueous alkaline solution made up with sodium carbonate or ammonia, although alkaline phosphates or strong organic bases are used. The absorbed hydrogen sulfide is expelled by a stream of air or by heat and vacuum or may be converted to sulfur by the action of air and a catalyst in another stage of the process.

The other important sulfur impurity is ordinarily **carbon disulfide,** which may contribute as much as 80 per cent of the organic sulfur present in manufactured fuel gases. In such gases, however, the total organic sulfur is relatively small, usually much less than the 30 grains per 100 cu. ft. of gas limit permitted by most states.

Other sulfur compounds, which may be present in small amounts, are the thiophenes, carbon oxysulfide, mercaptans, thioesters, and organic sulfides. [Repts. Am. Gas Assoc. Purification Committee. Morgan, "American Manufactured Gas Practice," 1931 ed., Chap. 23. "International Handbook of the By-Product Coke Industry," by W. Gluud, Am. ed. by Jacobson, Chap. 9, Reinhold, New York, 1932. Huff and Milbourne, *Am. Gas Assoc., Proc.*, 1930, p. 856; *Ind. Eng. Chem.*, **22,** 1213 (1930). Dotterweich and Huff, *Am. Gas Assoc., Proc.*, 1938, p. 699. Huff and Holtz, *Am. Gas Assoc., Proc.*, 1927, pp. 1431, 1436; *Ind. Eng. Chem.*, **19,** 1268 (1927), and **22,** 639 (1930). Denig and Powell, Liquid Purification, *Am. Gas Assoc.*,

INDUSTRIAL FUEL GASES

Table 17. Analyses of Fuel Gases*

Gas	Constituents of gas, % by volume									Sp. gr. air = 1.000	Cu. ft. of air required for combustion of 1 cu. ft. gas	B.t.u. per cu. ft. gross	B.t.u. per cu. ft. net	Products of combustion, cu. ft. per cu. ft. gas				Ultimate CO_2, %	Net B.t.u. per cu. ft. of the products of combustion	Flame temperature corrected for dissociation
								Illuminants												
	CO	CO_2	H_2	N_2	O_2	CH_4	C_2H_6	C_2H_4	C_6H_6					CO_2	H_2O	N_2	Total (dry)			
Natural gas, Texarkana	....	0.80	....	3.20	...	96.00	.....	...	...	0.57	9.17	967	873	0.97	1.92	7.29	8.26	11.7	80.2	3580
Natural gas, Cleveland	....	.....	....	1.30	...	80.50	18.20	...	...	0.65	10.70	1131	1025	1.17	2.16	8.50	9.67	12.1	81.1	3600
Natural gas, Oil City, Pa.	....	.....	....	1.10	...	67.60	31.30	...	...	0.71	11.70	1232	1120	1.30	2.29	9.26	10.56	12.3	81.7	3620
Retort coal gas (horizontal)	8.6	1.5	52.5	3.5	0.3	31.4	.....	1.1	1.1	0.42	5.00	575	510	0.50	1.21	3.99	4.49	11.2	83.5	3665
Coke-oven gas	6.3	1.8	53.0	3.4	0.2	31.6	.....	2.7	1.0	0.42	5.19	588	521	0.51	1.25	4.13	4.64	11.0	82.7	3660
Coke-oven gas, Koppers ovens	6.8	2.2	47.3	6.0	0.3	33.9	.....	2.6	0.9	0.44	5.23	591	525	0.54	1.23	4.19	4.73	11.4	82.3	3650
Carbureted water gas	33.4	3.9	34.6	7.9	0.9	10.4	.....	6.7	2.2	0.65	4.37	536	496	0.74	0.75	3.54	4.28	17.2	88.5	3815
Blue water gas	42.8	3.0	49.9	3.3	0.5	0.5	.....	...	...	0.53	2.26	308	281	0.46	0.51	1.82	2.28	22.3	89.7	3800
Theoretical water gas	50.0	.....	50.0	.....	...	.....	.....	...	...	0.52	2.39	325	298	0.50	0.50	1.89	2.39	20.9	90.3	3830
Anthracite producer gas	24.0	7.5	16.5	50.2	0.6	1.2	.....	...	...	0.85	1.05	134	124	0.33	0.19	1.36	1.69	19.5	65.6	3000
Bituminous producer gas	27.0	4.5	14.0	50.9	0.6	3.0	.....	...	...	0.86	1.24	150	140	0.35	0.19	1.49	1.84	19.0	69.2	3160
Blast-furnace gas	27.5	10.0	3.0	58.0	1.0	0.5	.....	...	...	1.00	0.78	102	100	0.38	0.04	1.21	1.59	23.9	61.0	2800
Oil gas (Protero 1920)	6.8	1.0	59.2	2.7	0.1	25.4	.....	3.8	1.0	0.35	4.91	575	510	0.47	1.21	3.91	4.38	10.7	84.2	3725

---

* From "Combustion," 2d ed., Table 20, p. 34, American Gas Association, 1926. Reproduced by permission.

Table 18. Composition of Natural Gases in Various Parts of the United States*

City	Constituents					Total
	CO_2	CH_4	C_2H_6	N_2	H_2S	
Little Rock, Ark	1.0	96.7	0.0	2.3	...	100.0
Bartlesville, Okla	2.8	89.1	4.2	3.9	...	100.0
Cleveland, Ohio	0.0	83.6	15.7	0.7	...	100.0
Los Angeles, Cal	25.1	59.1	14.5	1.3	...	100.0
Palestine, Ill	0.5	95.6	0.0	3.9	...	100.0
Buffalo, N. Y	0.0	88.1	11.5	0.4	...	100.0
Kansas City, Mo	0.8	84.1	6.7	8.4	...	100.0
Charleston, W. Va	0.0	76.8	22.5	0.7	...	100.0
Olney, Ill	0.0	37.5	59.6	1.7	1.2	100.0

* Burrell and Robertson, Compressibility of Natural Gas and Its Constituents, with Analyses of Natural Gas from 31 Cities in the United States, *U. S. Bur. Mines, Tech. Paper* 158, 1917.

Table 19. Typical Analyses of Gas from Koppers By-product Coke Ovens*

	CO_2	Illuminants	O_2	CO	H_2	CH_4	N_2	B.t.u. per cu. ft.	Sp. gr.
Straight gas Before removing benzol	2.2	3.5	0.3	6.8	47.3	33.9	6.0	591	0.44
Straight gas After removing benzol	2.2	2.6	0.3	6.9	47.8	34.2	6.0	562	0.42
Enriched gas Before removing benzol	2.6	4.3	0.2	6.3	46.3	35.0	5.3	630	0.45
Enriched gas After removing benzol	2.6	3.2	0.2	6.4	46.8	35.4	5.4	605	0.42
Lean gas Before removing benzol	2.1	2.0	0.3	6.0	57.0	27.0	5.6	531	0.38
Lean gas After removing benzol	2.1	1.0	0.3	6.1	57.5	27.3	5.7	500	0.35

* These analyses are taken from different plants and consequently represent different coals and somewhat different operating conditions.

Table 20. Typical Composition of Commercial Gases, Per Cent by Volume

Commercial gases	Carbon dioxide (CO_2)	Illuminants	Oxygen (O_2)	Carbon monoxide (CO)	Hydrogen (H_2)	Methane (CH_4)	Ethane (C_2H_6)	Nitrogen (N_2)	B.t.u. per cu. ft.	Candlepower
†Blau gas	0.0	51.9	.....	0.1	2.7	44.1	...	1.2	1704	48.2
†Acetylene gas	.....	96.0	0.8	.....	.....	.....	...	3.2	1350	225.0
†Gasoline gas	.....	1.5	18.5	.....	.....	C_6H_{14} = 10.3		69.7	514	16.0
†Blast-furnace gas	12.8	.....	0.1	26.5	3.5	0.2	...	56.9	100	
*Coal gas, Germany	3.01	5.09	0.65	8.88	46.20	C_nH_{n+2} 34.02		2.15	700	
*Coal gas, United States	0.45	4.00	1.50	6.00	46.00	40.00		2.05	755	
*Lignite gas	11.51	3.25	1.76	17.24	45.16	15.59		5.49	500	
*Wood-distillation gas	17.40	6.00	.....	31.50	18.30	21.70		5.10		
*Water gas	4.00	0.00	1.50	45.50	45.00	2.00		2.00	350	
*Pressure still gas	0.22	10.43	0.22	0.45	9.30	73.92		5.46		
*Oil gas	0.30	38.10	.....	0.50	3.40	57.70		.....	1390	
*Producer gas	2.50	.....	.....	26.00	3.00	0.5		56.0		
*Still gases from lubricating oil stills	3.0	3.5	.....	16.5	.....	77.0				

* From Cross, "Handbook of Petroleum, Asphalt and Natural Gas," Kansas City Testing Laboratory, *Bull.* 25, p. 547. 1928.

† From Fulweiler in Rogers "Industrial Chemistry," p. 474, Van Nostrand, New York, 1921.

Table 21. Industrial Fuel Gases Commonly Available

Component gases	CO	H_2	CH_4	C_2H_6	Illuminants	CO_2	O_2	N_2	Blue gases	Hydrocarbons	Inerts	B.t.u. per cu. ft.	Sp. gr.
Producer gas	25.3	13.2	0.4	....	....	5.4	0.5	55.2	38.5	0.4	61.1	129	0.878
Blue gas	38.3	52.8	0.4	....	....	5.5	0.1	2.9	91.1	0.4	8.5	295	0.521
Blue gas, 250 B.t.u. per cu. ft.	34.8	42.0	0.4	....	....	5.5	0.2	17.1	76.8	0.4	22.8	250	0.618
Coal gas	5.9	53.2	29.6	....	2.7	1.4	0.7	6.5	59.1	32.3	8.6	548	0.375
Coal gas – producer gas	7.3	51.3	26.6	....	3.2	2.1	0.9	8.6	58.6	29.8	11.6	527	0.431
Carb. water gas, no blow run	32.6	38.2	8.0	2.2	8.6	3.4	0.5	6.5	70.8	18.8	10.4	530	0.643
Carb. water gas, with blow run	24.1	32.5	9.0	2.2	10.3	4.6	0.6	16.7	56.6	21.5	21.9	530	0.703
Carb. water gas, low gravity	21.9	49.6	10.9	2.5	6.1	3.6	0.4	5.0	71.5	19.5	9.0	536	0.539
Carb. water gas, no blow run	25.7	32.6	8.6	7.2	13.1	6.1	0.6	6.1	58.3	28.9	12.8	603	0.675
Reformed gas, refinery	14.3	50.9	15.9	5.0	2.4	2.2	0.9	8.4	65.2	23.3	11.5	530	0.497
Reformed gas, natural	14.5	43.5	29.7	1.7	1.0	2.4	0.2	7.0	58.0	32.4	9.6	530	0.472
Refinery gas, high paraffin	1.2	6.1	4.4	72.5	15.0	...	0.2	0.6	7.3	91.9	0.8	1644	0.990
Refinery gas, high olefin	1.2	13.1	23.3	21.7	39.6	0.1	1.0	....	14.3	84.6	1.1	1468	0.890
Natural gas	....	....	78.8	14.0	....	0.4	...	6.8	....	92.8	7.2	1013	0.635
Natural gas	....	....	82.8	16.3	0.8	0.1	...	....	....	99.9	0.1	1145	0.650
Butane, commercial	....	....	....	....	....	...	...	....	....	100.0	....	3200	1.950
Propane, commercial	....	....	....	....	....	...	...	....	....	100.0	....	2550	1.523

* From Perry, "Gas Mixtures," *Proc. Am. Gas Assoc.*, p. 809, 1931.

Table 22. Fractionation Analyses*

(Percentage by volume on air-free basis)

Constituent	Low-temperature carbonization				High-temperature carbonization. Koppers ovens (scrubbed to remove benzene)	Water gas				Oil-cracking process	Mixed coal and water gas¶
	Karrick retort			McIntire retort		From coke		From anthracite			
	Internal heating by steam, 600°C.	External heating 500°C.	External heating 850°C.	External heating		Blue gas	Enriched gas	Blue gas	Enriched gas		
Carbon dioxide	18.6	15.7	8.6	8.8	2.2	3.0	0.9	3.9	3.3	(CO_2) 0.0	(CO_2) 2.63
Hydrogen sulfide	......	......	......	......	......	......	......	......	......	......	(O_2) 0.81
Nitrogen	1.05	0.0	0.9	1.5	1.0	9.5	6.8	1.6	4.2	0.0	4.23
Hydrogen	10.1	14.5	55.95	23.4	57.2	43.8	37.4	50.57	38.4	19.0	37.33
Carbon monoxide	11.5	8.6	14.9	4.7	5.8	43.2	35.0	42.9	31.0	0.8	13.25
Methane	40.3	43.75	15.0	46.4	29.2	0.5	8.1	1.0	12.7	27.1	31.13
Ethylene	1.9	1.30	1.00	1.60	2.50	0.0	6.7	0.03	6.9	33.5	6.05
Ethane	7.4	7.6	1.32	6.7	1.35	0.0	1.30	0.0	1.05	3.5	2.10
Propylene	1.8	1.4	0.58	1.30	0.29	......	1.50	......	0.87	8.7	0.60
Propane	2.7	3.2	0.46	2.0	0.11	......	0.25	......	0.08	0.3	0.43
Butylene	1.4	1.2	0.37	0.65	0.18	......	0.75	......	0.45	4.8	0.11
Butane	0.95	0.85	0.17	0.85	0.04	......	0.0	......	0.0	0.1	
Liquid hydrocarbons	2.3	1.9	0.75	2.1	0.13	......	1.30	......	1.05	2.2	(C_6H_6) 1.33
Total	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00
Liquid hydrocarbons†	0.12	0.10	0.04	0.11	0.006	......	0.055	......	0.044	0.097	
Calorific value of illuminants‡	2748	......	......	......	1840§	......	2077	......	1972	1935	2141

---

* Yant and Frey, *Ind. Eng. Chem.*, 19, 1359 (1927).

† Same as last item in analysis but determined as cubic centimeters of liquid per liter of gas.

‡ Gross B.t.u. per cubic foot at 60°F. and 29.92 in. Hg.

§ Calorific value of the same gas plus 0.7 per cent benzene—2108. This gas was not available unscrubbed. The value of 0.7 was determined in an available but less representative sample.

¶ Analysis by Burrell and Robertson. Not calculated air free.

Table 23. Composition of Hydrocarbon Group in Different Types of Gas*

(Figures in per cent)

Constituent	Low-temperature carbonization. Karrick retort†	Oil cracking	Carbureted water gas		High-temperature carbonization. Koppers ovens
			From coke	From anthracite	
	I	II	III	IV	V
A					
Ethylene	10.3	63.4	56.6	66.0	48.8
Ethane	40.3	6.5	11.1	10.0	26.3
Propylene	9.8	16.4	12.8	8.4	5.7
Propane	14.7	0.5	2.2	0.8	2.1
Butylene	7.6	9.0	6.2	4.3	3.5
Butane	5.2	0.2	0.0	0.0	0.8
Liquid hydrocarbons (as vapor)	12.1	4.0	11.1	10.5	12.8
Total	100.0	100.0	100.0	100.0	100.0
B					
Ethylene + ethane	50.6	69.9	67.7	76.0	75.1
Propylene + propane	24.5	16.9	15.0	9.2	7.8
Butylene + butane	12.8	9.2	6.2	4.3	4.3
C					
$\frac{\text{Ethylene} \times 100}{\text{Ethylene} + \text{ethane}}$	20.4	90.5	83.6	86.8	65.0
$\frac{\text{Propylene} \times 100}{\text{Propylene} + \text{propane}}$	40.0	97.0	85.3	91.3	73.1
$\frac{\text{Butylene} \times 100}{\text{Butylene} + \text{butane}}$	59.3	98.0	100.0	100.0	81.5
Liquid hydrocarbons, per cent unsaturated	75.0‡	100.0§	100.0¶	100.0¶	100.0
Hydrogen, per cent in gas	10.1	19.0	37.4	38.4	57.2

* Yant and Frey, *Ind. Eng. Chem.*, **19**, 1359 (1927).

† Per cent in total hydrocarbons excluding methane.

‡ Separate determination; no aromatics detected.

§ Roughly 25 per cent aromatic, 75 per cent olefinic; very little saturated.

¶ Assumed; largely benzene.

Table 24. Comparison of Fractionation Analysis with Usual Orsat Analysis*

(Figures in per cent)

Constituent	Low-temperature carbonization				High-temperature carbonization†		Carbureted water gas			
	Karrick retort				Koppers ovens		From coke		From anthracite	
	Steam at 500°C.		Steam at 850°C.							
	A	*B*	*A*	*B*	*A*	*B*	*A*	*B*	*A*	*B*
$CO_2 + H_2S$	22.3	22.3	11.5	11.5	2.4	2.4	0.9	0.9	3.3	3.3
N_2	1.2	0.0	0.6	0.0	0.9	0.5	6.8	6.9	4.2	3.6
O_2	0.2	0.0	0.2	0.4	0.0	0.0	0.0	0.0	0.0	0.0
H_2	3.3	3.3	43.1	43.0	56.4	56.5	37.4	37.2	38.4	38.6
CO	9.8	9.6	13.0	13.1	5.7	5.6	35.0	35.0	31.0	31.3
CH_4	23.5	25.6	21.4	23.8	29.3	29.6	7.9	7.9	12.6	12.7
C_2H_6	30.0	29.4	7.5	5.8	1.5	1.3	1.8	1.7	1.2	1.0
Illuminants	9.7	9.8	2.7	2.9	3.8	4.1	10.2	10.4	9.3	9.5
Total	100.0	100.0	100.0	100.5	100.0	100.0	100.0	100.0	100.0	100.0

A = fractionation analysis calculated into terms of Orsat analysis.

B = actual Orsat analysis of same sample.

* Yant and Frey, *Ind. Eng. Chem.*, **19**, 1360 (1927).

† A different sample of unscrubbed gas from same ovens.

Proc., 1933, p. 913. Denig, *Am. Gas Assoc., Proc.*, 1933, p. 903. Sperr, *Am. Gas Assoc., Proc., Tech. Sec.*, 1921, p. 282; 1923, p. 1200. Huff, *Proc. 2d Intern. Conf. Bituminous Coal* II, 1928, p. 814. Bottoms, *Ind. Eng. Chem.*, **23**, 501 (1931).]

In the **iron oxide process** for the removal of hydrogen sulfide, as generally practiced in America, the oxide, which is ordinarily rusted iron borings, bog-iron ore, or a by-product from the refining of bauxite, is mixed with wood shavings. Ground corncobs also make an excellent support, although wood is to be preferred. About 25 lb. oxide are used per bushel of shavings. With boring oxides, the rusting may take place on the shavings and a prepared purifying material may be offered. The shavings serve as a fluffing agent, permitting the ready passage of gas through the material and serving to expose the oxide to the gas stream. Oxides vary enormously in their purifying properties and this property is not dependent upon the iron content but upon the chemical and physical state of the oxide. Humidity effects play a most important part in the reaction. Data developed by Huff and Milbourne (*loc. cit.*) indicate that the fouling reaction is best maintained at a humidity of about 65 per cent. The fouled oxide is revivified by the action of oxygen in air in the presence of moisture; the study above mentioned showed that high humidities, just sufficiently under saturation to avoid the precipitation of liquid moisture, are best. Spent oxide may be revivified *in situ* simultaneously with fouling by admitting a small quantity of air with the gas, about 0.5 per cent more oxygen (on total volume basis) than is required for the reaction $2H_2S + O_2 \rightarrow 2S + 2H_2O$. The reaction is, however, more complex than this equation indicates. The revivification process is relatively slow and, when practiced simultaneously, humidity conditions favorable to it must prevail. The oxide may be revivified by removal from the box or by by-passing the box and passing air through it, but care must be taken to avoid overheating and explosions.

The oxide is placed in cast-iron or steel boxes of large cross section in order to effect contact with a minimum of pressure drop. Wooden trays are placed in the boxes and the oxide-shavings mixture is placed in two or three layers usually from 18 to 30 in. deep. If only a single layer is used, it may be about 4 ft. deep. The oxide is distributed over at least three or four boxes in series, and suitable valve arrangements make it possible to change the order to permit the placing of a badly fouled box out of the line or in a position where it may not receive much hydrogen sulfide and so be revivified.

A formula which allows for the several factors in calculating the area of the boxes is known as the Steere formula [*Bull.* 37, Steere Engineering Co., Detroit, Mich. or *Gas Age*, **43**, 227 (1919)]. This is

$$G = \frac{3000 \times (D + C) \times A}{S}$$

or

$$A = \frac{G \times S}{3000 \times (D + C)}$$

where G is the maximum amount of gas in cubic feet corrected to 60°F., to be purified per hour; D is the total depth in feet of oxide through which the gas passes consecutively. When a single catch box is used for two or more parallel sets of boxes, disregard the catch box in obtaining D; A is the cross-sectional area in square feet of oxide through which the gas passes in any one layer in the series; C is the factor: 4 for a two-box, 8 for a three-box, and 10 for a four-box series. Where a single catch box is used for two two-box sets, $C = 6$; S is the factor for grains hydrogen sulfide per 100 cu. ft. gas entering the purifiers as follows:

Grains H_2S per 100 Cu. Ft. Unpurified Gas	Factor S
1000 or more	720
900	700
800	675
700	640
600	600
500	560
400	525
300	500
200 or less	480

By-products in Gas Manufacture. In the carbonization of coal for the production of gas and coke, there are obtained as by-products: ammonia, light oils of the aromatic-hydrocarbon series, and tar. Other by-products, such as cyanides, are sometimes obtained.

Ammonia. The ammonia yields may vary from 20 to 26 lb. ammonium sulfate equivalent per ton of coal carbonized. In the distillation of the coal, there is condensed generally from 10 to 14 gal. gas liquor per ton of coal carbonized. This may contain about one-half the ammonia, of which 75 to 80 per cent will probably be in the **free** form and the remainder in the form of "fixed salts." The free form consists of ammonium salts such as the carbonate, the bicarbonate, the carbamate, the sulfide, the hydrosulfide, the cyanide, and the polysulfide. Some of these salts exist only in very small quantities if at all. The *free ammonia* in the liquor is chiefly carbonates and sulfides. All these decompose with heat and ammonia may be recovered by steam distillation. The *fixed ammonium salts* must be decomposed by the action of a base, such as lime. These salts may consist of the chloride, the sulfocyanide, the sulfate, the thiosulfate, the sulfite, the ferrocyanide, and the acetate. In addition there are phenols, amines, and other nitrogenous substances in the liquor. The fixed salts are largely the chloride, with some sulfocyanide and sulfate, but the composition depends upon the heating of the liquor in which chemical changes tending to build up fixed compounds occur.

The ammonia which remains in the gas is recovered by water scrubbing or by washing with a dilute solution of sulfuric acid in a lead-lined saturator. Today, the greater part of the ammonia recovered from the carbonization of coal is fixed as the sulfate. Free-ammonia and aqueous-ammonium solutions of commerce are more conveniently prepared in high purity by synthetic-ammonia processes. (Lunge, "Coal Tar and Ammonia," Van Nostrand, New York, 1916. Calvert, "The Manufacture of Sulfate of Ammonia and Crude Ammonia," Benn, London, 1917. Parrish, "Design and Working of Ammonia Stills," Van Nostrand, New York, 1924. Meade, "Modern Gas Works Practice," Benn, London, 1921. Gluud, "International Handbook of the By-Product Coke Industry," Am. ed. by Jacobson, Chap. 10, 1932. Morgan, *op cit.*, Chap. 22.)

The Recovery of Light Oils. The light oils, chiefly benzene, toluene, and xylene, existing in certain industrial gases, particularly coal gas and carbureted water gas, may be recovered by scrubbing these gases with wash oil and subsequently distilling the oil solution to volatilize the light oil dissolved therein

Table 25. Analysis of Light Oil

Source	Benzene, %	Toluene, %	Solvent, %	Wash loss, %	Res. below 200°, %	Paraffins in benzene		Paraffins in toluene		Gal. per 1000 cu. ft. original gas			Remarks
						Light oil, %	Benzene, %	Light oil, %	Toluene, %	Light oil	Benzene	Toluene	
Coal gas (horizontals):													
Denver	44.8	20.9	10.2	10.0	13.4	0.7	1.56	None	.....	0.253	0.113	0.053	
Seattle	68.5	17.7	3.7	2.5	7.2	0.2	0.29	0.2	1.3	0.304	0.208	0.054	
Newark	58.8	12.8	13.1	5.7	8.4	0.7	1.19	0.5	3.91	0.403	0.237	0.051	
Coke-oven gas:													
Chamber ovens (Seattle)	65.4	20.7	1.8	5.8	5.8	0.4	0.61	0.1	0.48	0.226	0.148	0.047	
High-volatile coal, Toledo (lean)	55.7	12.6	11.2	7.4	12.8	0.3	0.54	None	.....	0.3214	0.178	0.040	
Principally Kentucky (rich)	52.2	18.6	11.0	8.8	9.1	0.3	0.57	None	.....	0.3461	0.181	0.064	
Run of oven gas:													
Pittsburgh coal + 15% L.V.	44.3	18.2	11.2	11.6	14.7	None	.....	None	.....	0.3600	0.159	0.066	Boiling-point curves
Run of oven gas, Kentucky coal + 15% low volatile	59.5	14.4	5.8	5.1	15.2	None	.....	None	.....	0.4000	0.238	0.058	
Oil gas:													
Los Angeles	75.2	11.6	.....	1.4	7.2	4.6	6.1	*	.....	0.198	0.149	0.023	* No pure toluene cut obtained
Terre Haute	29.38	17.24	8.86	10.00	31.82	0.94	3.2	0.38	2.2	1.56	0.458	0.269	138% naphthalene
Vertical-retort gas:													
Providence	34.04	14.38	31.12	7.19	9.41	1.94	5.70	1.92	13.35	0.320	0.109	0.046	
Rochester	16.3	5.0	15.6	6.9	54.4	1.5	8.1	0.3	6.16	1.563	0.255	0.078	
	32.4	9.9	31.0	13.7	9.4	3.0	8.1	0.6	6.16	0.787	0.255	0.078	
Water gas:													
Chicago (Division St.)	45.5	20.8	18.5	14.7	.....	0.25	0.55	0.25	1.20	0.527	0.240	0.110	
Providence	35.06	24.57	14.78	13.31	12.28	None	....	None	.....	0.311	0.109	0.076	
Washington	22.1	18.8	15.1	15.5	28.2	0.3	1.36	None	.....	0.413	0.091	0.078	
Denver	52.6	23.5	4.1	11.5	8.0	0.3	0.57	None	.....	0.304	0.160	0.072	
Seattle	50.2	31.1	4.2	8.4	6.1	None	....	None	.....	0.508	0.255	0.158	
Newark	39.9	21.6	14.1	11.4	12.4	0.5	1.25	0.1	0.46	0.508	0.203	0.110	

The wash oil generally used in America for absorbing the light oil is a petroleum product usually known as straw oil, of which at least 90 per cent should distill between 250° and 350°C. It should have a specific gravity of less than 0.88 at 15°C. and be readily fluid at 4°C. This oil is circulated countercurrent to the gas flow through high slat-filled towers, and the quantity used is sufficient to keep the light-oil volume absorbed to between 2 and 3 per cent of the absorbing oil. In European practice, heavy tar oils are used as the absorbent.

From the absorbing towers the oil is sent to a continuous, direct-contact-type steam still where it is substantially stripped of its light-oil content, after which it is cooled and ultimately returned to the circulating system to be again pumped over the absorbing towers. Heat exchange between incoming oil and exit vapors and oil is practiced in order to secure good steam economy.

In earlier American practice, the light-oil vapors from the stripping still were condensed and charged into a so-called "crude" still. This was usually of the intermittent type and of large capacity to give good fractionation. Here it was broken into fractions, and from the still bottoms a certain amount of entrained wash oil, together with naphthalene and polymerized unsaturated hydrocarbons, was obtained. The still was of the indirect-steam type, equipped with column and dephlegmator. For "pure" products the fractions from the crude still were washed with acid and soda and redistilled.

More recent American practice prepares the pure products such as "pure" benzene, "pure" toluene, and motor fuel directly without passing through an intermediate crude still. Various treatments for the removal of undesirable gum-forming materials have been proposed, but the industry apparently usually depends upon agitation first with 66°Bé. commercial sulfuric acid in a lead-lined container, followed by caustic soda [Sperr, *Trans. Am. Inst. Chem. Engrs.*, **9**, 169 (1917). *Gas Age*, **41**, 393–397. Gluud, *op. cit.*, Chap. 11]. It is possible to conserve space and investment requirements by the use of stills and agitators of the continuous type. Consequently, there has recently been some interest in the development of such apparatus for benzol recovery.

Table 26. Average Composition of Coke-oven Light Oil

	Light oil, %	Coal, gal. per ton
Benzene	66	1.782
Toluene	15	0.405
Xylene	8	0.216
Other substances (principally hydrocarbons)	11	0.297
Total	100	2.700

In 1000 gal. of crude light oil from one source there were obtained: 680 gal. crude benzol, 140 gal. crude toluol, 50 gal. crude xylol, 55 gal. solvent naphtha, and 75 gal. residue containing entrained wash oil, polymerized hydrocarbons, naphthalene, and other materials.

Certain resin-forming hydrocarbons found in the solvent naphtha fraction of coal-tar light oils have recently received considerable attention. These are *styrene*, *indene*, and *coumarone*. The residue from the final distillation of certain chemically washed light-oil fractions boiling from 160° to 180°C. consists of *coumarone resin*. It is possible to manufacture this in various degrees of color and hardness. It finds use in lacquers, paints, printer's ink, paper sizing, and electrical insulation (*cf.* Gluud, *op. cit.*, p. 758).

Such resin-forming compounds, if not removed from manufactured gas, may lead to the deposition of gums and so cause stoppages at important points, such as pilots and regulators.

Vapor Pressure — Wash-oil Composition Relations — Light-oil Scrubbing. In the scrubbing operations involved in the solution of light oils in wash oil and in the consequent stripping operations, the aromatic hydrocarbons in solution in the wash oil follow Henry's law and Dalton's law. The application of these laws is somewhat complex, as may be expected from the complex nature of the mixtures. For ordinary design and operations it is usually sufficiently exact to assume that the aromatics obey Raoult's law; *i.e.*, $p = Px$, where p is the vapor pressure of the aromatic hydrocarbon (for example, benzol) in the solution; P is the vapor pressure of the pure hydrocarbon at the temperature in question; and x is its mol fraction. To evaluate x it is necessary to know not only the molecular weight of the other dissolved hydrocarbons but also, and particularly, the molal weight of the wash oil used. Since this is of petroleum origin, in the absence of direct determination, the molal weight may be approximated from the boiling point at atmospheric pressure by means of the relationships corresponding to those of the normal aliphatic hydrocarbons.

Specific Heat of Wash Oil. The specific heat of the wash oil is an important factor in the design and operation of light-oil scrubbing systems. In general, the specific heat of petroleum products is given by the expression

$$c = \frac{1}{\sqrt{d}}(0.388 + 0.00045t)$$

where c = specific heat, B.t.u./(lb.)(°F.)
d = specific gravity at 60°F/60°F.
t = temperature, °F.

This formula applies well for intermediate-base crude oils, is about 2 per cent high for oils from paraffin crudes, and is about 2 per cent low for oils from naphthene-base crudes. For highly cracked oils the values are not more than 4 per cent high (*Nat. Bur. Standards (U. S.), Misc. Pub.*, 97, Table 12, 1929).

For the **vapor pressures** of the more important constituents in light oils from gas-making processes, namely: benzol; phenol; toluol; *o*-, *m*-, and *p*-xylene; and naphthalene, see pp. 368 to 389 of Sec. 3.

Removal of Gasoline from Natural Gas. The removal of the lighter gasoline hydrocarbons from natural gas by oil scrubbing can be achieved on the same principles as are applied to the recovery of light oils from manufactured gas, due consideration being given to the partial pressures of the various constituents. Absorption is greatly favored by the elevated pressures sometimes available.

Tar. The pyrogenetic changes in the decomposition of coal during carbonization and in the conversion of oils to gas in oil and carbureted water gas making give rise to hydrocarbons of low vapor pressure which are deposited by the cooling gas stream and recovered as tar. The time of contact and temperature determine the nature of the hydrocarbons obtained. In coal-gas tar, they are ring hydrocarbons and their derivatives, and the higher temperatures now favored in carbureted water gas manufacture also produce a tar largely of the aromatic type.

With recent developments in the use of heavy oil as a carbureting agent, involving the throwing of oil upon the fire, there have appeared heavy water-gas tars of high free-carbon content. Some of these cause difficulty because of the stability of the emulsions created in their condensation. Under some conditions of operation, water gas tars made with bituminous coal as the generator fuel also give emulsion troubles.

For tar refining, as in the recovery of intermediates, tar acids, creosotes, and pitches, coal tar is used. Water gas tar in general lacks the nitrogenous bases and the phenols present in coal tar, and does not ordinarily give creosote oils and pitches of satisfactory quality, although it is widely used for blending with coal tar.

The usual yield of water gas tar is from 10 to 15 per cent of the oil used. Coal-tar yields vary with the mode of carbonization and the coal used. Coke ovens may yield from 6 to 11 gal., vertical retorts from 12 to 16 gal., and horizontal retorts from 10 to 14 gal. per ton of coal carbonized.

In the handling of tars, the change of the viscosity with temperature is very important. For typical coke-oven tars, this may be expressed as follows:

$$\log z = \log K - n \log T_c$$

where z is absolute-viscosity centipoises; $\log K$ is the constant varying from 6 to 11; n is the constant varying from 2 to 5; T_c is the temperature in degrees centigrade. [Huff, *Ind. Eng. Chem.* **15**, 1026 (1923).]

An examination of coal tar on a large scale has been made by Weiss and Downs [*Ind. Eng. Chem.*, **15**, 1022 (1923)]. Water gas tar has been examined by Downs and Dean [*Ind. Eng. Chem.*, **6**, 366 (1914)]. A recent extensive study of the composition of coal tar and light oil has been made by Fisher (*U. S. Bur. Mines, Bull.* 412). A bibliography of 255 references is appended.

Further sources: Warnes, "Coal Tar Distillation," 3d ed., Benn, London, 1923. Lunge, "Coal Tar and Ammonia," Van Nostrand, New York, 1916. Huff, *Chem. & Met. Eng.*, **26**, 113 (1922). Weiss, "Recent Progress in Science in Relation to the Gas Industry," Chap. 8, Am. Gas Assoc., 1926. Gluud, *op. cit.*, Chap. 12.

Table 27. Coke-oven Tar and Coke-oven Pitch

Characteristics of a Typical By-product Coke-oven Tar	
Moisture	1.50 %
Specific gravity (15°C.)	1.165
Lb. per gal	9.70
Coefficient of expansion per 1°C.	0.000685
B.t.u. per lb. (gross)	16,200
B.t.u. per gal. (gross)	157,140
Flash point	90°–100°C.
Free carbon	4.00 %
Sulfur	0.60 %
Ash	0.05 %
Distillation:	
Light oils (to 170°C.)	0.50 %
Middle oils (170°–270°C.)	19.50 %
Heavy oils (270°–350°C.)	20.00 %
Pitch	60.00 %

Characteristics of By-product Coke-oven Pitch	
Melting point	85°C.
Specific gravity (15°C.)	1.25
Flash point	160°–232°C.
B.t.u. per lb	15,500–16,000
Free carbon	20–25 %
Fixed carbon	30–35 %
Ultimate Analysis:	
Carbon	92–93 %
Hydrogen	3–5 %
Nitrogen	0.6–1 %
Oxygen	0.0–2 %
Sulfur	0.5–1 %
Ash	0.05–0.4 %

Table 28. Characteristics of Various Tars*

Genus	Species	Member	Submember	Sp. gr. at 15.5/15.5°C.	Engler viscosity sec. for 100 cc. at 100°C.	Organic insoluble in toluol and benzol, %	Fixed carbon, %	Ash, %	Tar acids, %	Sp. gr. of 235°–315° fraction A.S.T.M., distillation at 15.5°C.
Tars	Coal tars	Bituminous coal	Coke oven	1.15–1.26	30–100	3–17	14–40	0–0.5	1–4	1.02–1.05
			Horizontal gas retort	1.25–1.33	150–650	16–40	15–40	0–0.5	1–4	1.02–1.04
			Inclined gas retort	1.23–1.24	300	15–20	15–40	0–0.5	4–6	1.02–1.04
			Vertical gas retort	1.12–1.16	25–50	2–5.5	15–30	0–0.5	5–11	1.00–1.01
	....	....	Low-temperature processes	0.95–1.12	25–50	0–7	5–15	0–1.0	510–30	
			Blast furnace	1.15–1.30	80	10–25	10–30	10–15	5–10	
			Gas producer	1.12–1.20	100	5–25	10–35	0–2.5	3–9	
	Petroleum tars	Carbureted water gas	....	1.06–1.15	25–50	0.2–5.0	10–20	0–0.5	0	
		Oil gas	....	0.95–1.10	25	0–2.0	10–25	0–0.5	0	

---

* Table I from "Recent Progress in Science in Relation to the Gas Industry," Chap. VIII, Tar Refining and Tar Products, by Weiss, American Gas Association, 1926.

Following is the distillation record[*] for a typical tar from a by-product coke oven [excerpt from The Distillation of Coke-oven Tar and the Recovery of Some of Its Products by Huff, *Chem. & Met. Eng.*, **26**, 113 (1922)]:

Fraction No.	Temperature, °C.	Gal. per 1000 gal. tar	% by volume referred to the original tar volume
1	Atmospheric, 170	Oil, 14.5	1.45
	....	Water, 15.5	1.55
2	170–235	122	12.2
3	235–270	123	12.3
4	270–315	133	13.3
5	315–355	98	9.8
Residue and losses........	....	494	49.4

Weight of residue, 57 per cent of original tar.

The following products may be obtained from such tar:

	Gal. per 1000 gal. tar	% by volume
Ammonia water..................................	14.5 (0.6 lb. NH_3 est.)	
Light oil..................................	15 (naphthalene not determined)	1.5
Tar acids[*]	26	2.6

[*] The exact composition of these tar acids was not determined. The probable composition is crude phenol approximately 1 part cresylic acid approximately 2 parts.

Crude naphthalene	Lb. per 1000 gal.	% by weight
170°–235°C. fraction..................................	200	2.1
235°–270°C. fraction..................................	48	0.5
Total..................................	248	2.6

Crude anthracene cake[*]	Lb. per 1000 gal.		Weight, %	
	Without chilling	Additional recovery due to chilling	Without chilling	Additional recovery due to chilling
270°–315°C....................	46.5	9.4	0.48	0.098
315°–355°C....................	93.0	9.4	0.97	0.098
Total....................	139.5	18.8	1.45	0.196

[*] The exact percentage of anthracene was not determined.

Vapor Densities and Latent Heats of Vaporization of Coal-tar Fractions. The vapor densities of typical coal-tar fractions have been determined [Huff, *Ind. Eng. Chem.*, **10**, 1016 (1918)]. The mean-molal weights of the vapors approximate those of the important aromatic hydrocarbons found in

---

[*] In interpreting tar analysis, it should be remembered that it is not possible to attain in tar analyses the high precision of most chemical analyses. Moreover, from time to time the composition of the tar changes somewhat due to variations in the operating conditions and the coal supply.

the fraction, thus the 200° to 250°C. fraction has a vapor density approximating that of naphthalene. While the experimental values did not form a smooth curve, the following equation relating the molal weights with the corresponding boiling points under atmospheric pressure in degrees Kelvin (*i.e.*, °C. + 273) is sufficiently accurate for engineering purposes:

$$\log_{10} M = -2.058\left(\log_{10} \frac{1}{°\text{K.}}\right) - 3.476$$

From the molal weights, the latent heats of vaporization can be calculated by relations such as Trouton's rule or the Hildebrand function (see p. 638), which last for non-associated liquids (to which the class of coal-tar hydrocarbons appears to belong) may be expressed by the relationship:

$$\frac{L}{°\text{K.}} = -7.105\left(\log_{10} \frac{P}{°\text{K.}}\right) + 2.395$$

where L is the molal latent heat of vaporization in gram-calories per gram, and P is the pressure in atmospheres.

The following equations were derived by the writer from the experimentally determined vapor densities and give the latent heats of vaporization v in gram-calories per gram from the boiling point in °C. (t) at atmospheric pressure:

$$\log_{10} v = 2.062 - 0.000568t$$

and V in B.t.u. per lb. from the boiling point in °F. (T) at atmospheric pressure:

$$\log_{10} V = 2.328 - 0.0003157T$$

The Thermal Conductivity of Tar and Oil Films. (Calculated from data of Ernst in Royds, "Heat Transmission," p. 221, Van Nostrand, New York, 1921.) Range from 0.070 to 0.078 B.t.u./(sq. ft.) (hr.) (ft. thickness per °F.).

Specific Heats of Coal Tar and Tar Distillates. In calories per gram per °C. (= B.t.u. per pound per degree Fahrenheit).

Coal tars:

0.35 ± 0.05 at 40°C.
0.45 ± 0.05 at 20°C.

Tar distillates:

0.34 ± 0.04 at 15–90°C.

(From "International Critical Tables.")

Specific Heat of Pitch. (From Report of Fuel Research Board for year ending Mar. 31, 1928.)

	Softening point, ball and ring method, °C.	Mean specific heat, g.-cal./g./°C.	
		20°–50°C.	20°–85°C.
A. From vertical retort tar from Durham coal carbonized in Glover-West retorts. Tar distilled to 360°C. without steam	53	0.32_9	0.34_6
B. Medium-soft pitch from horizontal retort tar from Durham coal. Tar distilled and distillation completed with steam at 300°–320°C.	60	0.33_3	0.34_1
C. Pitch from low temperature tar. Tar distilled to 310°C. without steam	44	0.41_7	0.40_8

That the mean value for pitch *C* between 20° and 85°C. is lower than the value between 20° and 50°C. is explained as being due to the fact that this pitch has a softening point below 50°C. and latent heat is absorbed in the lower range of temperatures in melting the pitch. In the case of the first two samples the latent heat of fusion is supplied at a temperature higher than 50°C.

Table 29. Gas Com-

No.	Gas	Formula	Molecular weight*	Lb. per cu. ft.†	Cu. ft. per lb.†	Sp. gr. air = 1.000†	Heat of combustion‡ B.t.u.per cu. ft.	
							Gross	Net§
1	Carbon	C	12.01	....	....	....	....	....
2	Hydrogen	H_2	2.016	0.005327	187.723	0.06959	325.0	275.0
3	Oxygen	O_2	32.000	0.08461	11.819	1.1053		
4	Nitrogen (atms.)	N_2	28.016	0.07439¶	13.443¶	0.9718¶		
5	Carbon monoxide	CO	28.01	0.07404	13.506	0.9672	321.8	321.8
6	Carbon dioxide	CO_2	44.01	0.1170	8.548	1.5282		
	Paraffin series $C_nH_{2n} + 2$							
7	Methane	CH_4	16.041	0.04243	23.565	0.5543	1,013.2	913.1
8	Ethane	C_2H_6	30.067	0.08029¶	12.455¶	1.04882¶	1,792	1,641
9	Propane	C_3H_8	44.092	0.1196¶	8.365¶	1.5617¶	2,590	2,385
10	n-Butane	C_4H_{10}	58.118	0.1582¶	6.321¶	2.06654¶	3,370	3,118
11	Isobutane	C_4H_{10}	58.118	0.1582¶	6.321¶	2.06654¶	3,363	3,105
12	n-Pentane	C_5H_{12}	72.144	0.1904¶	5.252¶	2.4872¶	4,016	3,709
13	Isopentane	C_5H_{12}	72.144	0.1904¶	5.252¶	2.4872¶	4,008	3,716
14	Neopentane	C_5H_{12}	72.144	0.1904¶	5.252¶	2.4872¶	3,993	3,693
15	n-Hexane	C_6H_{14}	86.169	0.2274¶	4.398¶	2.9704¶	4,762	4,412
	Olefin series C_nH_{2n}							
16	Ethylene	C_2H_4	28.051	0.07456	13.412	0.9740	1,613.8	1,513.2
17	Propylene	C_3H_6	42.077	0.1110¶	9.007¶	1.4504¶	2,336	2,186
18	n-Butene (Butylene)	C_4H_8	56.102	0.1480¶	6.756¶	1.9936¶	3,084	2,885
19	Isobutene	C_4H_8	56.102	0.1480¶	6.756¶	1.9336¶	3,068	2,869
20	n-Pentene	C_5H_{10}	70.128	0.1852¶	5.400¶	2.4190¶	3,836	3,586
	Aromatic series $C_nH_{2n} - 6$							
21	Benzene	C_6H_6	78.107	0.2060¶	4.852¶	2.6920¶	3,751	3,601
22	Toluene	C_7H_8	92.132	0.2431¶	4.113¶	3.1760¶	4,484	4,284
23	Xylene	C_8H_{10}	106.158	0.2803¶	3.567¶	3.6618¶	5,230	4,980
	Miscellaneous gases							
24	Acetylene	C_2H_2	26.036	0.06971	14.334	0.9107	1,449	1,448
25	Naphthalene	$C_{10}H_8$	128.162	0.3384¶	2.955¶	4.4208¶	5,854**	5,654**
26	Methyl alcohol	CH_3OH	32.041	0.0846¶	11.820¶	1.1052¶	867.9	768.0
27	Ethyl alcohol	C_2H_5OH	46.067	0.1216¶	8.221¶	1.5890¶	1,600.3	1,450.5
28	Ammonia	NH_3	17.031	0.0456¶	21.914¶	0.5961¶	441.1	365.1
29	Sulfur	S	32.06	....	....	....	....	....
30	Hydrogen sulfide	H_2S	34.076	0.09109¶	10.979¶	1.1898¶	647	596
31	Sulfur dioxide	SO_2	64.06	0.1733	5.770	2.264		
32	Water vapor	H_2O	18.016	0.04758¶	21.017¶	0.6215¶		
33	Air	....	28.9	0.07655	13.063	1.0000		

From "Fuel-Flue Gases" published by the American Gas Association, New York, and reproduced by
All gas volumes corrected to 60°F. and 30 in. Hg dry. For gases saturated with water at 60°F., 1.73% of

* Calculated from atomic weights given in *J. Am. Chem. Soc.*, February, 1937.

† Densities calculated from values given in grams per liter at 0°C. and 760 mm. in the "International
was not available, the assumed value was taken as 0.0037 per °C. Compare this with 0.003662 which is the
Some of the materials cannot exist as gases at 60°F. and 30 in. Hg-pressure, in which case the values are
these materials are present, their partial pressure is low enough to keep them as gases.

‡ Converted to mean B.t.u. per pound (1/180 of the heat per pound of water from 32° to 212°F.) from

§ Deduction from gross to net heating value determined by deducting 18,919 B.t.u. per lb.-mol of water
Osborne, Stimson, and Fiock, *Nat. Bur. Standards* (U. S.), *Research Paper* 209.

¶ From 3d ed. of Combustion, published by American Gas Association.

‖ See data for Carbon, *Nat. Bur. Standards, Research Paper* 1141.

** The asterisk denotes that either the density or the coefficient of expansion has been assumed.

bustion Constants

Heat of combustion‡		Cu. ft. per cu. ft. of combustible						Lb. per lb. of combustible						Experimental error in heat of combustion
B.t.u. per lb.		Required for combustion			Flue products			Required for combustion			Flue products			
Gross	Net§	O_2	N_2	Air	CO_2	H_2O	N_2	O_2	N_2	Air	CO_2	H_2O	N_2	% + or −
14,093‖	14,093‖	...	...	...	...	...	...	2.664	8.863	11.527	3.664	.....	8.863	0.012
61,100	51,623	0.5	1.882	2.382	....	1.0	1.882	7.937	26.407	34.344	.....	8.937	26.407	0.015
4,347	4,347	0.5	1.882	2.382	1.0	...	1.882	0.571	1.900	2.471	1.571	.....	1.900	0.045
23,879	21,520	2.0	7.528	9.528	1.0	2.0	7.528	3.990	13.275	17.265	2.744	2.246	13.275	0.033
22,320	20,432	3.5	13.175	16.675	2.0	3.0	13.175	3.725	12.394	16.119	2.927	1.798	12.394	0.030
21,661	19,944	5.0	18.821	23.821	3.0	4.0	18.821	3.629	12.074	15.703	2.994	1.634	12.074	0.023
21,308	19,680	6.5	24.467	30.967	4.0	5.0	24.467	3.579	11.908	15.487	3.029	1.550	11.908	0.022
21,257	19,629	6.5	24.467	30.967	4.0	5.0	24.467	3.579	11.908	15.487	3.029	1.550	11.908	0.019
21,091	19,517	8.0	30.114	38.114	5.0	6.0	30.114	3.548	11.805	15.353	3.050	1.498	11.805	0.025
21,025	19,478	8.0	30.114	38.114	5.0	6.0	30.114	3.548	11.805	15.353	3.050	1.498	11.805	0.071
20,970	19,396	8.0	30.114	38.114	5.0	6.0	30.114	3.548	11.805	15.353	3.050	1.498	11.805	0.11
20,940	19,403	9.5	35.760	45.260	6.0	7.0	35.760	3.528	11.738	15.266	3.064	1.464	11.738	0.05
21,644	20,295	3.0	11.293	14.293	2.0	2.0	11.293	3.422	11.385	14.807	3.138	1.285	11.385	0.021
21,041	19,691	4.5	16.939	21.439	3.0	3.0	16.939	3.422	11.385	14.807	3.138	1.285	11.385	0.031
20,840	19,496	6.0	22.585	28.585	4.0	4.0	22.585	3.422	11.385	14.807	3.138	1.285	11.385	0.031
20,730	19,382	6.0	22.585	28.585	4.0	4.0	22.585	3.422	11.385	14.807	3.138	1.285	11.385	0.031
20,712	19,363	7.5	28.232	35.732	5.0	5.0	28.232	3.422	11.385	14.807	3.138	1.285	11.385	0.037
18,210	17,480	7.5	28.232	35.732	6.0	3.0	28.232	3.073	10.224	13.297	3.381	0.692	10.224	0.12
18,440	17,620	9.0	33.878	42.878	7.0	4.0	33.878	3.126	10.401	13.527	3.344	0.782	10.401	0.21
18,650	17,760	10.5	39.524	50.024	8.0	5.0	39.524	3.165	10.530	13.695	3.317	0.849	10.530	0.36
21,500	20,776	2.5	9.411	11.911	2.0	1.0	9.411	3.073	10.224	13.297	3.381	0.692	10.224	0.16
17,298**	16,708**	12.0	45.170	57.160	10.0	4.0	45.170	2.996	9.968	12.964	3.434	0.562	9.968	**
10,259	9,078	1.5	5.646	7.146	1.0	2.0	5.646	1.498	4.784	6.482	1.374	1.125	4.984	0.027
13,161	11,929	3.0	11.293	14.293	2.0	3.0	11.293	2.084	6.934	9.018	1.911	1.170	6.934	0.030
9,668	8,001	0.75	2.823	3.573	....	1.5	3.323	1.409	4.688	6.097	.....	1.587	5.511	0.088
3,983	3,983	...	...	...	...	...	...	0.998	3.287	4.285	SO_2 1.998	.....	3.287	0.071
7,100	6,545	1.5	5.646	7.146	SO_2 1.0	1.0	5.646	1.409	4.688	6.097	SO_2 1.880	0.529	4.688	0.30

---

permission.
the B.t.u. value must be deducted.

Critical Tables," allowing for the known deviations from the gas laws. Where the coefficient of expansion
coefficient for a perfect gas. Where no densities were available the volume of the mol was taken as 22.4115 l.
theoretical ones given for case of calculation of gas problems. Under the actual concentrations in which

data by Frederick O. Rossini, National Bureau of Standards, letter of Apr. 10, 1937, except as noted.
in the products of combustion. Osborne, Stimson, and Ginnings, *Mech. Eng.*, p. 163, March, 1935, and

The *coefficients of cubic expansion of tar and tar products* have been examined in the Barrett Co. laboratories and are reported by Weiss in the *Am. Gas Assoc. Monthly*, **3**, 501 (1921). The following summary is taken from that paper:

	°C.	°F.
Water-gas tar	0.000655	0.000365
Vertical-retort tar	.000640	.000355
Coke-oven tar	.000575	.000320
Horizontal gas-work tar	.000550	.000300
Pitches	.000460	.000255
Creosote oils	.000730	.000410
Carbolic oils	.000800	.000444
Gas drip (holder oils)	.00100	.000555
Phenol (liquid)	.000850	.000470
Ortho-cresol (liquid)	.000875	.000485
Meta and para cresols and xylenols	.000815	.000450
Crude tar acids	.000785	.000435
Pure benzol	.001280	.000710
Pure toluol	.001140	.000690
Pure xylol	.001060	.000570
Refined solvent	.001025	.000515
Heavy naphtha (crude)	.000880	.000490
Crude naphthalene (solid)	.000440	.000245
Refined naphthalene (solid)	.000310	.000170
Tar paints	.000630	.000370

Except for the naphthalene, all the above values apply exclusively to the liquid phase. The naphthalene values apply exclusively to the solid. The figures cannot be applied to a mixture of liquid and solid as unusual expansions or contractions may occur due to the change of state.

Gas Sampling and Storage for Analysis. While it is beyond the scope of this treatise to deal with methods of gas analysis, engineering operations frequently require the taking of gas samples and the storage of such samples for transportation and while awaiting analysis. Because of the ease with which gases diffuse, the taking of samples is not generally deemed difficult and frequently is given insufficient attention. Care must be exercised to ensure that the sample lines and sample containers are adequately purged. If the gas is under pressure, it is desirable to provide a tee which will permit the wasting of some excess past the container. Where the cross section from which the sample is to be taken is large as in chambers or some mains, a suitably designed perforated sample tube should be extended into the main or chamber. The holes should be drilled at regular intervals not exceeding 6 in., and the diameter of the tube should be at least 12 times the diameter of the holes in the side. To ensure substantially equal pressure upon all perforations, the offtake from the perforated tube should be taken from its mid-portion by a smaller inner tube. If the perforated tube is 1-in. pipe, this inner tube can conveniently be ¼-in. pipe. The sampling tube must be non-reactive and non-catalytic; and, if the gas concentrations are shifting, provision should be made for freezing immediately the equilibriums involved. Thus in sampling gases from high-temperature reactions, it is frequently desirable to sample at a high velocity with adequate water cooling.

Table 30. Heating Values for Gases and Vapors

	Mol. wt.	Deviation from perfect gas at 0°C. 760 mm. = 1+	Weight at 0°C. and 760 mm. Hg				$\frac{\rho_g (20°C.\ 760\ mm.)}{\rho_g (0°C.\ 760\ mm.)}$	$\frac{\rho_g (15.56°C.\ 760\ mm).}{\rho_g (0°C.\ 760\ mm.)}$	Heat of combustion, at 20°C.		60°F. (15.56°C.) cal._{15} per g.
			G. per l.	Lb. per cu. ft.	Uncertainty, %	Density if perfect gas, g. per l.			cal._{15} per g.	B.t.u. 60° per lb.	
	I	II	III	IV		V	VI	VII	VIII	IX	X
Gas:											
Acetylene	26.02	1.0106	1.173	0.07323	0.3	1.1607	1.07496	1.05829	11,950	21,510	11,950
Benzol	78.05	1.07*	3.73*	.2329	2.0	3.482	1.08*	1.06*	10,100	18,170	10,100
n-Butane	58.08	1.044	2.704	.1688	0.5	2.591	1.077*	1.060*	11,790	21,230	11,790
Isobutane		1.032	2.673	.1669	0.3		1.076*	1.059*	11,790	21,230	11,790
Butylene	56.06	1.033*	2.58*	.1611	2.0	2.501	1.08*	1.06*	11,580	20,840	11,580
Carbon monoxide	28.000	1.00072	1.2501	.07804	0.02	1.2492	1.07344	1.05710	2,415	4,347	2,415
Ethane	30.05	1.0119	1.3566	.08469	0.01	1.3407	1.074*	1.0575*	12,310	22,160	12,310
Ethanol	46.047	1.07*	2.20*	.1373	4.0	2.054	1.08*	1.06*	7,316	13,170	7,319
Ethylene	28.03	1.0078	1.2604	.07868	0.02	1.2506	1.075	1.058	11,920	21,450	11,920
Heptane	100.13	1.08*	4.82	.3009	4.0	4.467	1.08*	1.06*	11,540	20,760	11,540
Hexane	86.11	1.065*	4.09*	.2553	3.0	3.842	1.08*	1.06*	11,590	20,860	11,590
Hydrogen	2.0156	0.9993	0.08988	.005611	0.02	0.08992	1.0732	10.569	33,910	61,039	33,927
Hydrogene sulfide	34.08	1.0122	1.539	.09608	0.2	1.5205	1.076	1.059	3,930	7,080	3,930
Methane	16.031	1.0021	0.7168	.04475	0.02	0.7153	1.0738	1.0573	13,281	23,905	13,287
Methanol	32.031	1.063*	1.52*	.0949	3.0	1.429	1.08*	1.06*	5,703	10,265	5,706
Pentane	72.09	1.050*	3.38*	.2110	2.0	3.216	1.08*	1.06*	11,680	21,020	11,680
Pentylene	70.08	1.051*	3.29*	.2054	2.0	3.127	1.08*	1.06*	11,500	20,700	11,500
Propane	44.06	1.0275	2.020	.1261	0.1	1.9659	1.075*	1.058*	11,980	21,570	11,980
Propylene	42.05	1.020*	1.91*	.1192	0.5	1.876	1.075*	1.058*	11,680	21,020	11,680
Toluol	92.06	1.087*	4.46*	.2784	5.0	4.107	1.09*	1.07*	10,230	18,420	10,230
Solid:											
Carbon	12.00	....	....	....	....	....	....	....	7,853	14,140	7,853
Sulfur	32.06	....	....	....	....	....	....	....	2,212	3,982	2,212

* Estimated.

Table 30. Heating Values for Gases and Vapors—(*Concluded*)

	Uncertainty, %	Heat of combustion at 60°F. in B.t.u. 60 per cu. ft. at 60°F. and 29.92 in Hg dry		Heat of combustion at 60°F. *Commercial Standard* B.t.u. 60 per cu. ft. at 60°F. and 30 in. Hg saturated		Uncertainty, B.t.u. per cu. ft.	Heat of combustion at 60°F. B.t.u. 60 per cu. ft. at 60°F. and 30 in. Hg. as perfect gas saturated		Uncertainty, B.t.u. per cu. ft.	% H_2 by weight	% heat of vaporization of water formed to total heat of combustion
		Gross XI	Net XII	Gross XIII	Net XIV		Gross XV	Net XVI			XVII
Gas:											
Acetylene	0.4	1488	1437	1466	1416	7	1453	1403	5	7.77	3.415
Benzol	.2	(3990)	(3827)	(3930)	(3770)	90	3684	3534	7	7.77	4.08
n-Butane	.4	(3380)	(3118)	(3330)	(3072)	20	3200	2952	13	17.36	7.74
Isobutane	.4	(3344)	(3085)	(3294)	(3039)	17	3200	2952	13	17.36	7.74
Butylene	.4	(3170)	(2963)	(3123)	(2919)	70	3033	2835	12	14.37	6.52
Carbon monoxide	.05	320.9	320.9	316.1	316.1	0.16	315.95	315.95	0.16	0	0
Ethane	.4	1774	1622	1748	1598	7	1729	1580	7	20.13	8.59
Ethanol	.04	(1700)	(1540)	(1675)	(1517)	70	1575	1426	6	13.13	9.43
Ethylene	.4	1596	1495	1572	1472	6	1562	1463	6	14.37	6.33
Heptane	.2	(5900)	(5470)	(5800)	(5377)	240	5400	5006	11	16.01	7.29
Hexane	.3	(5020)	(4647)	(4950)	(4582)	160	4664	4317	14	16.38	7.43
Hydrogen	.02	324.21	273.96	319.41	269.90	0.10	319.57	270.04	0.06	100.0	15.50
Hydrogen sulfide	.7	(642)	(591)	(632)	(582)	4	626	577	5	5.91	7.90
Methane	.04	1012.2	911.5	997.2	898.0	0.5	995.4	896.4	0.40	25.15	9.952
Methanol	.04	(919)	(813)	(905)	(800)	30	854	755	0.3	12.59	11.60
Pentane	.4	(4180)	(3864)	(4120)	(3809)	95	3935	3638	16	16.78	7.55
Pentylene	.4	(4010)	(3747)	(3950)	(3691)	90	3766	3519	15	14.37	6.566
Propane	.4	2570	2364	2532	2329	10	2467	2269	10	18.30	8.062
Propylene	.4	2370	2217	2335	2184	17	2296	2148	9	14.37	6.466
Toluol	.3	(4785)	(4570)	(4716)	(450)	250	4401	4203	13	8.75	4.497
Solids:											
Carbon	.10										
Sulfur	.07										

Compiled by U.S. Bureau of Standards and U.S. Bureau of Mines—April, 1933.

Column XI $= \frac{\text{II} \times \text{X}}{\text{VII}} \times \frac{\text{lb./cu. ft.}}{\text{g./l.}} \times 1.8$

Column XII $= \text{XI} - \frac{\text{XI} \times \text{XVII}}{100}$

Column XIV obtained similarly from column XIII

Columns XIII and XIV obtained by multiplying columns XI and XII, respectively, by

$$\frac{30.000}{29.291} \times \frac{29.478}{30.000} \text{ or } 0.98519.$$

Columns XV and XVI are calculated from column V using the perfect gas law for the expansion.
Columns XIII and XIV use the standards adopted by the American Gas Assoc.

Table 31. Properties of Various Grades of Commercial Paraffin Hydrocarbons*

	Propanes	Butanes	Pentanes	Hexanes
Vapor pressure, lb. per sq. in. (gage)	G-3	G-4	G-5	G-6
70°F	120	33	9.5 lb. abs.	3.0 lb. abs.
90°F	165	53	14.3 lb. abs.	4.4 lb. abs.
100°F	195	65	2.6	5.4 lb. abs.
105°F	210	71	4.0	6.0 lb. abs.
130°F	300	110	14.0	9.7 lb. abs.
Temp. at which pressure is 0 lb. per sq. in. gage, °F	−44	15	95	150
Specific gravity of liquid (water = 1)	0.509	0.576	0.630	0.663
A.P.I. gravity of liquid, °A.P.I. 60/60°F	146.4	114.2	93.2	82.3
Initial boiling point, °F	−45	12	82	145
Final boiling point, °F	−40	30	100	165
Weight per gal. of liquid, lb	4.24	4.8	5.25	5.5
Mean coefficient of thermal expansion				
From 0° to 50°F	0.001316	0.000908	0.000766	0.000764
From 50° to 100°F	0.00174	0.00118	0.000940	0.000792
Specific gravity of gas (air = 1)	1.523	1.95	2.49	2.97
Specific heat of vapor at 60°F. (C_p)	0.475	0.458	0.448	0.442
Dewpoint at 14.7 lb., absolute, °F	−44	26	90	140
Cu. ft. of gas per lb. of liquid	8.49	6.7	5.17	4.21
Cu. ft. of gas per gal. of liquid	36	32	27	24
Limits of inflammability				
Gas % in gas-air mixture for lower explosive limit	2.25	2.0	1.5	1.1
Gas % in gas-air mixture for maximum rate of flame propagation	4.7	3.6	1.28	2.0
Gas % in gas-air mixture for upper explosive limit	7.4	6.6	5.5	4.2
Maximum rate of flame propagation, cm. per sec. in 2.5-cm.-diameter tube	82.4	82.5	83.0	83.5
Heating value				
B.t.u. per cu. ft	2,550	3,200	4,090	4,830
B.t.u. per lb	21,650	21,420	21,130	21,010
B.t.u. per gal	91,800	102,400	110,500	116,000
Latent heat of vaporization at boiling point				
B.t.u. per lb	186	170	155	143
B.t.u. per gal	788	830	811	788

These properties are approximate only and can be readily varied to meet exact specifications by slight changes in the composition of the various grades. Intermediate grades can be readily prepared by blending the basic grades indicated above.

* From *Proc. Am. Gas Assoc.*, 1928, Table II, p. 1296, by permission of the American Gas Association.

In general, for accurate work, samples should not be taken or stored over water as this frequently manifests undesirable differential solubility effects. Where mercury does not react with the constituents, it makes an excellent confining liquid. A sample container satisfactory for many purposes is made from a magnesium citrate bottle to which some mercury has been added. After the sample has been taken and the bottle is clamped shut, it is inverted to bring the mercury over the stopper, thus sealing this. Such sample bottles can be packed in a carrier designed to prevent tilting and can be shipped long distances.

An excellent discussion of gas sampling and handling will be found in Chap. II, Part 1, Sec. 2 of the "Gas Chemists' Handbook" (1929 ed.), published by the American Gas Association.

Errors in Gas Analyses Due to Deviations from the Simple Gas Laws. The assumption that all gases obey the simple gas laws may lead to a source of errors relating to the determination of gas composition. Such errors may arise in calculating the composition of high-thermal-content gases from combustion determinations if the partial pressure of the carbon dioxide in the combustion products is high (*cf. U. S. Bur. Mines, Tech. Paper* 54). The following data are taken from this paper.

Corrected equations indicate the magnitude of such errors when the partial pressures of the combustible gas and of the carbon dioxide produced by the combustion are 95 to 100 per cent of the total pressure. If the partial pressures differ from these, different molecular volumes, depending upon the partial pressures, are used. Tables showing molecular volumes of carbon dioxide and of ethane are subjoined. As the partial pressures diminish, the molecular volumes are seen to approach unity, *i.e.*, the volume of a perfect gas.

Table 32. Molecular Volumes of Carbon Dioxide at 20°C. and Various Partial Pressures

Pressure, mm.	Molecular volume	Percentage of total pressure
100	0.999	13.1
200	0.999	26.3
300	0.998	39.5
400	0.997	52.6
500	0.997	65.8
600	0.996	78.9
700	0.995	91.9
760	0.995	100.0

Table 33. Molecular Volumes of Ethane at 0°C. and Various Partial Pressures

Pressure, mm.	Molecular volume
100	0.999
200	0.997
300	0.996
400	0.995
500	0.994
600	0.992
700	0.991
760	0.990

Theoretical and Corrected Equations for Combustion of Certain Gases

Methane: CH_4; theoretical equation, $CH_4 + 2O_2 = CO_2 + 2\ H_2O$.
CH_4; corrected equation, $0.999\ CH_4 + 2.000\ O_2 = 0.994\ CO_2 + 2\ H_2O$.
0.999 volumes + 2.000 volumes − 0.994 volumes = 2.005 volumes = contraction
0.498 contraction = CH_4
1.005 CO_2 = CH_4

Ethane: C_2H_6; theoretical equation, $C_2H_6 + 3.5\ O_2 = 2\ CO_2 + 3\ H_2O$.
C_2H_6; corrected equation, $0.990\ C_2H_6 + 3.5\ O_2 = 1.988\ CO_2 + 3\ H_2O$.
0.990 volumes + 3.5 volumes − 1.988 volumes = 2.502 volumes = contraction
0.396 contraction = C_2H_6
0.497 CO_2 = C_2H_6

Carbon monoxide: CO; theoretical equation, $2\ CO + O_2 = 2\ CO_2$.
CO; corrected equation, $2.000\ CO + 1.000\ O_2 = 1.988\ CO_2$.
2.000 volumes + 1.000 volumes − 1.988 volumes = 1.012 volumes = contraction
1.976 contraction = CO
1.006 CO_2 = CO

Ethane: C_2H_4; theoretical equation, $C_2H_4 + 3\ O_2 = 2\ CO_2 + 2\ H_2O$.
C_2H_4; corrected equation, $0.983\ C_2H_4 + 3.000\ O_2 = 1.988\ CO_2$.
0.983 volumes + 3.000 volumes − 1.988 volumes = 1.995 volumes = contraction
0.493 contraction = C_2H_4
0.494 CO_2 = C_2H_4

Acetylene: C_2H_2; theoretical equation, $C_2H_2 + 2.5\ O_2 = 2CO_2 + H_2O$.
C_2H_2; corrected equation, $0.992\ C_2H_2 + 2.5000\ O_2 = 1.988\ CO_2 + H_2O$.
0.992 volumes + 2.500 volumes − 1.988 volumes = 1.504 volumes = contraction
0.659 contraction = C_2H_2
0.499 CO_2 = C_2H_2

Compressibility of Natural Gas at High Pressures. Deviations from the simple gas laws may be important in many engineering operations involving fuel gases. This deviation has been studied by a number of workers (Burrell and Robertson, *U. S. Bur. Mines, Tech. Papers* 131, 158. Johnson and Berwald, *U. S. Bur. Mines, Tech. Paper*, 539. American Gas Assoc., Gas Measurement Committee, Report 1, Natural Gas Dept.). As natural gas is more compressible under usual high pressures at ordinary temperatures than is called for by the simple gas laws, gas purchased at an elevated pressure gives a greater volume when the pressure is reduced than it would if the gas were ideal. Burrell and Robertson (*op. cit., Tech. Paper* 158) give the following formula for calculating the compressibility of natural gas:

$$D = aP_1 + bP_2 + cP_3 + dP_4 +, \text{etc.}$$

where D expresses the percentage deviation from Boyle's law, P_1; P_2; P_3; P_4; etc., represent the partial pressures of the respective constituents expressed in atmospheres under the conditions in question, and a, b, c, d, e, and f are characteristic factors for constituents of natural gas having the following values:

Methane	a	0.228
Ethane	b	0.90
Propane	c	1.9
Carbon dioxide	d	0.67
Nitrogen	e	0.01
Air	f	0.05

The work of Johnson and Berwald (*loc. cit.*) extends the study and gives deviation curves for characteristic natural gases. Contrary to the assumption implicit in the work of Burrell and Robertson, Johnson and Berwald found that the deviation curves were not always straight lines. For details, their work should be consulted.

This deviation may be of importance in measurements of natural gas with respect to (1) variations in pressure base, (2) gas measurements by orifice meter, (3) gas measurements by positive meter, (4) gas measurements by "non-critical flow prover," (5) gas measurements by critical flow prover, (6) in the estimation of gas reserves.

These are discussed by Johnson and Berwald.

For a discussion of equations of state and critical constants, see Physical and Chemical Calculations, pp. 618–623.

Gross and Net Heating Values. In the application of gases, there sometimes occurs some confusion on the subject of the gross and the net heating values, and it therefore appears desirable to clarify these. The **gross heating value** is the maximum utilizable heat in the products of combustion and is obtained only under such conditions that the steam is actually condensed to water. This is possible only if the products are cooled to the starting temperature, and even then there may be loss as the entering air may have been partially dry, and the flue gases must leave saturated.

In the usual applications, the excess air and the failure to cool the products of combustion generally render none of this latent heat available, and the reduced value obtained is designated the **net heating value.** When either value is used in calculations it should be properly designated at all times.

Proper definitions of the terms **gross** and **net heating values** and the use of 970 or 1040 B.t.u. as the latent heat of condensation of water [*cf.* Lichty and Brown, *Ind. Eng. Chem.*, **23**, 1419 (1931); and Porter, *Ind. Eng. Chem.*, **23**, 1433 (1931)]. The A.S.T.M. prescribes 1040.

The **net heating** values of gases at 60°F. and 29.92 in. of mercury may be calculated from the following relation:

$$H_n = H_g - \frac{859Wp}{T}$$

where H_n = net heating value of gas, B.t.u. per cu. ft.
H_g = gross heating value, B.t.u. per cu. ft.
W = cu. ft. of water vapor formed in the combustion of 1 cu. ft. of dry gas.
p = pressure of dry gas, in. of mercury.
T = temperature of gas in °R. (°F. + 460). For 60°F. and 29.92 in. of mercury the relation becomes

$$H_n = H_g - 49.4W$$

The density of moist gases is treated under Humidification (p. 1080) and Drying (p. 1479).

Data on Gas Manufacture

The Manufacture of Producer Gas. Producer gas is made by blowing humidified air into a deep ignited bed of solid fuel, usually coal or coke. The primary reaction is the combustion of the fuel, first giving CO_2 and N_2. As the gases progress, the CO_2 first formed is reduced to CO, and the water vapor is partially decomposed yielding CO, CO_2, and H_2. The thermochemistry of the various producer-gas reactions may be expressed as follows:

$$C + O_2 = CO_2 + 174{,}600 \text{ B.t.u.} \quad (1)$$

$$CO_2 + C = 2CO - 70{,}200 \text{ B.t.u.} \quad (2)$$

$$C + H_2O = CO + H_2 - 70{,}900 \text{ B.t.u.} \quad (3)$$
$$C + 2H_2O = CO_2 + 2H_2 - 71{,}600 \text{ B.t.u.} \quad (4)$$
$$CO + H_2O = CO_2 + H_2 - 700 \text{ B.t.u.} \quad (5)$$

There is some divergence of opinion concerning the exact mechanism of the reactions and their relative velocities, which cannot be considered here in detail. There is, however, evidence to show that the thin oxidation zone, immediately above the ash zone, gives predominatingly reaction (1). As the partial pressure of the free oxygen reaches a low value or is practically gone, carbon monoxide appears and rapidly increases. Shortly above this point, steam decomposition begins and free hydrogen appears rapidly at first and then increases only very slowly. This zone is sometimes designated the *primary reduction zone* and is relatively thin. In the higher or *secondary reduction zone*, no reduction of steam by carbon occurs. This secondary reduction zone serves chiefly as a heat interchanger, the hot gas serving to heat the incoming fuel. Above the secondary reduction zone is the *distillation* zone which is relatively unimportant with low-volatile fuels such as coke and anthracite, but which may contribute a very considerable quantity of thermal energy in the form of gaseous hydrocarbons from high-volatile fuels. Indeed, it may represent as much as 40 per cent of the heating value of the gas.

Above the fuel bed, the heating value of the gas may drop somewhat due to the Neumann reversion [*Stahl u. Eisen*, **33**, 394 (1913) and *T. V. deut. Ing.*, **58**, 1481, 1501 (1914)],

$$2CO \rightarrow CO_2 + C$$

and to the leakage of CO_2 and steam around the edges of the fuel bed and through blow holes.

The capacity of a producer is a variable quantity, depending chiefly upon the quality of the fuel supplied, the method of operation, the design of the producer, and the character of the demand for gas with respect to quality and quantity. With hand-firing conditions, gasification rates for short periods as high as 15 lb. fuel per square foot grate area per hour have been attained with gas coals having a low percentage of high-fusing ash. About 8 to 9 lb. per sq. ft. per hr. is obtained with good-quality bituminous coal, and with lower grade fuels the gasification rate may not exceed 6 to 7 lb. The limitations are imposed by the clinker conditions and the necessity of avoiding both the blowing over the fuel and the extension of the oxidizing and primary reducing zones to such heights that the relatively slow reduction of carbon dioxide to the monoxide does not have time to proceed sufficiently, and the gas discharged is consequently too high in CO_2 and too low in heating value. While this last disadvantage may be overcome to some extent by the use of a deeper fuel bed, and the excessive oxidizing temperatures may be reduced somewhat by the addition of more steam to the air blast, the blowing over of fuel presents a practical limitation on velocity conditions for any given grade of fuel. To avoid this with high gasification rates it is necessary to use mechanical operations which ensures uniform fuel-bed condition free from zones of excessive gas velocity. The fuel should also be carefully sized. Observing suitable precautions, American gas producers may average gasification rates of 15 lb.; and 25 lb. are possible with good coals. It is claimed that with mechanical poking and continuous ash removal, gasification rates as high as 50 lb. have been attained by some producers.

Increasing the depth of the fuel bed raises the temperature of the primary reduction zone and affords a longer time of contact. This gives better decomposition of the steam. According to Haslam [*Ind. Eng. Chem.*, **16**, 782 (1924)], the constituents of the producer gas come to an apparent equilibrium constant dependent upon the thickness of the fuel bed alone, and independent of gas velocity (rate of firing), ratio of pounds of steam to pounds of coal, or temperature of the exit gas. In this

$$K' = \frac{(CO_2)(H_2)}{(CO)(H_2O)} = 0.096L$$

in which L is the depth of the fuel bed in feet.

Excessive amounts of steam should be avoided, as these, in accordance with equilibrium considerations, raise the CO_2 concentration and produce steam in the exit gases, an inert diluent which removes heat from the fuel bed. On the other hand, the use of some steam is desirable, as its endothermic decomposition prevents excessive clinker formation, and serves to convert some of the sensible heat developed from the carbon-oxygen combustion into potential energy in the form of CO and H_2. The most desirable value is usually about 0.4 lb. water in the air blast per pound of coal gasified.

The importance of a satisfactory size and uniform space conditions in the producer bed has already been indicated. The coal should be spread uniformly, the fuel bed should be constantly poked, and the ash continuously removed to avoid channeling, and attendant unequal temperature and contact conditions.

Blauvelt [*Trans. Am. Inst. Mining Eng.*, **18**, 614 (1890)], discussing soft-coal producer-gas practice, emphasizes the importance of placing the producer so as to lose as little as possible of the sensible heat of the gas and to prevent the condensation of the hydrocarbon vapors. He recommends a high fuel bed, keeping the producer cool on top, thereby preventing the breaking down of the hydrocarbons and the deposit of soot and also reducing the CO_2 concentration. He advises the use of as much steam with air as will maintain incandescence.

For many purposes a clean producer gas must be used. As it is difficult to clean a hot gas, it is necessary to cool and scrub. The sensible-heat losses can, however, be greatly diminished by the use of water-jacketed side walls and waste-heat boilers, if the capacity and demand will warrant the necessary investment.

In addition to precautions indicated in the foregoing, it is important to avoid mixing the ash with hot coal in the poking; a good distribution of the entering air steam should be secured, and the steam control should be such that a constant air-steam ratio is maintained under constant load.

Practically any solid fuel can be used in producers, provided the density-surface conditions are such that the fuel will form a satisfactory combustion bed under reasonable velocities. Economic considerations, therefore, play a primary part in the choice of the fuel. For the higher B.t.u. producer gases it is necessary, of course, to use a fuel high in volatile combustible, and it should preferably be closely sized, non-coking, low in ash, with the ash non-clinkering, and low in sulfur. The fuel should also be low in moisture.

It is impossible to deal extensively with the action of different types of fuels, and different types of producers. The reader is referred to texts such as Haslam and Russell on "Fuels and Their Combustion," McGraw-Hill, New York, 1926, and Rambush, "Modern Gas Producers," Van Nostrand, 1923. Considerable valuable data are contained in publications of the *U. S. Bur. Mines.* such as *Bulls.* 7 and 13. The use of the net hydrogen-volatile-matter ratio of coals in certain gas-producer calculations is discussed in *Ind.*

Eng. Chem., **20**, 1371 (1928). Such calculations are, however, limited by difficulties in securing true average gas samples.

As an illustration of results which may be obtained in large coke-fired mechanical producers of the Kerpley type, the extensive data of Morris, *Am. Gas Assoc., Proc.*, 1922, pp. 39–45, may be consulted. His heat balance shows that 79 per cent of the input appeared in the calorific value of the gas and about 7½ per cent as useful high-pressure steam. The chief heat losses occurred in the scrubbers, cooling water, and radiation, totaling over 13 per cent.

The Manufacture of Water Gas. For public service distribution the water gas is generally carbureted with oil gas and is known as *carbureted gas*, the use of blue water gas being confined to certain industrial conditions and as a diluent for peak-load demands on natural gas, oil gas, or coal-gas distributing systems.

Since the manufacture of carbureted water gas involves necessarily the manufacture of blue gas, consideration of blue gas logically comes first. The blue-gas machine consists primarily of a generator which is a steel shell, lined with insulation and fire brick, and may vary in free internal diameter from about 3 to about 10 ft. This generator is equipped with an iron charging door at the top, ground to prevent leakage, and suitable clean-out doors at the side near the bottom. Connections at the top are made to lead away the hot gases, and, to admit steam for down run, air-blast, steam, and gas-outlet connections are provided at the bottom of the generator. The fuel is ordinarily charged to a depth of from 7 to 9 ft.

Solid fuels, properly sized, are employed. Anthracite coal and, later, coke have been favorite fuels and are still widely used; particularly, coke. Since the World War led to a shortage of these fuels, the use of certain bituminous coals was necessary and this has led to an extensive development with attending economies. Not all bituminous coals are satisfactory. Certain chemical and physical properties must exist which are, however, as yet not well defined. A non-coking coal which will not shatter on heating has obvious advantages.

The fire is lighted and blown with an air blast until the fuel bed attains satisfactory incandescence. The chemical reactions of the heated bed under blasting correspond to those of an air-blown producer. However, no water vapor, beyond that ordinarily present in the air, is admitted, and the velocity-contact conditions established seek to store the maximum amount of heat in the generator at the highest possible rate, while the combustion products are maintained high in carbon dioxide. Carbon monoxide in the blast products may represent an energy which cannot be economically used. The blast products may be used to heat checkerwork for oil cracking and for raising steam in a waste-heat boiler, or they are discharged to the atmosphere through a stack valve.

When the bed has attained a satisfactory temperature, and before the carbon monoxide production becomes excessive, the air supply is shut off and steam is admitted to produce the desired blue gas. The steam reacts with the carbonaceous fuel to produce carbon monoxide and hydrogen with some carbon dioxide, but the velocity, contact, and temperature conditions are so chosen that only low percentages of the dioxide are obtained. Under these conditions some small amounts of nitrogen, methane, and organic sulfur compounds are also present.

According to the reactions given for the gas producer, the endothermic decomposition of the steam rapidly reduces the fuel-bed temperature, which reduction favors the production of carbon dioxide rather than the desired monoxide; therefore it is necessary to discontinue the steam admission and subject the bed to air blasting until a satisfactory temperature is again attained. The gas making and blasting thus go on in cyclic fashion, with pauses for fuel charging and clinkering. The gas-making time lost for these two later operations has, however, been very greatly reduced, in fact almost eliminated, by the recent development of automatic grates and chargers.

It will be seen that blue-gas manufacture involves the balancing of the temperatures of the fuel bed, which must not be allowed to get too hot, thus giving excessive carbon monoxide losses in the blast; nor too low, involving carbon dioxide losses in the gas-making period.

If the steam is admitted solely from the bottom of the fuel bed, this rapidly grows too cold and may not again ignite when blasted. Moreover, the top of the bed grows excessively hot, involving higher sensible-heat losses. To overcome these difficulties, the steam run is divided, part up and part down through the bed. To prevent explosions, the steam flow just preceding and immediately after the air blast is upward, thus avoiding the mixing of air and combustible gas. To secure fires which are readily cleaned, it is customary to use a considerable excess of steam. Often 50 per cent of the steam used may not be decomposed.

The efficiency of blue water gas manufacture is necessarily low. Some of the heat losses include:

1. Losses as sensible heat in the gases and combustion products which leave the generator at high temperature.
2. Losses as sensible heat in clinker and ashes, and unconsumed fuel which may be removed from the generator.
3. Radiation and convection losses from the generator.
4. Losses due to the vaporization of water in the fuel.
5. Stand-by losses when the machine is under heat but idle.
6. Combustion losses during fire-cleaning periods.
7. Losses due to the use of excessive amounts of steam during the steam run.

The preparation of a heat and material balance upon a blue water gas machine is a difficult matter, and no standard practice has been developed. Some of the shortcomings of some published tests have been discussed by Travers [*Trans. Inst. Chem. Eng.*, **2**, 65 (1924)]. Recently the Water Gas Committee of the American Gas Association has been engaged in the development of a test code for public utility use with carbureted gas (*Proc. Am. Gas Assoc.*, p. 1507, 1930).

Some conception of heat and material balances which may be secured in blue water gas manufacture may be obtained from the data of Morris (*Am. Gas Assoc., Proc.*, 1922, pp. 39–45). Coke consumption amounted to about 36.2 lb.per 1000 cu. ft. of blue water gas, calorific value 287 B.t.u. per cu. ft. (gross); correspondingly, 2,230 cu. ft. of air and 51.9 lb. of steam were required. The calorific value of the gas made represented 62.7 per cent of the thermal value of the coke. The set was equipped with a waste-heat boiler and produced more steam than was used under the grate. The highest heat loss occurred in the form of unburned carbon in the ash, amounting to 12.9 per cent.

Carbureted Water Gas. Carbureted water gas consists largely of a mixture of a rich oil gas and blue water gas. The production of the latter has already been discussed, and from the principles mentioned it was shown that the gases leaving the generator at the end of the blast must of necessity carry a very considerable quantity of potential heat as carbon monoxide, in addition to the sensible heat due to the high temperatures at the top of the fuel bed. This heat is utilized in part by the maintenance of a satisfactory "cracking" temperature in two succeeding shells of approximately the same diameter as the generator. The shells are designated the **carburetor** and **superheater** in the order in which the blue gas passes through. The first or carburetor is about as high as the generator, but the superheater is much higher in order to maintain a satisfactory natural draft thus permitting generator charging from above.

Ordinarily the carburetor and superheater are filled with checker brick in order to provide extensive heat-transfer surfaces thus assisting in the cracking of the oil.

In the normal operating cycle, the generator and checker chambers are first brought to the requisite temperatures by air-producer operation with combustion of the producer gas by admitting secondary air between the generator and carburetor. The machine is then placed on the run, *i.e.*, during part of the cycle, steam is admitted and blue gas formed. This passes over to the top of the carburetor and downward, here meeting a fine spray of oil which is carried through the checker chambers and cracked to an oil gas and some tar. As the fire cools and the steam decomposition falls off, the generator is again blasted with air and the carburetor ignited with the secondary air blast until satisfactory temperatures are again attained, when the steam run is again begun.

With automatic opening and closing of the valves, these cycles can be arranged to follow one another after intervals of only a few minutes, and this is now generally done.

When extremely heavy petroleum oils are used, the quantity of checker surface in the carburetor is greatly reduced in order to avoid stopping the interstices with heavy carbon deposits. In addition, some oil may be thrown directly on the top of the fuel bed during the down-stream run. (*Cf.* Dashiell, *Am. Gas Assoc., Proc.*, 1930, p. 886. Also see Gas Production Committee Report at 1932 Convention of the Am. Gas Assoc., "Use of Heavy Oil for Carburetion" by Hartzel and Lueders.)

The carbureted water gas apparatus as well as the Pacific coast oil gas apparatus has been used for **reforming natural gas and oil refinery gas** (for references *cf.* Description, p. 2362). Carbureted water gas apparatus has also been converted for use in the manufacture of oil gas by replacing the generator fire with a refractory screen. Oil alone, suitably treated with air and steam, is used as the fuel in this process (Johnson, *Am. Gas Assoc., Proc.*, 1932, p. 892).

Following the introduction of bituminous coal as a fuel in water gas manufacture, important modifications were introduced into the fundamental water gas cycle. One of these is the back run in which the down-run steam is admitted at the top of the superheater, whence it passes backward to the lower part of the carburetor, thence to the top of the fuel bed. The blue gas produced is led directly to the wash box by a by-pass or back-run pipe. This arrangement does away with the hot valve between the carburetor and lower portion of the generator and uses instead a valve in the wash box. The process cools the top of the superheater and preheats the down-run steam and avoids unduly high temperatures in the checkers, thereby minimizing an objection often encountered in soft-coal operation due to the combustion of volatile matter during the blow. The gas leaving the bottom of the generator has a relatively low temperature because of heat interchange with the ash and clinker cooled by the blast. By passing the checkers and going directly to the wash box, it decreases waste-heat losses from the checkers. Important thermal economies are claimed for it. The carbureting oil is added during the up-run. The process is discussed in the *Annual Reports* of the Water Gas Committee of the American Gas Association for 1924 and 1925. A very similar process, save that the steam is admitted at the top of the generator, is the **Chrisman down-run process.** This is described in the 1925 *Report.*

To bring the fuel bed to temperature in bituminous-coal operation while avoiding the overheating of the checkers, the blow-run operation is sometimes used. In this, during the latter part of the blow, the carburetor air is cut off and the stack valve through which the combustion products are wasted is closed, thus forcing the air-producer gas from the generator into the holder.

The **reversed air-blast process** is related to the blow run and the back run. The reverse blast is admitted to the top of the superheater at the portion of the cycle generally used for the blow run and follows the path of the back run. It is described by Howard [*Am. Gas Assoc. Monthly*, 7, 579–584 (1925)].

American practice in water gas making has recently tended toward automatic grate operations which greatly reduce the labor charges and stand-by losses for cleaning and clinkering. The first of these to be successfully used was the A.B.C. grate of the Western Gas Construction Company described by Ramsburg at the First International Coal Conference (*Proc.*, 1926, pp. 514*ff.*). Automatic grates are now offered by the Semet-Solvay Company, the United Gas Improvement Company, and the Gas Machinery Company, and the operations of some of these have been discussed in *Reports* of the Water Gas Committee of the American Gas Association (*Am. Gas Assoc., Proc.*, 1929, p. 1245; 1930, p. 1573; 1931, p. 1172).

The Cracking of Oil in Gas Making. The cracking of oil to hydrocarbons of low molecular weight is of very considerable importance to the gas manufacturer who relies upon oil gas or carbureted water gas. The literature relating thereto is very extensive as cracking is an old art. The manufacture of gas from oil was discussed in England as early as 1792. The considerations involved are many and complex and have been dealt with at some length in recent books among which may be mentioned those of Leslie, of Gruse, and of Hurd. Experimental data directly applicable to problems of the carbureted water gas maker will be found in a paper by Downing and Pohlman (*Am. Gas Inst., Proc.*, 1916, p. 587). A résumé of some of the important reviews and findings was given by Huff (*Am. Gas J.*, October, 1929, p. 47).

Temperature and time of contact are important variables in the cracking and are, in certain measure, interdependent. Actually, the operator of the water gas machine is usually obliged by load considerations to maintain adequate capacity, and consequently the most important variable at his command is the temperature of the cracking chambers. In *U. S. Bur. Mines, Bull.* 203, are summarized various prior investigations on temperature as follows:

1. The candles per gallon reach a maximum between 1300° and 1350°F.
2. The percentage of illuminants reaches a maximum between 1300 and 1400°F.
3. The percentages by volume of methane and hydrogen increase with increasing temperature.
4. The volume of gas increases with the temperature.
5. The percentage of carbon formed from the cracking of the oil increases with the temperature.
6. The percentage of tar formed decreases with increasing temperature.

7. The B.t.u. per gal. oil increases with the temperatures and quite often the maximum is not reached under 1500°F.

The above conclusions apply, in general, to temperatures up to 1600°F., which is about the upper limit of the experiments made, so far as data are available.

The control of the process of cracking involves not only thermodynamics but also chemical kinetics, and equilibrium is not reached in practice. Nevertheless, a graphical summary of thermodynamic relationships in cracking, as developed by Francis [*Ind. Eng. Chem.*, **20**, 281 (1928)] is of considerable interest. This author has studied the data for the free energies of the formation of methane, ethane, octane, ethylene, acetylene, benzene, toluene, naphthalene, and cyclohexane, and has plotted the equations with the ordinates reduced to the free energy of formation per carbon atom as shown in Fig. 20. This shows stability relations between the various hydrocarbons at various temperatures. In using the chart it should be remembered that only those compounds are stable whose free energies of formation are negative. Thus it will be noted that only the paraffins and the higher olefines have any thermodynamic stability, and that above 260°C. methane is the only stable hydrocarbon. Francis points out that this seems to be inconsistent with the ordinary cracking reactions in which large quantities of olefines and some aromatics are produced from the paraffin hydrocarbons. Actually, however, this contradiction is not present because thermodynamics does not show the mechanism by which the reaction proceeds, and the formation of intermediate products is more a matter of kinetics than thermodynamics. Thus, while at equilibrium conditions, all of these compounds whose free energies of formation are positive give carbon and hydrogen; under the commercial conditions of cracking, true equilibrium is not obtained and intermediate products are instead obtained. The quantity and composition of these are determined by relative-reaction velocities. Francis points out that no paraffin hydrocarbons should be capable of direct condensation to a higher hydrocarbon; but if lower ones are formed simultaneously with the higher ones, there is no change in free energy of gaseous reactions.

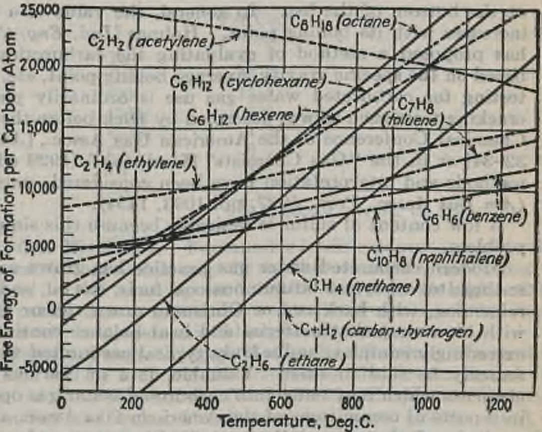


FIG. 20.—Stability relations of various hydrocarbons. [*From Francis, Ind. Eng. Chem.*, **20**, 281 (1928).]

The chart may be used for a number of interesting observations: thus it will be seen that if it is proposed to make benzene from methane the temperature of 850°C. must be exceeded if a substantial yield is desired.

The theory and art of oil cracking has advanced markedly in recent years. In the theory the more notable advances have occurred in developments involving the conception of free radicals and chain reaction kinetics. As yet, however, such theories have not reached a stage where they contribute directly to oil gas or carbureted water gas practice.

The chemical composition of the oil used for carbureting has, of course, some considerable importance but commercial oils are in general so complex chemically that the evaluation from its chemical composition is not convenient or certain. In general, a saturated paraffin may be said to rank highest in carbureting efficiency, with olefines somewhat less efficient. Naphthenes are better than aromatics, and asphaltics have been given the lowest value. Others place the asphaltics above the aromatics and place unsaturates at the bottom of the list. In general, the value of a hydrocarbon fraction increases with its boiling point. Holmes [*Ind. Eng. Chem.*, **24**, 325 (1932)] has proposed a method of evaluating the carbureting power of a gas oil, based on the specific gravity, average boiling point, etc. Chief reliance in oil testing for carbureted water gas use is ordinarily placed upon small-scale cracking tests such as was described by Dick before the 1933 Production and Chemical Conference of the American Gas Assoc., [*Am. Gas J.*, **138** (No. 6), 32–34] or in the "Gas Chemists' Handbook" (1929 ed., p. 66). Analytical methods and interpretation have been considered, conspicuously by Mighill (*Am. Gas Assoc., Proc.*, 1927, pp. 1093, 1454).

A low content of sulfur is desirable because this simplifies the purification problem.

Modern carbureted water gas practice has grown so involved (embracing anthracite, coke, and bituminous coal fuels, gas oil, and heavy oil operations, reforming, with back run or Chrisman down run or reversed air blast, and with blow run) that material and heat-balance considerations have become exceedingly complex, and a truly typical carbureted water gas operation can scarcely be said to exist. Valuable data on various combinations of the elements which may enter into carbureted water gas operation may be found in reports of committees of the American Gas Assoc. and in the annual proceedings of that association. An informative and complete analysis of a carbureted water gas operation in Great Britain is presented in the 7th Report of the Research Sub-committee of the Institute of Gas Engineers of Great Britain [*Gas J.*, **158**, 800–827 (1922)].

Where gas oil is employed for carbureting, oil is not thrown on the fire, and reforming is not practiced, it is possible to follow the oil efficiencies obtained by a procedure known as the Providence modification of the Pacific coast method.

This is described by the Subcommittee on Uniform Oil Efficiency of the Water Gas Committee of the American Gas Assoc. in the 1926 and 1927 Proceedings of that association.

Oil Gas. Pacific Coast Methods. On the Pacific coast, except in the state of Washington, manufactured gas for city distribution has generally been made from California residuum oils. The processes used are somewhat akin to the water gas process. There is a heating period in which oil is burned to bring the generator to temperature, and the products of combustion are wasted. This is followed by a making period in which heated steam, carbon, and oil interact to yield stable gas together with some tar and lampblack. The make is followed by the blow in which carbon deposits are burned off by air. The length of the cycle varies but is ordinarily much longer than the water gas cycle.

One of the major units is the large two-shell **Jones oil-gas generator.** Heat and material balances on such a generator have been reported by Pike and West [*Ind. Eng. Chem.*, **21,** 104 (1929)]. Because of the widespread distribution of natural gas, oil gas has become of relatively less importance on the West coast, and the introduction of a development of the Jones process, in which the lampblack was gasified, was halted in consequence (*Am. Gas Assoc., Proc.*, 1930, p. 1349).

In the Pike and West tests the regular two-shell Jones oil gas generator gave about 53 per cent of the input heat in the form of purified gas. In addition, lampblack accounted for about 18.5 per cent and tar about 3.3 per cent of the thermal input.

The oil gas made had the following analysis:

	Per Cent by Volume
H_2	45.9
CH_4	26.2
CO	13.2
C_2H_4	3.2
C_6H_6	1.2
CO_2	6.5
O_2	0.5
N_2	3.3
	100% (B.t.u. per cu. ft. 579.)

Single-shell Oil Gas Methods. For smaller loads recourse is had to the single-shell oil gas machines, which may be further classified, according to methods of operation, in the **heat up and make down** or **straight shot** and the **heat and make down** types. One drawback possessed by these machines is the relatively large amount of lampblack produced.

References on Pacific Coast Oil-gas Methods. Further information on the above methods will be found in the treatise on the Production of Oil Gas by the Educational Committee of the Pacific Service Employees' Association under the direction of Cowles, Henderson, and Yard (1922); Yard in the *Am. Gas Assoc. Monthly*, **7,** 741–743 (1925), and in the section on Oil Gas Machines in the 1925 Report of the Water Gas Committee, the American Gas Association (*Am. Gas Assoc., Proc.*, 1925, p. 1278).

The Dayton Process. While the foregoing oil gas methods have been very successful on the Pacific coast, owing no doubt in large measure to the ready availability of suitable oil supplies, they have not been widely used elsewhere. For small plants in other parts of the country the Dayton process has been proposed. This utilizes small cast-iron retorts which are first brought to temperature by combustion of oil in the refractory setting about the retort. The supply of combustion oil is then greatly decreased, and a mixture of oil and air is admitted to the retort through an atomizer. Partial combustion and cracking follow. Continuous production of gas having any desired heating value between 300 and 560 B.t.u. per cu. ft. is said to be possible without the use of much combustion oil in the setting. For gases above 560 B.t.u., a considerable amount of heating oil is required as the air supplied for these higher B.t.u. values is insufficient to maintain the retort at the requisite temperature.

The oil consumption is said to be about 4 gal. per 1000 cu. ft. of 450 B.t.u. gas. In addition about 0.3 gal. tar is recovered. The heat economy is high, 82.7 per cent of the input appearing in the potential heat of the gas and 7 per cent in the tar.

Following is an analysis of the gas made (450 B.t.u.):

	Per Cent by Volume
CO_2	6.1
Unsaturated hydrocarbons	14.7
O_2	0.9
CO	5.6
Saturated hydrocarbons	7.8
H_2	1.7
N_2	63.2
Specific gravity	1.02
Theoretical flame temperature	3700°F.

This gas possesses so high a gravity that it cannot conveniently be used for stand-by or peak equipment in systems distributing mixed coal and water gas.

The original Dayton process has been improved to increase the capacity and eliminate the provisions for external heating, but with means for heat

Table 34. Comparison of Principal Operating Results, Coal-gas Tests*

	Utica Koppers gas ovens	Rochester U.G.I. verticals	Lowell horizontals	Stamford Glover-West continuous vertical through retorts
Coal:				
Proximate analysis (dry basis):				
Volatile	35.2	35.8	35.1	34.3
Fixed carbon	57.7	57.6	58.4	58.4
Ash	7.1	6.6	6.5	7.3
Sulfur	0.9	1.0	0.9	0.9
Moisture as charged (wet basis)	2.1	2.5	1.67	2.2
Coal gas:				
Cu. ft. at 30 in., 60° satur., per ton dry coal	11,315	11,347	12,110	14,300
Cu. ft. at 30 in., 60° satur., per lb. dry coal	5.66	5.67	6.06	7.15
Heating value, B.t.u. per cu. ft	573	540	542	530
B.t.u. in gas, per lb. dry coal	3,245	3,065	3,280	3,790
Average analysis, %:				
CO_2	2.2	1.7	2.40	3.0
Illuminants	4.0	3.1	3.05	2.8
O_2	0.8	0.5	0.75	0.2
H_2	46.5	49.7	47.95	54.5
CO	6.3	6.9	7.35	10.9
CH_4	32.1	29.9	27.15	24.2
N_2	8.1	8.2	11.35	4.4
Dry coke produced as % dry coal carbonized	70.2	70.7	68.6	70.0
Gal. dry tar per ton dry coal	12.4	11.4	11.6	14.6
Ammonia per ton dry coal, lb	6.4	5.9	5.0	3.4
Producer fuel, lb. per ton coal	292	318	229	275
Producer fuel, % coal carbonized	14.6	15.9	11.5	13.7
Waste-heat steam from and at 212°F. per lb. producer fuel, lb	0.59	1.96	1.14	5.74
Steaming based on dry coal, %	....	1.0	....	12.0

* *Am. Gas Assoc., Proc.*, 1928, p. 1118.

exchange. Such generators may have capacities of 1,000,000 cu. ft. of gas a day. Heating values greater than 500 B.t.u. cannot be made from oil, and the usual output may have values from 300 to 500 B.t.u. Such gas can be mixed with natural gas to replace a high specific gravity, high B.t.u. manufactured gas and may be used to some advantage to meet peak demands on natural gas systems.

A modification of the Dayton process, known as the **Faber** process uses as raw material natural gas, refinery gas, or liquefied hydrocarbon gases obtained from natural gas. This is essentially a gas re-forming process.

Further information will be found in an article by Binnall [*Ind. Eng. Chem.*, **13**, 242 (1921) and in *Am. Gas Assoc.*, *Proc.*, 1930, pp. 1361, 1482].

Coal Gas. Coal gas is produced by the pyrogenetic decomposition of suitable coals in externally heated refractory retorts. As the potential heat of the gas made usually represents only about 20 per cent of the total heat input, the economic success of this means of gas production is largely determined by the form value of the other products, particularly the coke, which usually represents some 50 or 60 per cent of the heat input. Considerable attention is therefore given to the selection of proper coals in order to secure cokes of salable quality. While in certain localities very important domestic markets have been developed, the great markets lie in the metallurgical industries. Consequently, recent coal-gas production has tended to follow by-product metallurgical practice.

In general, three types of carbonizing processes may be said to be in operation in the American gas industry: (1) horizontal and (2) vertical retorts; and (3) coke ovens. Comparative heat and material balances have been made upon certain units representing such processes by a committee of the American Gas Assoc. (*Proc.*, 1928, p. 1115). The operating results shown in Table 34 have been taken from this report.

Combustion Data

Ignition Temperatures. The term "ignition temperature" is sometimes used as a characteristic property of a substance. Any such implication is, however, in error. A system ignites when the rate of gain of heat due to the oxidation reaction is greater than the rate of loss of heat. It follows therefore that ignition temperatures are dependent upon the properties of the particular system in question and are not characteristic of the igniting substance alone. The term as ordinarily applied is used to signify the temperature at which rapid combustion occurs in ordinary air.

In ignition temperature measurements two methods are used to avoid error due to slow combustion prior to ignition: (1) by exploding the gaseous mixture by adiabatic compression and (2) by preheating independently each stream of gas and air before mixing.

Limits of inflammability are dependent in part upon the conditions of determination. When a weak source of ignition is used, certain mixtures near the limits may not inflame. The limit is lower for upward propagation of flame, and the limits are wider as the tube diameter is increased, although usually not markedly wider above diameters of 5 cm. except in certain cases, such as acetylene whose upper limit is much increased by enlarging the tube diameter above 5 cm. In closed tubes the length of the tube may also affect the results.

Normal variations of atmospheric pressure do not appreciably affect the limits of inflammability. The effect of larger variations in pressure is neither simple nor uniform but is specific for each inflammable mixture. Reduction

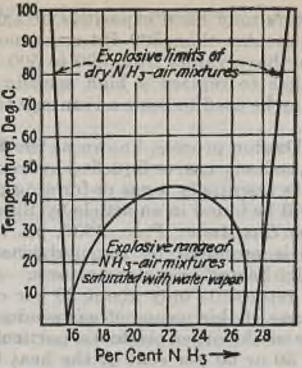


FIG. 21.—Limits of inflammability of ammonia-air mixtures.

Table 35. Ignition Temperatures (at Atmospheric Pressure) of Gases, Liquids, and Solids*

Substance	Ignition temperature in air		Ignition temperature oxygen	
	°F.	°C.	°F.	°C.
Hydrogen (H_2)	1076–1094	580–590	1076–1094	580–590
Carbon monoxide (CO)	1191–1216	644–658	1179–1216	637–658
Methane (CH_4)	1202–1382	650–750	1033–1292	556–700
Ethane (C_2H_6)	968–1166	520–630	968–1166	520–630
Propane (C_3H_8)	....	....	914–1058	490–570
Ethylene (C_2H_4)	1008–1018	542–547	932–966	500–519
Acetylene (C_2H_2)	763–824	406–440	781–824	416–440
Hexane† (C_6H_{14})	909	487	514	268
Decane ($C_{10}H_{22}$)	865	463	396	202
Benzol (C_6H_6)	1364	740	1224	662
Toluol ($C_6H_5CH_3$)	1490	810	1026	552
Phenol (C_6H_5OH)	1319	715	1065	574
Aniline ($C_6H_5NH_2$)	1418	770	986	530
Methyl alcohol (CH_3OH)	....	....	1031	555
Ethyl alcohol (C_2H_5OH)	1036	558	797	425
Propyl alcohol (C_3H_7OH)	941	505	833	445
Isopropyl alcohol (C_3H_7OH)	1094	590	954	512
n-Butyl alcohol (C_4H_9OH)	842	450	725	385
Amyl alcohol ($C_5H_{11}OH$)	768	409	734	390
Ethyl ether ($(C_2H_5)_2O$)	649	343	352	178
Glycerine ($C_3H_5(OH)_3$)	932	500	777	414
Acetone ($(CH_3)_2CO$)	1292	700	1054	568
Sugar ($C_{12}H_{22}O_{11}$)	725	385	712	378
Cylinder oil	783	417	608	320
Pennsylvania crude	601	367	468	242
Gas oil	637	336	518	270
Kerosene	563	295	518	270
Acetaldehyde (CH_3CHO)	365	185	284	140
Benzaldehyde (C_6H_5CHO)	356	180	334	168

* From Haslam and Russell "Fuels and Their Combustion," McGraw-Hill, New York, 1926.

† Ignition temperatures for hexane and for other substances in the remainder of this table are for dry air and dry oxygen.

Table 36. Approximate Limits of Inflammability of Single Gases and Vapors in Air at Ordinary Temperature[1] and Pressures[2]

Gas or vapor	Limits in air, %				Limits in oxygen, %				Oxygen percentage below which no mixture is inflammable	
	Lower		Higher		Lower		Higher		Nitrogen as diluent of air	Carbon dioxide as diluent of air
Hydrogen	4.1	*4.0*	74	.....	...	*4.0*	94	....	5.0	5.9
Ammonia	.....	*16*	.....	*27*	...	*15*	..	*79*		
Hydrogen sulfide	.....	*4.3*	.....	*45.5*						
Carbon disulfide	1.25	.....	44	*50*						
Carbon monoxide	12.5	.....	74	.....	...	.....	..	....	5.6	5.9
Methane	5.3	(*5.0*)	14	*15*	...	*5.4*	..	*59*	12.1	14.6
Ethane	3.2	.....	12.5	.....	...	*4.1*	..	*50*	11.0	13.4
Propane	2.4	.....	9.5	.....	...	.....	..	....	11.4	14.3
Butane	1.85	.....	8.4	.....	...	.....	..	.....	12.1	14.5
Isobutane	1.8	.....	8.4							
Pentane	1.4	.....	7.8	.....	...	.....	..	....	12.1	14.4
Isopentane	.....	*1.3*								
Hexane	1.2	.....	6.9	.....	...	.....	..	....	11.9	14.5
Heptane	.....	*1.0*	.....	*6.0*						
Octane	0.95									
Nonane	.....	*0.8*								
Decane	.....	*0.7*								
Ethylene	3.0	*2.75*	29	.....	3.1	*2.9*	80	....	10.0	11.7
Propylene	.....	*2.0*	.....	*11.1*	2.1	.....	53	....	11.5	14.1
Butylene	.....	*1.7*	.....	*9.0*						
Amylene	.....	*1.6*								
Acetylene	2.5	(*2.5*)	.....	80						
Benzene	1.4	.....	6.7							
Toluene	1.4	*1.3*	.....	*6.7*						
o-Xylene	.....	*1.0*	.....	*6.0*						
Cyclopropane	.....	*2.4*	.....	*10.3*	..	*2.45*	..	*68*		
Cyclohexane	1.3	*1.2*	8.3							
Methyl cyclohexane	1.2									
Methyl alcohol	7.3	*6.7*	.....	36*	...	.....	..	....	*10.3*	*13.5*
Ethyl alcohol	4.3	*3.3*	.....	19*						
Propyl alcohol	.....	*2.5*								
Iso-propyl alcohol	.....	*2.6*								
Butyl alcohol	.....	*1.7*								
Iso-butyl alcohol	.....	*1.8*								
Amyl alcohol	.....	*1.2*								
Isoamyl alcohol	.....	*1.2*								
Allyl alcohol	.....	*2.4*								
Methyl ethyl ether	.....	*2.0*	.....	*10.1*						
Ethyl ether	1.9	*1.85*	48	.....	...	*2.1*	..	*82*		
Vinyl ether	.....	*1.7*	.....	*27*	...	*1.8*	..	*85*		
Ethylene oxide	3.0	.....	80							
Propylene oxide	2.1	.....	21.5							
Dioxan	.....	*2.0*	.....	*22*†						
Diethyl peroxide	.....	*2.35*								
Acetaldehyde	.....	*4.0*	.....	*57*						
Croton aldehyde	.....	*2.1*	.....	*15.5*†						
Furfural	2.1‡									
Paraldehyde	.....	*1.3*								
Hydrocyanic acid	.....	*5.6*	.....	*40*						

Table 36. Approximate Limits of Inflammability of Single Gases and Vapors in Air at Ordinary Temperature[1] and Pressures[2]—(*Concluded*)

Gas or vapor	Limits in air, %				Limits in oxygen, %				Oxygen percentage below which no mixture is inflammable	
	Lower		Higher		Lower		Higher		Nitrogen as diluent of air	Carbon dioxide as diluent of air
Acetone	3.0	*2.55*	11	*13*	...	.....	..	....	*13.5*	*15.6*
Methyl ethyl ketone	.....	*1.8*	.....	*10*						
Methyl propyl ketone	.....	*1.5*	.....	*8.5*†						
Methyl butyl ketone	.....	*1.2*	.....	*8*†						
Methyl formate	5.9	*5.0*	20	*23*†						
Ethyl formate	2.7	*2.7*	13.5	16.5†						
Methyl acetate	.....	*3.1*	.....	*15.5*†						
Ethyl acetate	2.0	*2.2*	8.5	*11.5*†						
Propyl acetate	2.0	*1.8*	.....	*8.0*†						
Isopropyl acetate	.....	*1.8*	.....	7.8†						
Butyl acetate	1.7§									
Ethyl nitrite	.....	*3.0*								
Methyl chloride	10.7	*8.0*	17.4	*19*	...	*8.0*	..	*66.0*		
Methyl bromide	.....	*13.5*	.....	*14.5*						
Ethyl chloride	4.2	*4.0*	14.3	*14.8*						
Ethyl bromide	.....	*6.7*	.....	*11.2*						
Ethylene dichloride	6.2	.....	15.9†							
Dichloroethylene	9.7	.....	12.8							
Propylene dichloride	.....	*3.4*	.....	*14.5*‡						
Amyl chloride	.....	*1.4*								
Vinyl chloride	4.0	.....	22.0							
Diethyl selenide	.....	*2.5*								
Tin tetramethyl	.....	*1.9*								
Lead tetramethyl	.....	*1.8*								
Pyridine	.....	*1.8*	.....	*12.4*						

[1] Except as noted.

[2] Coward and Jones, Limits of Inflammability of Gases and Vapors, *U. S. Bur. Mines, Bull.* 279, 1938 p. 125.

Values in italics were obtained in closed apparatus. Values in parentheses are for turbulent mixtures.

* At 60°C.

† At 100°C.

‡ At 125°C.

§ At 30°C.

of pressure below 760 mm. narrows the range of flammability by raising the lower limit and decreasing the higher limit. At a suitably low limit the limits coincide; below this point there is no propagation of flame. Increase of pressure above atmospheric does not always widen the limits. In some mixtures this increase narrows the limits and a mixture which can propagate flame at atmospheric pressure may not be able to do so at higher pressures. The flammability of hydrogen-air and carbon monoxide-air mixtures (downward propagation of flame) is narrowed at both limits by a moderate increase in pressure above atmospheric. Under analogous conditions the range of each gas in the paraffin series is narrowed on the lower limit side and widened at the upper limit side.

Table 37. Approximate Limits of Inflammability of Some Industrial Mixtures of Gases and Vapors in Air at Ordinary Temperatures and Pressures, Per Cent

Gas or vapor	Lower limit, by volume	Higher limit, by volume	Gas or vapor	Lower limit, by volume	Higher limit, by volume
Benzene	1.1	......	Natural gas	4.8	13.5
Gasoline	1.4	6.5	Illuminating gas	5.3	31
Water gas	6–9	55–70	Blast-furnace gas	35	74

The above was taken from *U. S. Bur. Mines, Bull.* 279, 1938 ed., p. 126. The lower limit for blast-furnace gas was obtained in a small or closed vessel. The limit figures apply only to particular samples; analytical data will be found in the bulletin cited. By the use of Le Chatelier's law the limits of similar mixtures can be calculated.

Limits of Inflammability of Various Fuel Gases (from *U. S. Bur. Mines, Bull.* 279, 1938 ed., p. 118).

The limits of "explosibility" or inflammability of various fuel gases in air were determined in a tube 4 cm. in diameter and 79.6 cm. in length, one end being permanently closed and the other sealed with a diaphragm of thick, varnished paper. Ignition was obtained by a coil of iron wire that was caused to glow by a 10- to 15-amp. current. The results were:

Table 38. Limits of Inflammability of Various Fuel Gases

Composition	Coal gas (?)	Coal gas (?)	Town gas (?)	Water gas (?)	Semi-water gas (?)	Producer gas (?)
Carbon dioxide	3.2	2.0	3.6	4.2	4.4	3.6
Unsaturated hydrocarbons	4.4	3.6	2.6			
Oxygen	1.0	0.4	0.4	0.4	0.4	0.4
Carbon monoxide	9.0	8.8	17.6	40.4	34.2	29.2
Hydrogen	47.8	52.4	45.4	43.0	34.4	10.0
Methane	27.0	27.0	18.6			
Nitrogen	7.6	5.8	11.8	12.0	26.6	56.8
Lower limit, %	8.5	9.5	10.4	13.5	16.0	31.0
Higher limit, %	25.0	29.0	30.5	68.5	70.5	74.0

Limits of Inflammability of Mixtures of Inflammable Gases and Vapors (from *U. S. Bur. Mines, Bull.* 279, 1938 ed., pp. 7*ff.*).

A simple formula, of additive character, was advanced by Le Chatelier to connect the lower limits of single gases with the lower limit of any mixture of them. It is

$$\frac{n_1}{N_1} + \frac{n_2}{N_2} = 1$$

in which N_1 and N_2 are the lower limits in air for each combustible gas separately and n_1 and n_2 are the percentages of each gas in any lower-limit mixture of the two in air.

The formula expresses the fact that, for example, a mixture of air, carbon monoxide, and hydrogen, which contains one-quarter of the amount of carbon monoxide and three-quarters of the amount of hydrogen necessary to form a lower-limit mixture, will be a lower-limit mixture. If the formula expresses experimental facts, the lower limits of inflammability form a series of inflammability equivalents for the individual gases of a mixture.

The formula also leads to the deduction that lower-limit mixtures if mixed in any proportions give rise to mixtures that are also at their lower limits; or, vice versa, the formula may be deduced from the latter statements as a postulate.

The formula may be generalized to apply to any number of combustible gases, thus:

$$\frac{n_1}{N_1} + \frac{n_2}{N_2} + \frac{n_3}{N_3} + \cdots = 1,$$

and, so far as it expresses experimental results truly, may be applied to high-limit mixtures, with the appropriate rewording of the definition of n_1 . . . *and* N_1 . . . .

A small algebraic transformation gives a more useful formula for calculating the limits of any mixture of combustible gases which obeys it, as follows:

$$L = \frac{100}{\dfrac{p_1}{N_1} + \dfrac{p_2}{N_2} + \dfrac{p_3}{N_3} + \cdots}$$

in which p_1, p_2, p_3 are the proportions of each combustible gas present in the original mixture, free from air and inert gases, so that

$$\frac{p_1}{N_1} + \frac{p_2}{N_2} + \frac{p_3}{N_3} + \cdots = 100$$

An example of the use of the formula will make its application clear: To calculate the lower limit of a "natural gas" of the composition

Methane	80 per cent (lower limit, 5.3 per cent)
Ethane	15 per cent (lower limit, 3.22 per cent)
Propane	4 per cent (lower limit, 2.37 per cent)
Butane	1 per cent (lower limit, 1.86 per cent)

gives

$$L = \frac{100}{\dfrac{80}{5.3} + \dfrac{15}{3.22} + \dfrac{4}{2.37} + \dfrac{1}{1.86}} = 4.55 \text{ per cent}$$

The accuracy of the formula has been tested carefully for many mixtures. The results are discussed separately in the appropriate sections later. In general, it may be said that, while the formula is often correct or very nearly so, there are some marked exceptions. It seems that the limits (lower and higher) of mixtures of hydrogen, carbon monoxide, and methane taken two at a time or all together and of water gas and coal gas may be calculated with approximate accuracy. The same is true for mixtures of the simpler paraffin hydrocarbons, including "natural gas." Sometimes, however, the differences between the calculated and observed values are very large; . . . Many of the greater discrepancies are found with upward-propagating flames, especially when one of the constituents is a vapor, such as ether or acetone, capable of giving rise to the phenomenon known as a "cool flame." Le Chatelier's law is useful when its applicability has been proved, but it must not be applied indiscriminately.

An extension of the law to apply to other atmospheres than air . . . is that when limit mixtures are mixed the result is a limit mixture, provided that all constituent mixtures are of the same type; that is, all are lower-limit mixtures or higher-limit mixtures. This law holds, for example, for methane in a range of oxygen-nitrogen mixtures and in air-carbon dioxide, air-argon, and air-helium mixtures, except near the point at which lower and higher limits meet, where the proportion of inert gas is large. It holds also for mixtures of hydrogen, methane, and carbon monoxide, in a wide range of mixtures of air, nitrogen, and carbon dioxide, and may therefore be used to calculate the limits of inflammability of mine-fire gas and of the atmospheres after a mine explosion, of blast-furnace gas, of automobile-exhaust gas, and of the gases from solid explosives.

A brief account of the method of calculating limits of complex industrial gases, such as those just mentioned, follows; greater detail will be found in the original account (*U. S. Bur. Mines, Tech. paper* 450).

The chief gases in these mixtures are hydrogen, carbon monoxide, methane, nitrogen, carbon dioxide, and oxygen. The procedure is as follows:

1. The composition of the mixture is first recalculated on an air-free basis; the amount of each gas is expressed as a percentage of the total air-free mixture.

2. A somewhat arbitrary dissection of the air-free mixture is made into simpler mixtures, each of which contains only one inflammable gas and part or all of the nitrogen or carbon dioxide.

3. The limits of each mixture thus dissected are read from tables or curves. (See Fig. 22.)

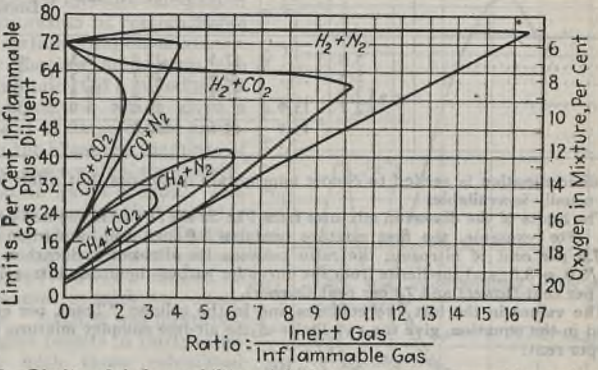


FIG. 22.—Limits of inflammability of hydrogen, carbon monoxide, and methane containing various amounts of carbon dioxide and nitrogen. (*From Coward and Jones, Limits of Inflammability of Gases and Vapors, U. S. Bur. Mines Bull.* 279, p. 9, 1938 *rev.*)

4. The limits of the air-free mixture are calculated from the figures for the dissected mixtures obtained in (3), by means of the equation:

$$L = \frac{100}{\frac{p_1}{N_1} + \frac{p_2}{N_2} + \frac{p_3}{N_3}}, \cdots$$

where p_1, p_2, p_3 . . . are the proportions of the dissected mixtures, in percentages, and N_1, N_2, *and* N_3 . . . are their respective limits.

5. From the limits of the air-free complex mixture thus obtained the limits of the original complex mixture are deduced.

The following is an example of the calculation applied to a mine-fire atmosphere. It contained:

Constituent	Per cent	Air free, per cent	Constituent	Per cent	Air free, per cent
Carbon dioxide	13.8	15.9	Methane	3.3	3.8
Oxygen	2.8	0	Hydrogen	4.9	5.7
Carbon monoxide	4.3	5.0	Nitrogen	70.9	69.6

1. The composition on an air-free basis, also given above, is found thus: The amount of air in the mixture is 2.8 × 100/20.9 = 13.4 per cent. The air-free mixture is therefore 86.6 per cent of the whole. When the original amounts of carbon dioxide, carbon monoxide, methane, and hydrogen are divided by 86.6 and multiplied by 100, the "air-

free" percentages are obtained. The nitrogen figure is the difference between 100 and the sum of these percentages.

2. The inflammable gases are paired off with the inert gases separately to give a series of dissected mixtures, as shown in the following table:

Combustible	Per cent	CO_2, per cent	N_2, per cent	Total, per cent	Ratio of inert to combustible	Limits from Fig. 22	
						Lower	Higher
CO	5.0	....	17.5	22.5	3.5	61	73.0
CH_4	3.8	....	20.9	24.7	5.5	36	41.5
H_2	$5.7\begin{cases}3.0\\2.7\end{cases}$	.... 15.9	31.2	34.2 18.6	10.4 5.9	50 32	76.0 64.0
Total	....	15.9	69.6	100.0			

Some discrimination is needed to choose appropriate quantities, but a fair latitude of choice usually is available.

3. The limits of the dissected mixtures from Fig. 22 are shown in the last two columns above. For example, the first mixture contains 5.0 per cent of carbon monoxide and 17.5 per cent of nitrogen; the ratio between its nitrogen and carbon monoxide is 17.5/5.0 = 3.5; and the limits from the curve for carbon monoxide-nitrogen mixtures are 61 per cent (lower) and 73 per cent (higher).

4. The values in the last two columns and in the column "Total, per cent," substituted in the equation, give the two limits of the air-free complex mixture, calculated to 0.5 per cent:

$$\text{Lower limit} = \frac{100}{\frac{22.5}{61} + \frac{24.7}{36} + \frac{34.2}{50} + \frac{18.6}{52}} = 43 \text{ per cent}$$

$$\text{Higher limit} = \frac{22.5}{73} + \frac{100}{\frac{24.7}{41.5} + \frac{34.2}{76} + \frac{18.6}{64}} = 61 \text{ per cent}$$

The range of inflammability of the air-free complex mixture is therefore 43 to 61 per cent.

5. As the air-free mixture is 86.6 per cent of the whole, the limits, in air, of the mine-fire atmosphere are 43 × 100 ÷ 86.6 and 61 × 100 ÷ 86.6, or 50 and 70 per cent, respectively.

The novice's difficulty with such calculations is in stage 2, where an appropriate amount of inert gas has to be chosen to pair with each combustible gas in turn. The ratio of inert to inflammable gas must not be so high that the mixture falls outside the extreme right of the corresponding curve in Fig. 22. A little practice will soon enable this difficulty to be overcome.

It need only be added that if the amount of inert gas is so great that a complete series of inflammable mixtures cannot be dissected the air-free mixture is not inflammable. Moreover, the air-free mixture may be inflammable, but when its limits are multiplied by the appropriate factor in the final stage of the calculation the result may be greater than 100 for each limit; the original mixture is then not capable of forming an explosive mixture with air because it contains too much air already. Finally, if the lower limit of the original mixture is less than 100 and the higher limit greater than 100, the mixture is inflammable *per se* and would explode if a source of ignition were present.

About 20 examples, which cover a wide range of industrial gases, have been tested by experiment. The calculated and observed limits agree within 2 or 3 per cent, excepting one higher-limit figure for a mixture that contained an unusually large amount (nearly 24 per cent) of carbon dioxide.

Speed of the Propagation of Flame. Flame speeds are of importance in certain problems of gaseous combustion. Thus in questions of interchangeability it is usually difficult to substitute a major quantity of a slow-burning gas for a fast-burning gas. The same holds for problems in burner design where flash back and blowoffs must be considered. In heat transfer, questions of combustion intensity or flame output are related to flame speeds. Mixture compositions below or above which flame speeds are zero determine the lower or upper limits of inflammability.

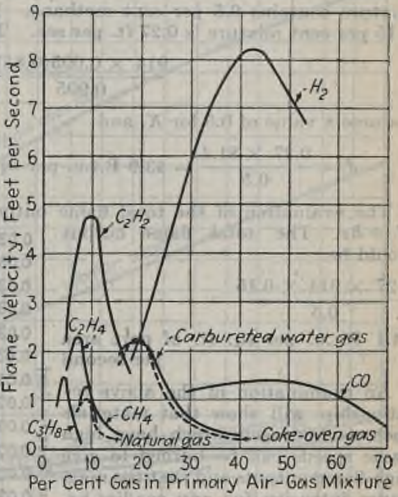


FIG. 23.—Flame velocities of various gas-air mixtures. Data on individual gases from *Bur. Standards J. Research*, **17**, 7–43 (1936). Data on natural gas, coke-oven gas, and carbureted water gas from "Combustion," Am. Gas Assoc., 1932. (*As compiled by M. A. Elliott and A. R. T. Denues of the Bureau of Mines for Marks, "Mechanical Engineers' Handbook," McGraw-Hill Book Company, Inc.*)

Payman has proposed the use of the Le Chatelier mixture calculation for the estimation of certain flame speeds in certain complex mixtures. Whenever the law holds true, the percentage amount of a mixed gas of known composition which gives a certain speed of flame can be calculated from the observed amounts of constituent gases which, separately, give the same flame speeds. The formula is the same as that used for calculating the dilution limits. There are many marked exceptions and the application cannot be deemed general enough to constitute a law. Natural gas mixtures, however, give results in fairly close agreement with those calculated (*U. S. Bur. Mines, Tech. Paper* 427).

Checking Flame Propagation. Flame propagation is impossible in a **coal gas + air** mixture when the tubes are smaller than 0.08 in. or 2 mm. in inside diameter. **Methane + air** mixtures will not propagate flame through tubes whose internal diameter is smaller than 0.142 in. or 3.6 mm. **Hydrogen,** however, has been found to flame through tubes whose internal diameter was 0.035 in. or 0.9 mm.

Specific Flame Output or Intensity of Combustion. A conception known as specific flame output, or intensity of combustion, is sometimes used in comparing gaseous fuels. This is expressed by the relation $J = (u \times H)/K$, where J = specific flame output, B.t.u. per square foot of port area per second; u = flame velocity in feet per second; H = heat developed per cubic foot of air-gas mixture as will be further explained; and K = ratio of the port area to the area of the inner cone. The result may be expressed as primary flame output when based on the heat developed in the primary combustion or as total flame output when based upon the total heat developed in both primary and secondary combustion.

The calculation of primary flame output involves the relationship

$$H = \frac{hx_t(1 - x)}{(1 - x_t)}$$

where h = the net B.t.u. value of the gas.
x = the decimal fraction of combustible gas in the primary mixture.
x_t = decimal fraction of combustible gas in the stoichiometric mixture for complete combustion.

Assume that the primary air-gas mixture contains 15 per cent methane whose net B.t.u. at 60°F. and 30 in. of Hg = 914 and that the stoichiometric mixture contains 9.5 per cent methane. From Fig. 23 the flame speed of a 15 per cent mixture is 0.27 ft. per sec. Therefore

$$\frac{914 \times 0.095(0.85)}{0.905} = 81.4$$

Assume a value of 0.5 for K, and

$$J = \frac{0.27 \times 81.4}{0.5} = 43.9 \text{ B.t.u. per sq. ft. of port area per second}$$

The evaluation of the total flame output is made similarly except that $H = hx$. The total flame output would be

$$\frac{0.27 \times 914 \times 0.15}{0.5} = 74.1 \text{ B.t.u. per sq. ft. of port area per second}$$

An examination of the above relationship will show that materials such as hydrogen which have high flame speeds can be burned to give high flame outputs and that the maximum intensity for any gas lies between the theoretical mixtures and the mixture having the maximum flame velocity [*Gas u. Wasserfach*, **14**, 1012 (1931); **79**, 17 (1936). *Z. Ver. deut. Ing.*, **80**, 1275 (1936)].

Flame Temperatures. In many industrial processes using combustion systems, the flame temperature is of considerable importance. Thus, it may largely determine the temperature gradient and hence the character and efficiency of the heat transfer. It is, however, exceedingly difficult to determine the temperature of the flame with accuracy, and it is therefore customary to compare fuels on the basis of their **theoretical flame temperatures.** This is found by suitable substitution in the following:

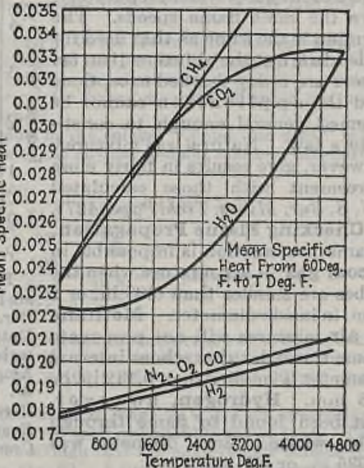


FIG. 24.—Mean specific heats from 60°F. to T°F. (B.t.u. per cubic foot per degree Fahrenheit). The cubic foot is measured at 60°F. and 29.92 in. Hg.

$$\text{Theoretical flame temperature} = \frac{\begin{matrix}\text{heat of}\\ \text{combustion}\end{matrix} + \begin{matrix}\text{sensible heat}\\ \text{in fuel}\end{matrix} + \begin{matrix}\text{sensible heat}\\ \text{in air}\end{matrix}}{\begin{pmatrix}\text{total quantity of}\\ \text{combustion products}\end{pmatrix} \times \begin{pmatrix}\text{their mean}\\ \text{specific heat}\end{pmatrix}}$$

The calculation of theoretical flame temperatures is most simply made on a trial-and-error basis, *i.e.*, a probable temperature is assumed and corresponding mean specific heat values are chosen. If the sum of the heats divided by the chosen heat capacities gives a temperature higher than that assumed, it is necessary to make a new trial by assuming a new and higher temperature

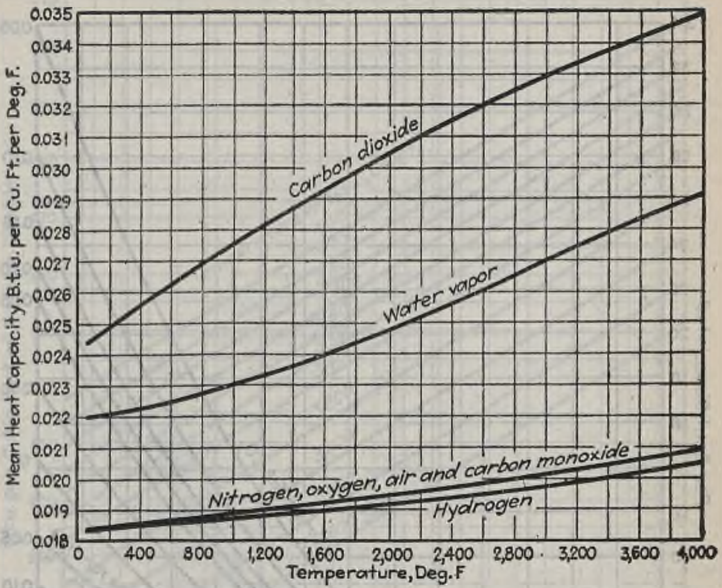


FIG. 25.—Mean heat capacities from 60°F. to given temperature. The cubic foot is measured at 60°F. and 29.92 in. Hg.

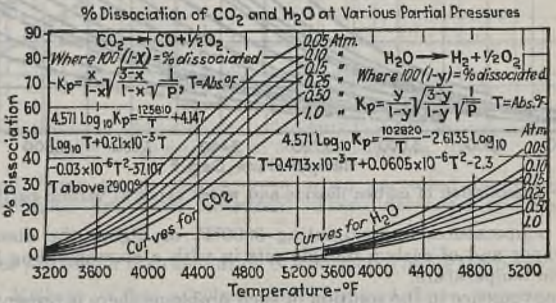


FIG. 26.—Dissociation of carbon dioxide and water at various partial pressures.

and choosing again a corresponding specific heat value, and so on, until values of flame temperature and mean specific heat are found to correspond. It is always necessary to use the low or net heating value of the gas as the latent heat of condensation of the water formed is not available to raise the temperature of the products of combustion in the flame.

The flame temperatures thus found are the uncorrected theoretical temperatures. If these temperatures are low, no further correction is necessary, but at temperatures approximating 3000°F. or higher the dissociation of water vapor and of carbon dioxide sets in with a corresponding lowering of the flame temperature.

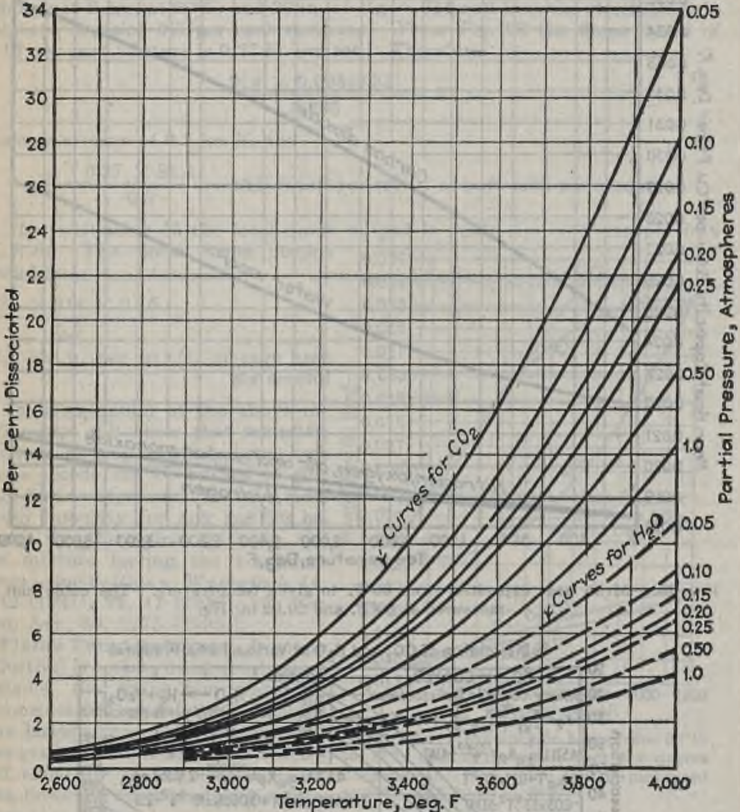


FIG. 27.—Dissociation of carbon dioxide and water vapor at various partial pressures.

For convenience in the solution of such problems there is presented a curve showing the sensible-heat content per cubic foot of the ordinary combustion products and curves showing the dissociation of carbon dioxide and water at high temperatures. These are from the booklet "Combustion," 1932 edition, and are reproduced by the courtesy of the American Gas Assoc.

A more detailed discussion of the calculation of flame temperatures will be found in an article by Jones *et al.*, *J. Am. Chem. Soc.*, **53**, 869 (1931), on the flame temperature of hydrocarbon gases.

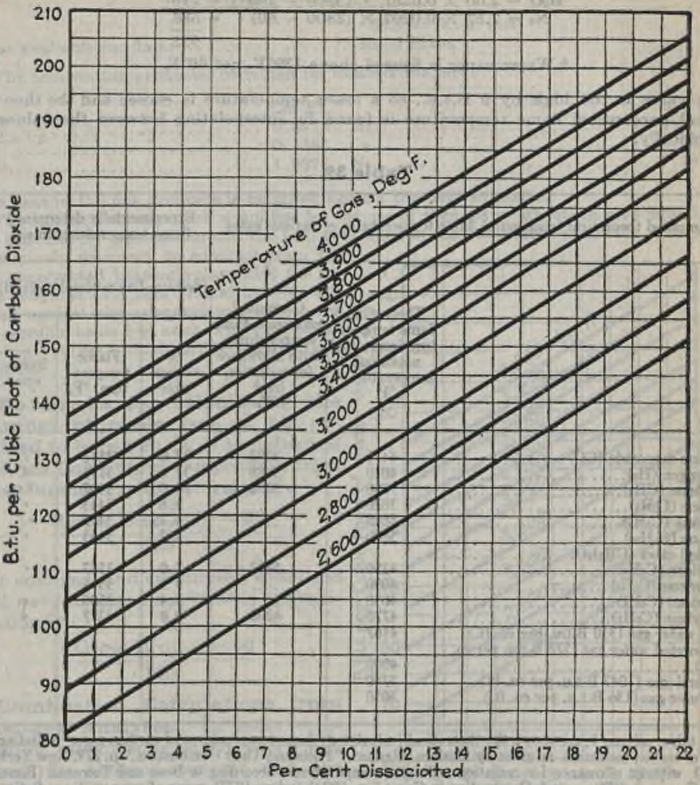


FIG. 28.—Heat content of carbon dioxide, above 60°F., corrected for dissociation. The cubic foot is measured at 60°F. and 29.92 in. Hg.

Calculated flame temperatures, even when corrected for dissociation, are usually higher than those measured. This difference may be as much as 100°F.

A problem will render the calculation of a corrected flame temperature clear.

Example. Methane (CH_4); calculate the uncorrected flame temperature. One cubic foot of methane at 60°F. and 29.92 in. Hg gives a net heating value of 916 B.t.u. (from Thomsen's data), and the following combustion products (measured at 60°F. and 29.92 in. Hg): CO_2, 1.00 cu. ft.; H_2O, 2.00 cu. ft.; N_2, 7.565 cu. ft.

From Fig. 25 giving various mean specific heat values and an assumed flame temperature, 3800°F., the following comparison is made:

Heat in

$$\begin{aligned} & & & \text{B.t.u.} \\ CO_2 &= 1.00 \times 0.0345 \times (3800 - 60) &&= 129 \\ H_2O &= 2.00 \times 0.0287 \times (3800 - 139^*) &&= 210 \\ N_2 &= 7.57 \times 0.0207 \times (3800 - 60) &&= \underline{586} \\ & & & \ 925 \end{aligned}$$

* Water vapor is figured above 139°F. not 60°F.

This value is too high by 9 B.t.u., so a lower temperature is chosen and the theoretical uncorrected flame temperature is found by interpolating between the values graphically.

Table 39.

Computed theoretical (maximum) flame temperatures of various gases			Experimentally determined flame temperatures in air		
	1	2	(Sodium-line reversal method)		
	Theoretical flame temperature (computed maximum temperature)[a] °F.	Computed maximum flame temperatures, with allowance for radiation loss[b] °F.	% combustible	Flame temperature, °F.	Reference
Carbon monoxide (CO)	4475	3092	32.0	3812	c
Hydrogen (H_2)	4010	3488	31.6	3713	d
Methane (CH_4)	3750	3038	10.0	3407	d, e
Ethane (C_2H_6)	3820	.....	5.8	3443	e
Propane (C_3H_8)	3840	.....	4.15	3497	e, f
Butane (C_4H_{10})	3870	.....	3.2	3443	e
Methyl ether $(CH_3)_2O$					
Ethylene (C_2H_4)	4250	3452	7.0	3587	e
Propylene (C_3H_6)	4090	.....	4.5	3515	e
Butylene (C_4H_8)	4030	.....	3.4	3506	e
Acetylene (C_2H_2)	4770	4208	9.0	4217	d
Blue water gas (310 B.t.u. per cu. ft.)	4167				
Carbureted water gas (578 B.t.u. per cu. ft.)	4090				
Natural gas (1047 B.t.u. per cu. ft.)	3740				
Producer gas (136 B.t.u. per cu. ft.)	3050				

In table, column 1, are shown the theoretical (calculated) flame temperatures of various gases, including that of carbon monoxide, as given by Haslam (Haslam, "Fuels and Their Combustion," p. 279, New York 1926), without allowance for radiation. Similar computations, according to Bone and Townend (Bone, and Townend, "Flame and Combustion in Gases," p. 199, London, 1927), using allowances for radiation according to Helmholtz and Callendar's experiments and mean specific heats after Holborn and Henning, are shown in column 2.

[a] Values according to Haslam.

[b] Values according to Bone and Townend.

[c] Lewis and von Elbe, "Combustion, Flames and Explosions of Gases" Macmillan, New York, 1938.

[d] Jones, Lewis, and Seaman, *J. Am. Chem. Soc.*, **53**, 3992 (1931).

[e] Jones, Lewis, Friauf, and Perrott, *Ind. Eng. Chem.*, **53**, 869 (1931).

[f] Loomis and Perrott, *Ind. Eng. Chem.*, **20**, 1004 (1928).

Methane (CH_4), corrected flame temperature: However, the value calculated above makes no allowance for the heat lost by the dissociation of the carbon dioxide and the water, which, as is shown in Fig. 27, is appreciable at these temperatures. From the statement of the flue-gas composition it will be seen that 9.5 per cent of the flue products are CO_2, and 19 per cent are water vapor. From Fig. 27 it will be seen that the dissociation of CO_2 at 3600°F. and 0.095 atm. amounts to 12.85 per cent; and of H_2O

at 3600°F. and 0.19 atm. to 3.13 per cent. The net heating value must be lowered by the respective amounts of CO and H_2 produced:

$$\begin{aligned} 1.00 \times 0.1285 \times 323 &= 41.5 \\ 2.00 \times 0.0313 \times 273 &= 17. \\ &\overline{58.5} = \text{B.t.u. lost} \end{aligned}$$

Heat available for flame.................. 857.5 B.t.u.

The combustion products, corrected for dissociation, are:

	Cu. Ft.		Cu. Ft.
CO_2..........................	0.8715	H_2..........................	0.064
CO..........................	0.1285	O_2..........................	0.096
H_2O..........................	1.9374	N_2..........................	7.565

The heat in the flue products is balanced against the heat available.

The calculations are much simplified by the use of Figs. 28 to 30, which give directly the heat lost by dissociation and the heat contained in the dissociated products. Thus, it is merely necessary to subtract from the net uncorrected heating value (916) the direct readings 41.5 plus 17 and balance the result against the appropriate readings for the sensible heats lost as shown in the sensible-heat curves, which are given in Figs. 28 to 30.

Formula for calculating the number of cubic feet of air required to burn 1 cu. ft. of an industrial fuel gas (varying in composition from blast furnace gas to natural gas):

$$\frac{(\text{Gross B.t.u.})^{1.06} - 52}{100}$$

For coal gas, and carbureted water gas and natural gas, a convenient approximation is

$$\frac{(\text{Gross B.t.u.}) - 50}{100}$$

Combustion Calculations from Fuel-gas Analyses. In most combustion calculations involving fuel gases, the thermal content and the carbon and hydrogen contents of the gas are estimated from the analysis. In all such estimations there arise two sources of possible error, namely, the determination of the chemical composition of (1) the illuminants and (2) the paraffin hydrocarbons. The illuminants are ordinarily estimated by absorption in bromine water, or in fuming sulfuric acid, and may vary widely in composition. In plants running repeated analyses, it is customary to ascertain the composition of the gas by analysis, and the thermal content by a gas calorimeter. Since the simple constituents of the gas have definite thermal values, these are subtracted from the calorimeter reading and the difference assigned to the illuminants and paraffins.

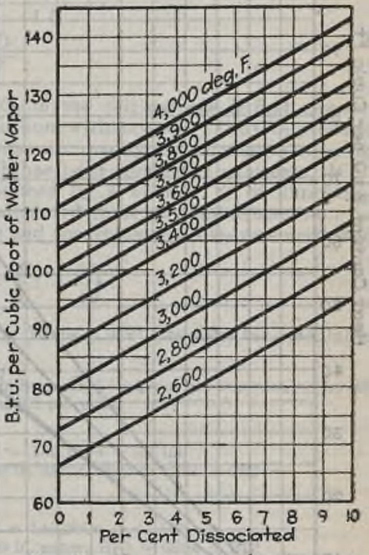


FIG. 29.—Heat content of water vapor above 60°F. corrected for dissociation. The cubic foot is measured at 60°F. and 29.92 in. Hg.

By the use of a slow combustion over copper oxide, carbon monoxide and hydrogen may be estimated independently of the paraffin hydrocarbons,

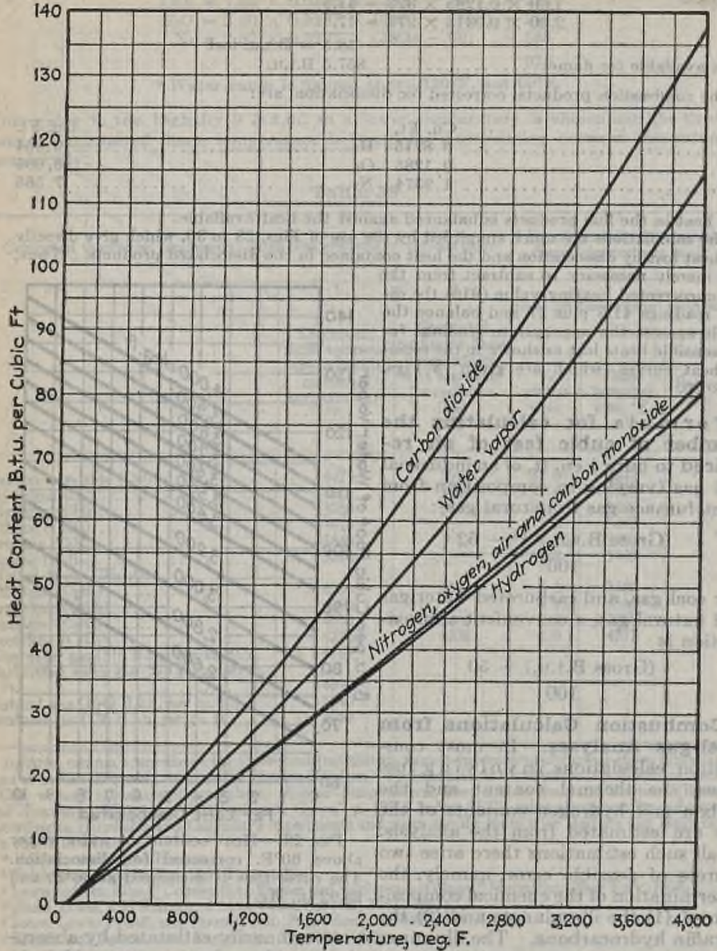


Fig. 30.—Heat content of gases from 60°F. to given temperature. The cubic foot is measured at 60°F. and 29.92 in. Hg.

which may then be burned over a platinum spiral and some knowledge of the composition of the paraffins obtained. From repeated determinations of such information it is usually possible to assign an approximate composition and heating value to the illuminants.

In the case of isolated analyses, such a procedure is not available. The matter has been considered by Watson and Ceaglske [*Ind. Eng. Chem.*, Anal. ed. **4**, 70 (1932)] who use a "triple-combustion" scheme of fuel-gas analysis with the Bureau of Mines apparatus. The two combustions above mentioned are extended to include an additional total combustion with oxygen, and from the data obtained the complete average composition and heating values of both the paraffins and the illuminants may be calculated to an accuracy of within 2 per cent, using equations developed by the above-mentioned authors. An exception occurs when the gas contains an appreciable percentage of acetylene, for which the equations do not apply.

If Δv is the contraction in volume accompanying the second combustion (of the paraffins over the platinum spiral), n is the number of carbon atoms in the average paraffin formula (C_nH_{2n+2}), and V is the volume of paraffins in the sample; then

$$V = \frac{\Delta v - 0.5\ CO_2}{1.5}$$

$$n = \frac{CO_2}{V}$$

For the third combustion, the CO_2 in the original gas is first removed and the entire residue burned over platinum with oxygen. The CO_2 produced by the combustion of the illuminants can readily be calculated from the preceding steps in the analysis which has been conducted as usual. If the illuminants are C_aH_b, the CO_2 produced by a volume V' of illuminants is $V'a$, and the value of b can be calculated from $\Delta v'$, the contraction due to the combustion of the illuminants and the volume of CO_2 produced, thus

$$\Delta v' = V'\left(1 + \frac{b}{4}\right)$$

The following equations are used by Watson and Ceaglske for calculating the heating values:

> The heating values of the lower paraffin-hydrocarbon gases are represented by the following equations with errors of less than 0.5 per cent.
> Total heating value:
>
> Gram-calories per gram-mol = 156,700n + 56,100
> B.t.u. per cu. ft. at 60°F., 30 in. satur. H_2O = 732n + 262
>
> Net heating value:
>
> Gram-calories per gram-mol = 146,200n + 45,500
> B.t.u. per cu. ft. at 60°F., 30 in. satur. H_2O = 683n + 212
>
> The heating values of the lower unsaturated hydrocarbon gases of average formula C_aH_b are satisfactorily represented by the following formulas:
> Total heating value:
>
> Gram-calories per gram-mol = 98,200a + 28,200b + 28,800
> B.t.u. per cu. ft. at 60°F., 30 in. satur. H_2O = 459a + 132b + 135
>
> Net heating value:
>
> Gram-calories per gram-mol = 98,200a + 22,900b + 28,800
> B.t.u. per cu. ft. at 60°F., 30 in. satur. H_2O = 459a + 107b + 135

The above values are not valid for acetylene.

The Flow of Gas through Pipe Lines*

Where conditions permit the application and the necessary data are available, the flow of gas in pipe lines can be most logically dealt with in accordance with Reynolds criterion as discussed in Sec. 6 of this handbook. The application has been limited because quantitative values of the viscosity relationships in commercial gas mixtures are generally not available. Another difficulty arises because gas flowing in long lines under high pressure may undergo a marked change in density.

The problem presented by the application of Reynolds criterion to flow problems such as are handled by simple gas engineering flow formulas has been considered in a paper by Huff and Logan (*Am. Gas Assoc., Proc.*, 1935, p. 687).

These authors show that at ordinary temperatures a function of the specific gravity of a gas referred to air can be substituted for the absolute viscosity of common fuel gases. The density viscosity relationship is $13.8s^{0.129} = z \times 10^3$, where s is the specific gravity of the gas referred to air as 1 and z is the absolute viscosity of the gas in centipoises. With this simplification for low-pressure flows, in which the change in density of the gas is not important and in which the flow is turbulent, it is possible to derive a flow formula resembling former formulas, but resting upon the firm foundation of Reynolds criterion and consequently applicable to a wide variety of conditions. This formula is

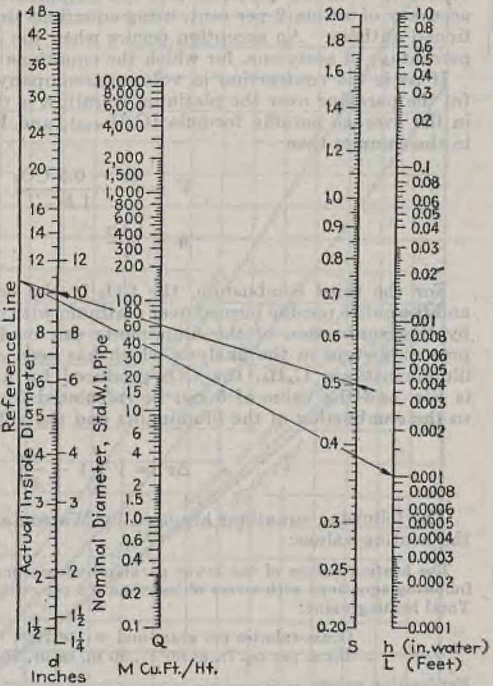


FIG. 31.—Flow of gas in commercial pipes.

$$Q = \frac{1281h^{0.545}d^{2.631}}{S^{0.468}(L')^{0.543}} \text{ cu. ft. per hr.}$$

where h = the pressure drop, in. of water.
d = the diameter of the pipe, in.

---

* See Sec. 6.

S = specific gravity of the gas at the temperature and pressure in question relative to air at room temperature and 30 in. of mercury.
L' = the length of the pipe, yd.

If it is desired to express this formula in feet of pipe L, the relationship becomes:

$$Q = \frac{2331 h^{0.543} d^{2.631}}{S^{0.468} L^{0.543}} \text{ cu. ft. per hr.}$$

A nomograph for the convenient graphical solutions of the above is given in Fig. 31.

Many other low-pressure formulas have been suggested. One such a formula is that developed by Spitzglass [*Armour Engineer*, March and May, 1917; also *cf. Am. Gas J.*, **96**, 274 (1917)]. This formula is

$$Q = 1910 \sqrt{\frac{D^5 H}{WL\left(1 + \frac{3.6}{D} + 0.03D\right)}}$$

where Q = discharge of gas, cu. ft. per hr.
D = diameter of the pipe, in.
H = frictional drop of pressure, in. of water.
W = specific gravity of gas (air = 1).
L = length of pipe, yd.

Figure 32 (from *U. S. Bur. Mines, Tech. Paper* 310, p. 23; see also *Am. Gas Light J.*, April 22, 1912) shows a diagram computed by Spitzglass to cover this formula.

Problem. To illustrate the use of this diagram, assume that the gas volume is 48,000 ft. per hr., the source of the gas is 100 yd. from the exhauster, the gas has a specific gravity of 1.1, and it is desired to keep, with the exhauster maintaining a vacuum of 20 in. water, a vacuum at the source of the gas of 5 in. water. Then, to find the diameter of pipe required, start with 48,000 in column 2, follow the dotted line horizontally to the line indicating the length of pipe (100 yd.), vertically to the line representing the specific gravity of the gas (1.1), and then horizontally across the diagram to the vertical line representing drop in pressure (15 in.); read the intersecting line, representing the diameter of the pipe, which falls in the column corresponding to that in which the volume of gas was shown. The diameter of the pipe required is 6 in.

Within recent years, the distribution of gas at moderately high pressures has become very important. In such systems, the gas pressure is measured in pounds per square inch, and a somewhat different formula applies:

$$Q = 64.3 \sqrt{\frac{D^5 PA}{WL\left(1 + \frac{3.6}{D} + 0.03D\right)}}$$

where Q = discharge of free gas, cu. ft. per hr.
D = diameter of the pipe, in.
P = friction drop in pressure, lb. per sq. in.
A = average pressure $\left(\frac{\text{initial} + \text{final}}{2}\right)$, lb. per sq. in., gage.
W = specific gravity of gas (air = 1).
L = length of pipe, miles.

Figure 33 for the convenient solution of problems involving this formula is given. Its use is similar to that illustrated for the low-pressure problems.

The factors are read in the order Q, L, A, W, D, P, turning at right angles at the termination of each projected line to project to the intersection corresponding to the next factor.

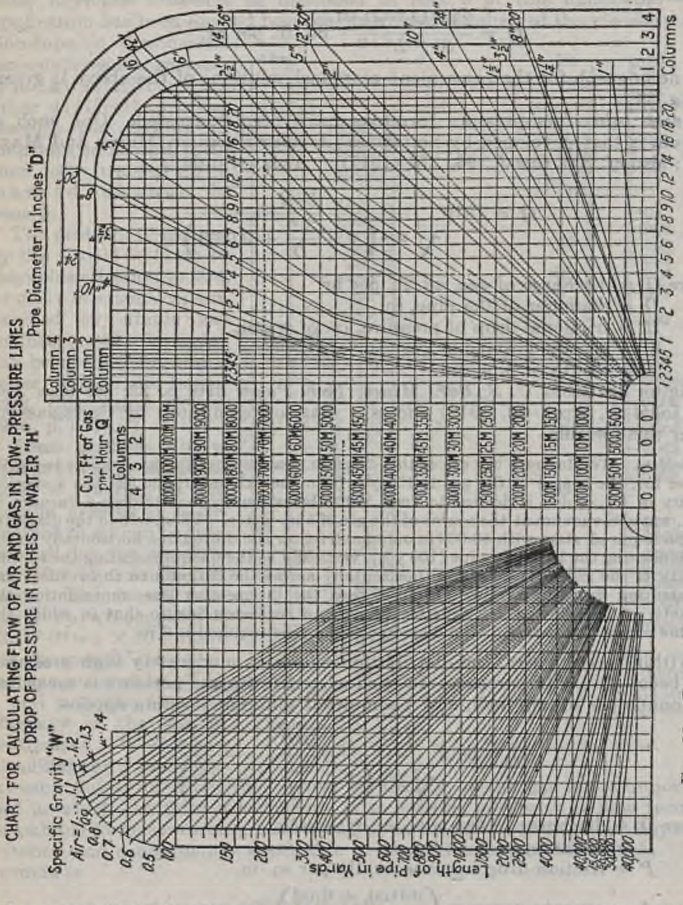


FIG. 32.—Drop of pressure in inches of water for air and gas in low-pressure lines.

Recently the U. S. Bureau of Mines in cooperation with the Natural Gas Department of the American Gas Association has been studying the flow of natural gas through **high-pressure** transmission lines. In this important work, sometimes involving lines hundreds of miles in length, the choice of an accurate formula is very desirable. The reported results obtained favor the use of a formula developed by Weymouth [*Trans. Am. Soc. Mech. Eng.*,

34, 1091–1104 (1912)] (cf. Berwald and Johnson, *U. S. Bur. Mines, Rept. Investigations*, 3153, December, 1931). This is:

$$Q_c = 18.062\frac{T_0 + t_1}{P_0}\sqrt{\frac{(P_1^2 - P_2^2)d^{5\frac{1}{3}}}{GTL}}$$

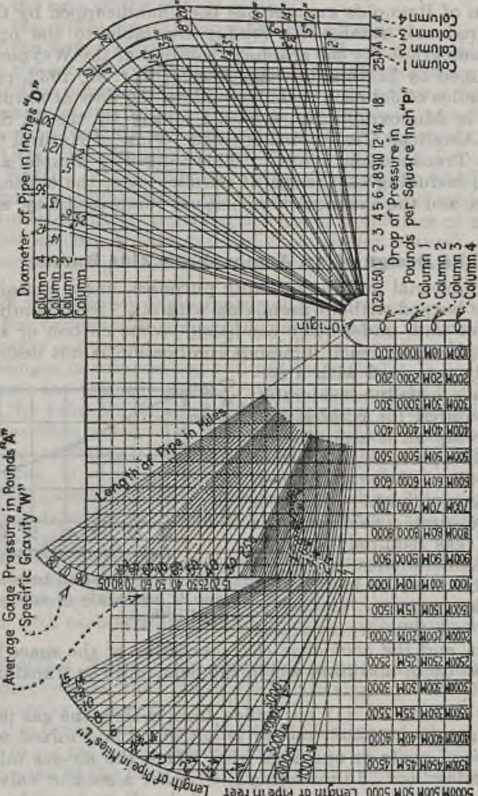


FIG. 33.—Diagram for calculating flow of air and gas in high-pressure lines.

where Q_c = number of cubic feet of gas flowing per hour, corrected to 60°F., and base pressure (14.4 lb. per sq. in.).
P_1 = absolute inlet-line pressure, lb. per sq. in.
P_2 = absolute outlet-line pressure, lb. per sq. in.
T = absolute flowing temperature, °F., *i.e.* $(T_0 + t_2)$.
L = length of line, miles.
T_0 = absolute temperature, 460°F.

P_0 = absolute pressure base (14.4 lb. per sq. in., as generally adopted in natural-gas engineering practice).

t_1 = 60°F.

t_2 = observed flowing temperature, °F.

G = specific gravity of gas (air = 1).

Weymouth's treatment more nearly approaches the flow treatment based on Reynolds number (see pp. 799*ff*. on Flow of Fluids). However, the practical application of Reynolds number has been handicapped by the fact that the data required are usually not readily available to the operating gas engineer. A nomograph for the graphical solution of the Weymouth formula has been developed by Mathis (*Gas Age-Record*, Apr. 30, 1932, pp. 536–538).

Extensive studies of Johnson and Berwald on the flow of natural gas have been compiled in Monograph No. 6 issued jointly by the U. S. Bureau of Mines and the American Gas Assoc., entitled "Flow of Natural Gas through High Pressure Transmission Lines." This considers simple and complex piping systems, including the design of parallel lines, the storing of natural gas in pipe lines, and the chemical composition, compressibility, and viscosity of natural gas.

The Design of an Atmospheric Gas Burner

For certain industrial and illuminating purposes, gases containing hydrocarbons may be burned without premixing with air. Such combustion gives a luminous flame and, under some conditions, some carbon or smoke. For many purposes, however, such luminous combustion is not desired, and the gas is ordinarily premixed with a certain amount of air, usually considerably less than is required for combustion. When this primary air is aspirated by a jet of gas at low pressure, the burner is said to be an atmospheric gas burner.

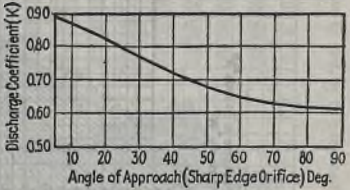


FIG. 34.—Relation between orifice constant *K* and angle of approach for sharp-edged orifice.

In the design of burners for such combustion, the first consideration is the volume of gas required; the next is the flame characteristics; thus, if a sharp, hot flame must be had, a high primary air-gas ratio is necessary. The minimum gas pressure available must be known, and the geometric characteristics of the space in which the combustion is to occur and from which the heat must be transferred must be developed.

The burner consists of: (1) an orifice from which the gas is discharged, (2) an injecting tube in which the air is entrained and mixed with the gas, and (3) a series of ports or openings from which the air-gas mixture is discharged for combustion. In installing the burner a cock or valve capable of regulating the gas flow is ordinarily provided. Care must be taken so to place the burner that air, substantially uncontaminated with combustion products, is freely available at the air-shutter opening of the mixing tube under conditions of maximum demand, and provision must also be made to permit the ready flow of the requisite secondary air to the ports and to permit the free discharge of the combustion products. The inner cone of the flame must never impinge an object, as the incomplete combustion resulting may produce dangerous amounts of carbon monoxide.

Orifice. In selecting an orifice, a high coefficient of discharge is desirable, as entrainment of more air may be obtained. An orifice of the sharp-edged type is preferred. The rate of flow of gas from such an orifice may be computed by means of the formula

$$Q = 1658.5AK\sqrt{\frac{H}{D}}$$

where Q = the quantity of gas flowing, cu. ft. per hr.
A = the area of the orifice, sq. in.
K = the orifice constant or discharge coefficient.
H = the orifice pressure, in. of water (above the atmosphere).
D = the specific gravity of the gas as it approaches the orifice (air = 1).

Up to an orifice pressure of 10 in. of water, no serious error due to gravity changes because of increased pressure at the orifice is introduced.

The various values of K for different types of sharp-edged orifices are shown in Fig. 34 (*Nat. Bur. Standards U. S., Tech. Paper* 193; *Sci. Paper* 359).

Air-gas Ratio. On p. 2411 were given formulas by which the air required for the complete combustion of ordinary fuel gases could be computed. Many manufactured city gases require about 5 volumes of air to one of gas, with a corresponding air-gas ratio of 5; while for natural gas requiring for complete combustion 10 volumes of air per volume of gas, the air-gas ratio is 10.

In Fig. 23 were shown flame speeds of mixtures of a number of fuel gases with air. For flexible atmospheric burner operation it is desirable to operate with a primary air-gas ratio in the mixing tube of such a character that ordinary changes in composition will not be critical. In older manufactured gas appliances the primary air-gas ratio is from 1½ to 2, *i.e.*, from 30 to 40 per cent of the total air required. With advance in the technique of burner design, it has been possible to secure much higher percentages of aeration in the primary mixture. Where access of secondary air to the ports is somewhat restricted, as in some radiant-type heaters fitted with ceramic grilles, the percentage primary aeration may be as much as 80 per cent. Such high primary air-gas ratios are, however, more critical and are avoided where flexibility is important.

If the primary air-gas ratio is too low, the flame lengthens unduly and may produce carbon monoxide if impingement results. The tips may turn yellow and the combustion smoky. Too high primary air-gas ratio may lead to strike backs with carbon monoxide production, or blowoffs which are equally undesirable.

The Injecting Tube. To secure maximum air entrainment with a minimum gas pressure, the injecting or air-mixing tube should be in the form of a venturi (refer to Sec. 6).

In *Bur. Standards Tech. Paper* 193 the following are given as some items of importance in regard to the injector tube:

a. The change of the lines of approach of the inlet to the outlet should be gradual.

b. The approach should follow approximately a curvature which should not be less than 3-in. radius for a ⅝-in. throat, and other sizes should be in proportion. For this size tube, the orifice should discharge about 1.5 in. from this throat.

c. The outlet angle should be about 2 degrees. The outlet tube should not be too short. Six throat diameters is the minimum length for good service. A satisfactory area of the throat can be determined by multiplying the area of the ports by 0.43.

The Bureau of Standards (*Circ.* 394) develops the relationship between the burner design and the momentum of the gas stream with particular reference to the entrainment of the air. The product of the mass and velocity of the jet of gas, as it leaves the orifice, bears a sufficiently constant ratio to the momentum of the mixture of gas and primary air as it passes any definite cross section of the burner to permit the use of a constant in most

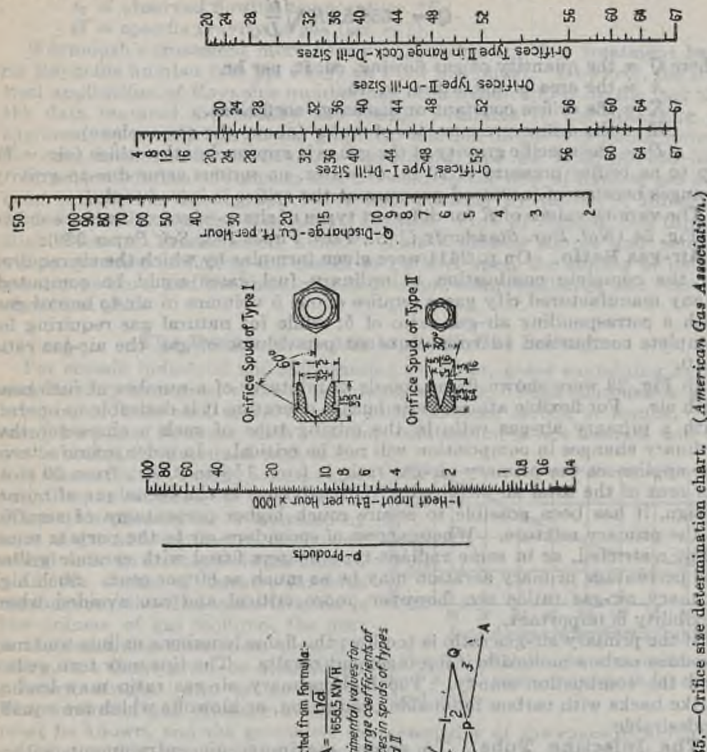


FIG. 35.—Orifice size determination chart. (*American Gas Association.*)

problems of design, *i.e.*, $MV/mv = C$. The value of C is the same for burners of different sizes but of the same geometric design. The considerations are approximate and subject to some modification as the burner heats up and with increasing pressure and low gravity gases. According to this, geometrically similar burners have a characteristic injecting constant

$$k = \frac{Q}{q}\sqrt{\frac{aD}{d}}$$

where k = the burner constant.
q = the quantity of gas flowing in unit time.
Q = the volume of the air-gas mixture flowing in unit time.
a = the area of the orifice.
D = the density of the air-gas mixture.
d = the density of the gas.

The value of k should be determined experimentally but should not be assumed to be greater than $0.8\sqrt{P}$, where P is the total port area, although well-designed burners, properly aligned with a sharp-edged orifice, have shown values as great as $1.2\,P$.

A number of useful relationships may be derived from this treatment of the burner constant.

It has been recommended that free area of the mixer head, through which the air is admitted, be approximately fixed at 1.25 times the port area.

The Burner Head. Considerations of importance relating to the burner head are the port area, the depth and coefficient of discharge of the ports, the individual port area, the spacing of the ports, the burner volume, and the distribution of secondary air.

For natural gas, butane, and propane 12,000 to 15,000 B.t.u. can be burned per square inch of port area; for manufactured gases, from 22,000 to 26,000 B.t.u. per sq. in. of port area. For universal burners for use on either fuel, around 20,000 B.t.u. per sq. in. should be selected. In general, the total port area should be made as large as possible. The figures above are for minimum areas.

The port relations must be such that the flame will not strike back owing to too low a velocity of air-gas flow nor must it blow off owing to too high a velocity. To secure high air-gas ratios without strike back, ample momentum must be imparted to the air-gas mixture to ensure adequate pressure within the burner head to cause a proper velocity of flow through deep burner ports.

Kowalke and Ceaglske (*Am. Gas Assoc., Proc.*, 1929, pp. 662–686) develop certain relationships between the air-gas ratios and the ratio of throat area to port area and orifice area to throat area. The work was further developed (*Proc. Wis. Utilities Assoc.*, 1930). The equations obtained in this work have been rearranged ("Combustion," 1932 ed., published by American Gas Assoc.) as follows:

$$A_P = \frac{0.85}{C} \times (R + 1) \times \frac{d + R}{d} \times A_o$$

where A_P = the port area, sq. in.
A_o = the area of the orifice, sq. in.
C = coefficient of discharge of the ports.
R = air-gas ratio.
d = density of the gas.

The value 0.85 is characteristic of the burner and is applicable only to a well-designed burner, such as those using injecting tubes meeting the suggestions of the Bureau of Standards. Values from 0.35 to 0.7 have been found.

The general form of the above equation is as follows:

$$A_P = \frac{K^2 A_0}{CF} \times \frac{(R+1)(R+d)}{d}$$

where K is the discharge coefficient of the gas orifice (values ranging from 0.75 to 0.85) and F is the injector characteristic.

Arrangement of ports in more than two rows without access of secondary air between them should be avoided. Burner ports should not be drilled too close to the injector tube, and the air-gas stream should not be deflected against certain ports. Care should be taken to secure substantially equal pressure on all ports. It is difficult to estimate the coefficient of discharge of the ports because of the various ways in which ports may be made. For rough approximations, values from 0.6 to 0.8 may be used, although values from 0.45 to 1.00 have been found. Drilled ports are preferred. No gauze should be placed in the burner. For small iron or steel burners the burner head, tube, and injector should be cast in one piece and not cemented.

Individual port areas are important factors in stability and blowoff or strike back. The following areas should not be exceeded:

For manufactured gas: No. 40 M.D.S. (0.0075 sq. in.).

For natural gas: No. 30 M.D.S. (0.013 sq. in.).

For propane and butane: No. 32 M.D.S. (0.0106 sq. in.).

Universal burners require small ports.

Wills [*Western Gas*, August, 1931 (this author gives another approach to the problem of burner design)] recommends that the free cross-sectional area of the burner head be three times the area of the ports to be supplied. If an even distribution of heat is required, the difference in velocity of the air-gas mixture between the first and last ports must not be too great.

Air-shutter Opening. The Bureau of Standards recommends that the possible air-shutter opening should be made so large that the linear velocity through the opening does not exceed 4 or 5 ft. per sec.

GAS AS AN INDUSTRIAL FUEL

Recent years have witnessed a marked extension in the use of gas as a fuel for industrial purposes. On the preference for gas for these purposes, the Surface Combustion Company states:

> By establishing practice with gas as a fuel, a standard of comparison is set up which permits a determination of performance in advance of actual operation that cannot otherwise be obtained . . . The fuel which is amenable to elimination of most variables in application is clean gas . . .
>
> With gas it is possible to eliminate the uncertainty of flame length, heat distribution, excess oxygen or fuel, rate of flame propagation, and all those variables which are characteristic of other fuels . . . It is our belief that the use of other fuels represents at the present time an expression of economic conditions which are transient in character. From the specifications of control, availability, cleanliness, efficiency in utilization, and national economic soundness, gas has . . . marked advantages over competing fuels.

Furnace Atmospheres. Three classes of furnace atmospheres have been recognized, namely, the oxidizing, the neutral, and the reducing atmospheres.

The American Gas Association characterizes these as follows:

Name of atmosphere	Oxygen, %	Carbon monoxide and hydrogen
Oxidizing	Over 0.05	Not over 0.5% total $CO + H_2$
Neutral	Not over 0.05	Not over 0.5% total $CO + H_2$
Reducing	Not over 0.05	Over 0.5% total $CO + H_2$

This classification is for purposes of definition and clarification only and does not rest upon the behavior of the furnace atmosphere toward specific materials such as metals at elevated temperatures.

Carbon dioxide, water vapor, and oxygen will scale metals, while CO and H_2 are reducing. Neutral atmospheres produced by ideal combustion conditions are actually scaling. A better classification of furnace atmospheres is based upon the effect upon the product as scaling, carburizing, decarburizing, nitriding, bright annealing, etc. Much of the information on furnace atmospheres is empirical in nature and based upon imperfect analytical procedures.

In furnaces in which atmospheres must be controlled, two classes may be recognized:

1. Furnaces in which the work is in contact with the combustion atmosphere.
2. Furnaces in which the heating is indirect and the work is subject to a controlled or controllable atmosphere, as in muffle furnaces.

Certain remarks and precautions about the choice of industrial burner systems are apropos here. The systems will be described later. Thus atmospheric burners should not be used for the production of reducing atmospheres as the low burner-head pressure may permit the admission of pockets of excess air and consequent explosions. Pressure burners are to be preferred over atmospheric burners where wide flexibility is desired. Diffusion-flame burners possess certain advantages where low flame temperatures and high rate of heat transfer is desired. These burners allow precipitations of carbon and a blanket of reducing gas. Nozzle-mixing burners do not allow satisfactory control of atmospheres on turn-down conditions. Premixed systems are very successful in the production of controlled atmospheres.

In the so-called protective atmospheres water vapor is frequently found very deleterious and must be removed. Furnace technique for special work has now become so developed that it is possible to remove undesirable gases, such as water vapor and carbon dioxide from combustion gases almost completely.

It will be obvious to the chemical engineer that the action of furnace atmospheres and their effective control involves a consideration of the thermodynamics of the various oxidizing and reducing actions which the product may undergo and the thermodynamics of the furnace atmosphere. Thus, in bright annealing, the constituents in the furnace atmosphere must not dissociate to give available oxygen in partial pressures sufficient to repress the dissociation of the oxide of the product, as otherwise the product would tarnish or scale. It is obviously impossible to discuss the many applications of gas to controlled furnace atmospheres and the precautions that must be observed. The American Gas Association through appropriate committees on industrial gas research has and is engaged upon studies that are helpful, and the reports of such committees and booklets of that association, particularly "Combustion" and "Fuel-flue Gases" (in press, 1939) may be consulted.

Industrial Combustion Systems. Combustion systems have been classified by the American Gas Association* as follows:

- I. Atmospheric.
- II. Pressure.
  - *A.* Air compressed
    - (1) without automatic proportioning.
    - (2) with automatic proportioning.
  - *B.* Gas compressed
    - (1) without automatic proportioning.
    - (2) with automatic proportioning.
  - *C.* Mixture compressed
    - (1) without automatic proportioning.
    - (2) with automatic proportioning.

It will be impossible to describe in detail the various types of apparatus available for these systems, therefore merely the general principles involved will be given, somewhat along the lines adopted by the Gas Association.

The object of the various combustion systems is to provide accurate regulation of the air-gas ratio in the mixture supplied to the burner ports and an accurate control of the rate of gas supply.

Diffusion-flame Burners. Since the above classification was prepared, the industrial use of burners in which there is no premixing of air with gas has extended greatly.

These burners depend upon the diffusion of all the air required for combustion. They are known as diffusion-flame or luminous-flame burners. They are characterized by a luminous flame of high radiant emissivity and a low flame temperature. They will be referred to in the concluding paragraphs (p. 2430).

I. The Atmospheric System

For operations requiring temperatures up to 1000°F., and where an ample supply of secondary air to the combustion chamber is assured, atmospheric burners can be applied successfully. Properly designed burners will operate without flashing back when the gas flow is reduced to one-tenth the rated consumption, thus permitting thermostatic control. The simplicity of the atmospheric system causes it to be widely used, but the necessity for a positive draft and the long time required to attain furnace temperatures from 500° to 1000°F. often causes it to be rejected in favor of a special combustion system.

Some of the considerations involved in the design have already been set forth in the division entitled The Design of an Atmospheric Gas Burner. Additional considerations follow:

1. Each individual burner port should receive its supply of secondary air by drilling the ports in raised teats when this is possible, and the ports should be so arranged that secondary air is readily accessible to every individual port.
2. The burner body should be as small as possible consistent with good operation in order to minimize flash-back objections.
3. No cement joints should be used as these may crack under heat and leak gas.

---

* In the booklet "Combustion" 2d ed., 1926, prepared under the direction of the Industrial Gas section of the association. Permission to use material contained therein is gratefully acknowledged.

4. Burners should be made of cast iron.

5. Air mixers should have flat faces and be capable of being rigidly fixed at any desired adjustment.

6. Burners and manifolds should be rigidly connected to their appliance.

7. Standard A.G.A. orifice spuds and gas cocks are available for gas consumptions up to 80 cu. ft. per hr. and should be used when possible.

8. The use of wire gauzes in burners should be avoided.

9. The burner should, where possible, be placed close to the appliance without causing the cone to impinge and without causing floating from the banking of burned gases.

10. Free exit of burned gases from under the heated surface must be provided by placing the burner sufficiently low.

11. A draft inducing an excess of secondary air should be reduced by a permanent restriction in the flue. No dampers should be used.

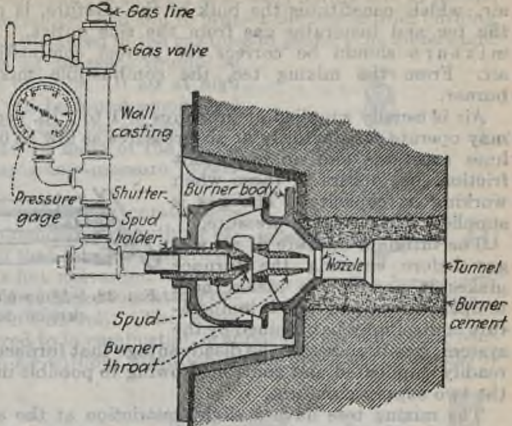


Fig. 36.—Atmospheric industrial gas-burner installation with individual air control to each burner.

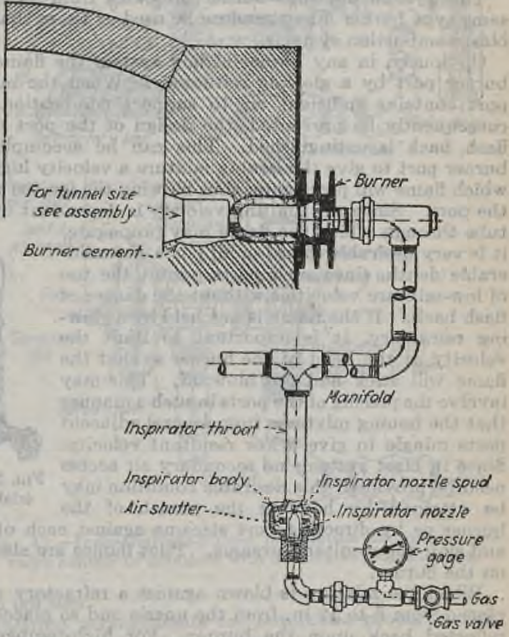


Fig. 37.—Atmospheric industrial gas burner with manifold mixing.

Figure 36 illustrates one of the many forms of atmospheric burners.

II. Pressure Systems

A(1): Air Compressed without Automatic Proportioning (Two-valve Control). Next to the atmospheric system, the simplest combustion system consists of independently controlled air and gas lines meeting in a mixing tee. The air, which constitutes the bulk of the mixture, is passed straight through the tee and inspirates gas from the side outlet. The proportions of the mixture should be correct for perfect combustion, without secondary air. From the mixing tee, the combustible mixture is passed to the burner.

Air is usually supplied at pressures of 1 to 2 lb. per sq. in., but the system may operate satisfactorily at air pressures as low as 6 in. water, if the supply lines are large and no important friction loss occurs in these when working at capacity. The gas is supplied at the service pressure.

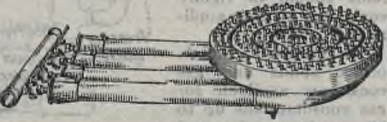


FIG. 38.—Atmospheric industrial gas burner, circular, four-range capacity.

The intimate mixture of air and gas before entering the furnace makes it possible to attain high temperatures, and good temperature control may be obtained by the system; but it possesses the disadvantage that furnace atmospheres cannot be readily duplicated and controlled owing to possible independent variations in the two supply pressures.

The mixing tees have a slight restriction at the air-supply end to create a suction at the point where the gas enters the mixture. Many mixing tees are made from a simple pipe fitting without an inspirating device.

This system develops burner pressures from ½ to 8 in. water, and the same type burner may therefore be used as those that are employed in other blast-combustion systems.

Obviously, in any burner of any system the flame may be held near the burner port by a glowing refractory. When the mixture issuing from the port contains sufficient air to support combustion, and flash back must consequently be prevented, the design of the port must be such that such flash back is extinguished. This can be accomplished by designing the burner port to give the issuing mixture a velocity higher than the velocity at which flame will propagate, thus blowing the ignited mixture to the mouth of the port. Since the limiting velocity is dependent upon the diameter of the tube through which the flame may propagate, it is very desirable to use small ports of considerable depth, since such ports permit the use of low-mixture velocities without the danger of flash back. If the flame is not held by a glowing refractory, it is important to limit the velocity at the head of the burner so that the flame will stick and not blow off. This may involve the placing of the ports in such a manner that the issuing mixtures from several adjacent ports mingle to give a low resultant velocity. Since in blast systems no secondary air access need be provided, this desirable condition may be attained by hooding the mouth of the burner or by directing port streams against each other to interfere, blend, and slow the resultant streams. Pilot flames are also used to hold the flame on the burner.

FIG. 39.—Atmospheric industrial gas burner, ribbon type.

When the mixture is blown against a refractory material, this should be placed from 6 to 12 in. from the nozzle and so placed that the gases are not reflected back upon the burner. For high-temperature work the nozzle should be bedded into a refractory wall with side walls flaring out to permit the mixture to reach the refractory bed and to protect the nozzle from burning out. Check valves should be placed in the gas lines to protect the meters and gas-supply system. The air pressure should be supplied by low-pressure blowers. If air at high pressures is available, it will be found economical to install an air inspirator so that the energy in the high-pressure air will draw in most of the air required for combustion, or a low-pressure blower should be installed.

The pressure systems without automatic proportioning are not recommended for anything except installations so small that the expenditure for more elaborate systems is not warranted.

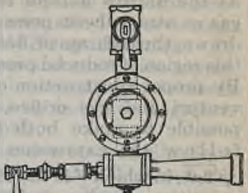


Fig. 40.—Gas burner with automatic proportioning.

A(2): Air Compressed with Automatic Proportioning (Single-valve Control). The automatic proportioning referred to in combustion is applied to the automatic control of the ratio of fuel to air without attention by the operator. Because of fluctuating supply pressures and changing flow rates, separate valve controls for fuel and air cannot be manipulated in such a manner as to ensure the maintenance of a given furnace atmosphere at all times. Consequently, various devices have been developed to regulate the amount of mixture admitted to a burner or a set of burners. A homogeneous constant-composition mixture is thus ensured. The composition can, of course, be set as desired.

A thoroughly mixed and correctly proportioned gas-air blend suitable for complete combustion is the ideal mixture to discharge from the burner nozzle. A minimum of combustion space is required because combination occurs immediately upon ignition.

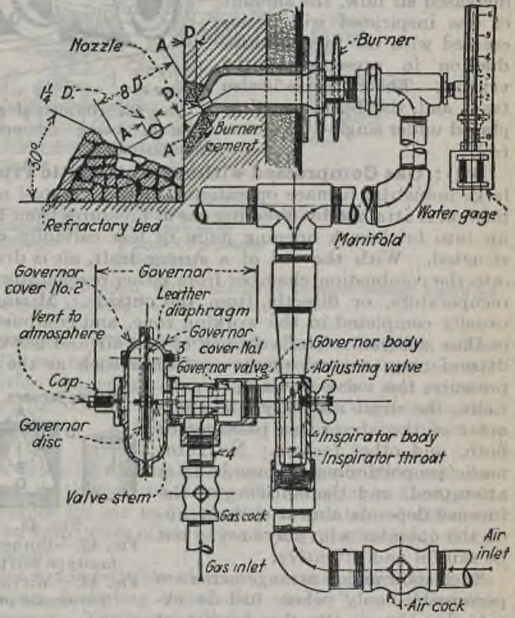


Fig. 41.—Industrial gas-burner installation showing single-valve control to manifold with automatic proportioning.

Figures 40 and 41 illustrate arrangements for automatic proportioning.

The system referred to in this section operates by utilizing the energy in the air to inspirate gas which has been reduced to atmospheric pressure by a diaphragm regulator. The air is supplied at from 1 to 2 lb. per sq. in. At the narrow part of the venturi, the pressure is below atmospheric, and gas at atmospheric pressure is drawn through an orifice into this region of reduced pressure. By proper construction of the venturi and gas orifice, it is possible to make both fluids follow the expression $V = \sqrt{2gh}$, in which V is the stream velocity; g is the acceleration due to gravity = 32.16 ft. per sec.; and h is the head or pressure difference causing the flow. A given air flow will inspirate a given gas flow; with increased air flow, the amount of gas inspirated will be increased with the increased reduction in pressure at the venturi. The converse is also true, and the burner is thus placed under single-valve control.

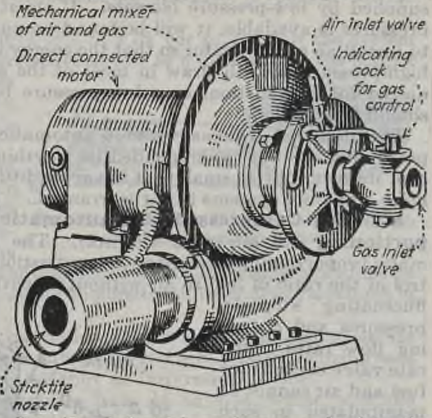


Fig. 42.—Industrial gas burner and mixer, the mixture being compressed without automatic proportioning.

B(1): Gas Compressed without Automatic Proportioning. In many large industrial furnace operations, such as in steel mills, the requisite combustion is attained by directing gas at pressures from 1 lb. up into brickwork opening more or less carefully constructed. With the aid of a strong draft, air is drawn into the combustion chamber from either regenerators or recuperators, or directly from the outside. Mixing is usually completed in the working zone, and combustion is thus delayed. The volumes of the air and gas used depend upon a great many conditions such as the gas pressure, the valve or damper positions, the draft intensity, the character of the air or gas passages or both, and other factors. No automatic proportioning or premixing is attempted, and the efficiency of the furnace depends almost entirely upon the operator who may or may not be skillful and attentive.

Such combustion arrangements are permissible only when fuel is extremely cheap. (Further treatment of this system of combustion will be found in Trinks, "Industrial Furnaces," Wiley, New York, vol. I, 1923, vol. II, 1925.)

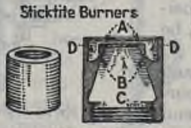


Fig. 43.

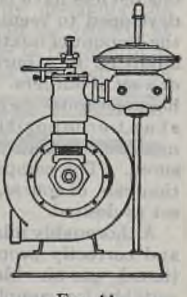


Fig. 44.

Fig. 43.—Burner to hold the flame on fan-type mixture compression system.

Fig. 44.—Mixture compressed with some automatic proportioning—fan type.

B(2): Gas Compressed with Automatic Proportioning. This system is generally termed the *high-pressure* system. The compressed gas is carried to the point of application without the use of fire screens, fire checks, etc. All the air for combustion is injected by the gas through the use of patented inspirators which proportion the amounts of air and gas at the appliance. The air-gas ratio is determined by the size of the orifice admitting gas to the device, and the maintenance of the desired proportions depends upon the principles indicated under $A(2)$, Air Compressed with Automatic Proportioning.

C(1): Mixture Compressed without Automatic Proportioning. In this system, a fan blower draws air from the atmosphere and gas from a supply at normal service pressure into the fan casing where a homogeneous mixture is at once formed. A double valve turns the air and gas up and down together. This system is designed for constant heat requirements.

This type system can be used chiefly for single-burner nozzle applications and for work which requires little or no turn down and which will permit the fan to be placed close to the furnace without overheating. Multiple burner use, involving long manifold piping must be avoided, as flash backs may occur with variable burner pressures. Figure 42 shows such an installation and Fig. 43 shows the type of nozzle used.

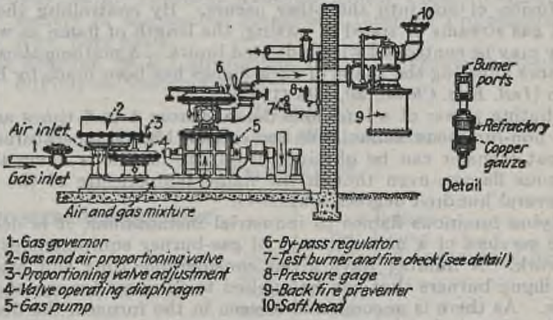


FIG. 45.—Industrial-gas air-mixture system with automatic proportioning, manifold burner type, showing burner construction.

C(2): Mixture Compressed with Automatic Proportioning. The fan blower described in $C(1)$ can be greatly improved in reference to automatic proportioning by the addition of a gas governor. If this is done, and suitable admission ports are chosen for the gas and the air, a certain degree of automatic proportioning is obtained. A fan blower so equipped is shown in Fig. 44.

A much more elaborate type, in which all the air for combustion mingles with the gas homogeneously is that of the Kemp Company; it is illustrated in Fig. 45. Admission valves for air and gas with a number of different settings are provided to give such ratios of air to gas as may be desired. Once set, the ratio remains constant.

Since the system handles a combustible mixture, various precautions are used to prevent back firing, including "back-fire checks" and specially designed burners with screens. Explosion relief heads are also furnished, but these are added precautions very rarely, if ever, needed.

Another subdivision is used to classify **mixing compressors drawing in only a portion of the air required,** usually less than that necessary to support combustion. The air-gas ratio is usually 2:1. This mixture injects the remaining air at the burners. The amount injected here will of course depend somewhat upon the furnace conditions and burner adjustments, so the system is not completely automatic in proportioning. Flash back may occur in these burners if the mixture flow is reduced to one-third normal operations. Automatic heat control of the gradual shutdown type is therefore applicable only when the low consumption is greater than one-third the normal.

Diffusion-flame or Luminous-flame Burners. It is claimed that diffusion-flame or luminous-flame burners offer the following advantages:

1. An increased rate of heat transfer.
2. More uniform temperature control.
3. Less maintenance of furnace refractories.
4. Greater degree of turn down of burners.
5. Less scale is formed, and such scale as may be formed is more easily removed.

Diffusion- or luminous-flame burners admit little or no primary air to the burner. The air and gas streams emitted from the burner nozzle should be free of turbulence and should travel along the furnace at the same speed, burning taking place only at the surfaces of the individual streams of gas and air as diffusion of one into the other occurs. By controlling the velocity of air and gas streams or speed of mixing, the length of flame as well as the luminosity may be controlled within desired limits. A mathematical analysis of the factors affecting the shape of such flames has been made by Burke and Schumann [*Ind. Eng. Chem.*, **20**, 998 (1928)].

The radiating power of a luminous flame is from 4 to 6 times as great as that of a non-luminous flame; thus for certain high temperatures a faster rate of heat transfer can be obtained from the luminous flame than from non-luminous flames, even though the flame temperature of the luminous flame is several hundred degrees lower.

In applying luminous flames to industrial installations, it is desirable to secure the services of a manufacturer of gas-burner equipment experienced in such work. A number have given considerable time and attention to luminous-flame burners that can be applied to furnaces for various types of operations. As there is uncombined oxygen in the furnace, the design must be such that this is kept away from the work if scaling is to be avoided. While the luminous carbon particles indicate temporary incomplete combustion, burners properly designed and operated will give complete combustion of the fuel before it leaves the furnace.

Luminous-flame burners can be supplied in a design such that if the gas is purchased on an off-peak basis, oil can be burned in them when the gas supply is interrupted. Gas varying from 350 to 1500 B.t.u. can be used in the same burner and the change-over can sometimes be made without loss of furnace temperature. If gas is available at a sufficiently high pressure, the requisite air may be inspirated; if not, the air may be supplied by a fan. [Other references on luminous-flame burners: Wood, *Gas Age-Record*, **67**, 351 (1931); Dare, Paper before Am. Gas Assoc. Industrial Gas Sales Conference, Cleveland, March, 1939].

22

SECTION 22

POWER GENERATION AND MECHANICAL POWER TRANSMISSION

BY

A. D. Blake, M.E., Editor, *Combustion;* Formerly Managing Editor, *Power;* Member, American Society of Mechanical Engineers; Licensed Professional Engineer, New York State.

R. B. Purdy, M.E., Formerly Associate Editor, *Power;* Member, American Society of Mechanical Engineers; Licensed Professional Engineer, New York State.

William Staniar, M.E., Mechanical Power Transmission Specialist, E. I. duPont de Nemours & Co.

REVISED BY

R. B. Purdy and William Staniar

CONTENTS

PAGE

POWER GENERATION

MECHANICAL POWER TRANSMISSION

POWER GENERATION

BY A. D. BLAKE AND R. B. PURDY

References: Croft, rev. by Purdy, "Practical Heat," McGraw-Hill, New York, 1939. Croft, rev. by Tangerman, "Steam-engine Principles and Practice," McGraw-Hill, New York, 1938. Croft, rev. by Purdy, "Steam Boilers," McGraw-Hill, New York, 1937. Ripper, "Steam Engine Theory and Practice," Longmans, New York, 1932. Stadola (trans. by Loewenstein), "Steam and Gas Turbines," McGraw-Hill, New York, 1937. Barnard, Ellenwood, and Hirshfeld, "Heat Power Engineering," Wiley, New York, 1926, 1933. Gebhardt, "Steam Power Plant Engineering," Wiley, New York. Butterfield, Jennings, and Luce, "Steam and Gas Engineering," Van Nostrand, 1938. Craig and Anderson, "Steam Power and Internal Combustion Engines," McGraw-Hill, New York, 1931. Norris and Therkelsen, "Heat Power," McGraw-Hill, New York, 1939. Severns and Degler, "Steam, Air and Gas Power," Wiley, New York, 1939. Walker and Crocker, "Piping Handbook," McGraw-Hill, New York, 1939. Baumeister, "Fans," McGraw-Hill, New York, 1935. Keller, Marks, Weske, "The Theory and Performance of Axial-flow Fans," McGraw-Hill, New York, 1937. Rudorff, "The Superheater in Modern Power Plants," Pitman, New York, 1939. Powell, "Boiler Feed Water Purification," McGraw-Hill, New York, 1927. Morrison, "Diesel Engineering Handbook," Diesel Publications, New York, 1940. Anderson, "Diesel Engineering," McGraw-Hill, New York, 1938. Von Bongart, "Diesel Engines," Van Nostrand, New York, 1938. Creager and Justin, "Hydro-electric Handbook," Wiley, New York, 1927. Barrows, "Water Power Engineering," McGraw-Hill, New York, 1927. Clower, "Lubricants and Lubrication," McGraw-Hill, New York, 1939. White, "Engineering Materials," McGraw-Hill, New York, 1939.

STEAM PRESSURES

No definite rule can be given for the selection of steam pressures as much depends upon local conditions such as size of plant, investment warranted, cost of fuel, load factor, use of exhaust steam, or simplicity of operation.

In central-station work, practice is divided as between 900 lb. per sq. in. and 1200 to 1400 lb. per sq. in., with a noticeable trend toward the higher pressure in many of the newer stations. The 1200 to 1400 lb. per sq. in. station will show a reduced heat consumption per kilowatt-hour of 10 to 15 per cent compared with a 300 to 400 lb. per sq. in. station, but to offset this there is usually an increased investment cost of at least 5 per cent. Where fuel is very cheap, or the load factor low, this increase in efficiency may not warrant the additional fixed charges of the high-pressure plant. It is felt that 1400 lb. per sq. in. is the limit in pressure that should be employed without reheating the steam between the high- and low-pressure elements of the turbine, otherwise too much moisture will be present in the low-pressure stages. These pressures contemplate initial steam temperatures of 900° to 950°F. which are the highest generally employed with present materials.

The accompanying curves (Fig. 1) reproduced from a paper by Frank S. Clark before the World Power Conference in Tokyo, October, 1929, present the results of some studies on high and low steam pressures for central stations employing the regenerative cycle (the cycle in which steam is extracted from one or several stages of the turbine for feed-water heating).

An increasing number of process industries are now employing 400 lb. per sq. in. steam pressure in their power boilers; a smaller number around 600 lb. per sq. in. and a few still higher pressures; the Ford Motor Co. at Detroit has adopted 1400 lb. per sq. in. and the Philip Carey Co. at Lockland, Ohio, 1800 lb. per sq. in.

The governing factors in an industrial plant employing exhaust steam for process are the power requirements, exhaust pressure, and quantities of the process steam required (see Heat Balance*). A boiler pressure should be selected such that the steam in passing through the turbine or engine will produce the required amount of power and make available sufficient exhaust at the desired pressure (or pressures if an extraction turbine is employed) without an appreciable amount being wasted. It is seldom possible to operate for long with a perfect balance between power and process steam demands, and it is, therefore, better so to proportion the plant that any deficiency in process requirements will be made up by boiler steam through a reducing valve than to have an excess of exhaust steam go to waste. In some plants the installation of a mixed-pressure turbine may be warranted to utilize such excess exhaust.

In an industrial plant utilizing the exhaust steam for process, it is usually possible to generate 1 kw.-hr. on 4500 to 5000 B.t.u. or about one-third that required in a straight condensing plant. Where very little or no steam is required for process, the power requirements and local conditions will determine whether it is more economical to generate or to purchase power and put in a boiler plant for heating requirements at relatively low pressure.

In a few cases arrangements are in force between central stations and large industrial units whereby the former supply the latter with the exhaust from one or more high-pressure turbines and as much power as may be required. These set-ups involve rather complicated contracts to take care of fixed charges, demand factor, etc., but under certain conditions the arrangement is advantageous to both parties from an economic standpoint.

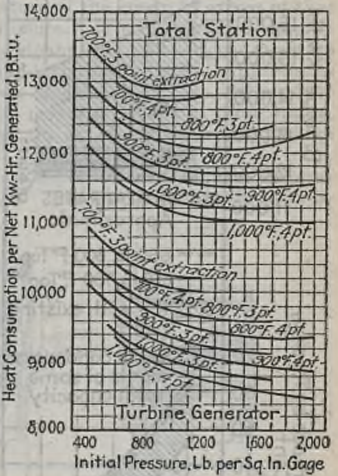


FIG. 1.—Variation of heat-consumption rate with initial pressure and temperature for a plant operating on the regenerative cycle. (*Clark*, *World Power Conference, Tokyo*, 1929.)

High-pressure boilers in this country have followed, in general, the designs of boilers for moderate pressures with certain modifications such as increased drum and header thickness to withstand the higher pressure and different arrangement and distribution of heating surfaces. Early high-pressure drums were forged from the solid ingot or forge welded. Modification of the "A.S.M.E. Boiler Code," in 1931, permitted fusion welding under prescribed safeguards, and since then practically all power boiler drums have been fabricated by fusion welding. Abroad, high-pressure boilers have followed special designs, and in general they are of much smaller capacity than those in this country. The highest pressure commercial boiler is the Benson type which generates steam at the critical pressure of 3200 lb. per sq. in. although experimental boilers have been constructed and actually operated at 5000 lb. per sq. in. pressure. One utility plant is nearing completion and will operate at 2500 lb. per sq. in. pressure, employing boilers with natural circulation. An industrial plant is operating at about 2200 lb. per sq. in. pressure.

---

* P. 2471.

It is most essential that close attention be paid to the condition of the feed water for high-pressure boilers. For this reason in many such plants, process steam is supplied by the heat of the exhaust through the medium of evaporators, so that pure condensate will be returned to the boilers.

The high steam pressures and temperatures that may now be used for power generation have resulted in many utilities adding capacity by superimposing high-pressure, high-temperature units upon existing low-pressure stations and in so doing also raising plant efficiency. Superposition or "topping," as it is often called, involves the addition of high-pressure, high-temperature boilers and a high-pressure turbine designed to exhaust at the old boiler pressure to the existing turbines.

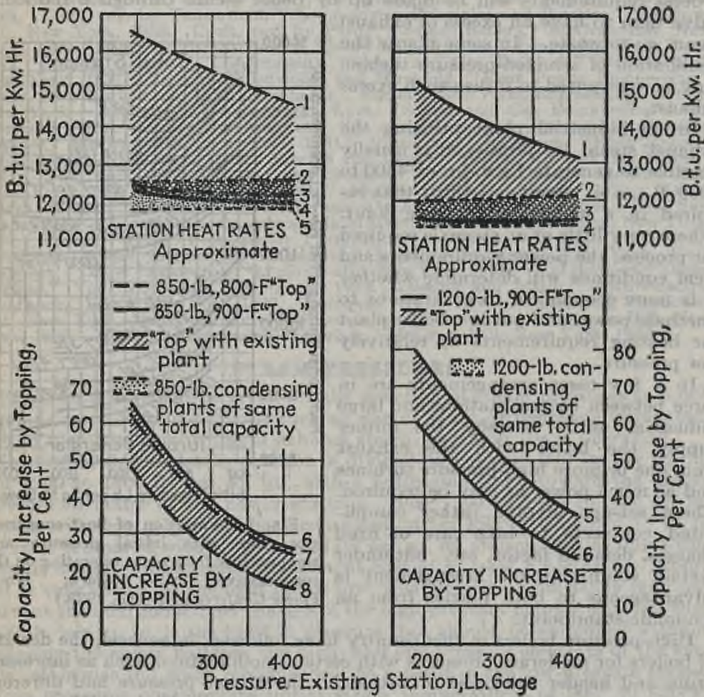


FIG. 2.—Approximate range of plant heat rate and capacity increase when topping existing plants, *i.e.*, superimposing a high-pressure plant upon an existing low-pressure plant. Curve 1 applies to tops added to existing small plants of early design. Curves 3 and 4 apply to tops added to existing plants of recent design containing large units. Curves 2 and 5 show heat rates for new condensing plants of the same capacity operating at the top pressure. Curve 2 is for small units, curve 5 for large units.

Topping can be employed to similar advantage in many industrial plants, though the conditions are somewhat more complicated by the use of process steam. In a plant where the power load has grown more rapidly than its demand for process steam, consideration should be given to a high-pressure top. The gain in efficiency and the capacity that may be added by topping depends upon the pressure and temperature of the existing plant and pressure and temperature of the top. The curves (Fig. 2, taken from an article by Clark in *Power*, 1937) show the per cent capacity increase and the heat rates that may be expected when topping existing plants that operate at pressures from 200 to 400 lb. per sq. in and for tops of 850 and 1200 lb. per sq. in.

STEAM TEMPERATURES

The amount of superheat is the temperature of the steam in excess of that corresponding to saturation at the given pressure. It is customary to employ enough superheat to ensure practically dry steam (10 to 12 per cent moisture is maximum) at the exhaust from condensing turbine or engine. Appreciable moisture in steam passing through the lower stages of a turbine will produce erosion of the blading, besides lessening the power produced in those stages. When the prime mover exhausts to process, the superheat need only be high enough to provide dry process steam.

In the condensing steam cycle, efficiency increases with increase in temperature as well as with increase in pressure. The limiting factor in increasing steam temperatures has been materials for superheaters, valves, and fittings. Until quite recently the upper limit in American practice has been around 750°F., although a few European plants have been designed for 800° to 850°F. Advances in metallurgy have warranted an increase in temperature, and many plants in this country are operating with a total steam temperature of 900° to 950°F. One experimental boiler and turbine unit of 10,000 kw. capacity at the Delray Station of the Detroit Edison Co. has operated with steam at 1000°F.

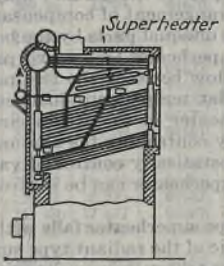


Fig. 3.

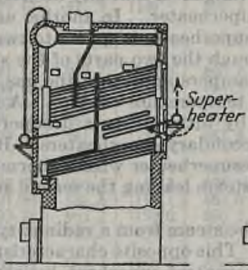


Fig. 4.

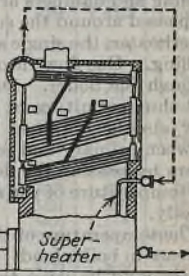


Fig. 5.

Fig. 3.—Superheater placed above boiler tubes.
Fig. 4.—Interdeck type of superheater.
Fig. 5.—Radiant superheater in rear furnace wall.

This advance in steam temperatures has made it possible to design plants for pressures as high as 1700 lb. per sq. in. without provision for reheating the steam between the high- and low-pressure elements of the turbine. With lower steam temperature and high pressure, reheating was employed in order to reduce the moisture content of the steam at the low-pressure stages of the turbine to a value that would not cause blade erosion.

In industrial plants where the exhaust from the prime mover is used in process it is usually sufficient to employ a total initial steam temperature of 650° to 700°F.; sometimes lower temperatures suffice.

The superheater, as a rule, is an integral part of the boiler setting and is interposed in the path of steam flow between the outlet from the boiler drum and the steam supply to the prime mover. In special cases where an independent control of superheat is desired, a separately fired superheater, apart from the boiler setting, may be employed. Such installations, however, are not common.

Superheaters are of two types, namely, **convection** and **radiant**. The former utilizes the sensible heat of the products of combustion and is placed either above the boiler tubes or between the upper and lower banks of tubes, in which case it is known as the **interdeck** type. The **radiant** superheater utilizes the radiant heat of combustion and is placed directly in the furnace—usually on one or more sides of the furnace. These types are illustrated in Figs. 3, 4, and 5, as applied to a horizontal water-tube type of boiler.

Steam Temperature Control (see Sec. 17). As the steam temperatures become higher the safety margin becomes less, and close control of the temperature becomes a necessity. When only a convection-type superheater is used, the steam temperature rises with an increase in boiler load. This rising temperature may be controlled by decreasing the proportion of flue gas passing over the superheater surface. One type of control involves special baffling and dampers arranged so that any desired part of the gas flow may be by-passed around the superheater. In another arrangement of compensating superheater, the single superheater is split into two unequal parts by the boiler baffling. Gas flow through the two parts of the superheater traces two paths through the boiler. Dampers apportion the gas flow between the two paths and thus permit exact temperature control. A flat temperature-load curve may also be obtained by introducing a desuperheater in the steam circuit between primary and secondary superheaters. By controlling the amount of steam by-passing the desuperheater with a thermostatically controlled valve, the temperature of the steam leaving the second superheater can be controlled exactly.

The temperature of the steam from a radiant-type superheater falls with an increase in boiler load. This opposite characteristic of the radiant type superheater may be used to control superheat by connecting it in series with a convection superheater which will compensate for the characteristic of the radiant superheater and so give uniform superheat with varying load.

Starting up of a boiler may impose very severe conditions on a superheater located in a hot zone unless an artificial steam flow is provided from another boiler or by bleeding steam from the superheated steam header.

Thermal Properties of Steam. Various tables of the thermal properties of water vapor have been published, and the most recently completed table is that published by Wiley, and prepared by J. H. Keenan, Associate Professor, and F. G. Keyes, Professor, both of Massachusetts Institute of Technology. The Steam Research Committee of the American Society of Mechanical Engineers has for a number of years sponsored researches on the properties of steam, the results of which were largely used in the preparation of these tables. The steam table data are within the tolerances set by the Third International Steam Tables Conference held in 1934.

STEAM BOILERS

The types of steam boilers employed in American power-plant practice fall into two general classes, namely, fire tube and water tube. The former are confined largely to small and medium-sized installations and moderate

Table 1. Properties of Saturated Steam[1]

Abs. press., lb. per sq. in.	Temp., °F.	Sp. vol., cu. ft. per lb.		Heat of the liquid, B.t.u.	Latent heat of evap., B.t.u.	Total heat of steam, B.t.u.	Internal energy of evap., B.t.u.	Entropy		
		Steam	Water					Water	Increase during evap.	Steam
p	t	v_g	v_f	h_f	h_{fg}	h_g	u_{fg}	s_f	h_{fg}/T	s_g
1	101.74	333.6	0.01614	69.70	1036.3	1106.0	974.6	0.1326	1.8456	1.9782
2	126.08	173.73	.01623	93.99	1022.2	1116.2	957.9	.1749	1.7451	1.9200
3	141.48	118.71	.01630	109.37	1013.2	1122.6	947.3	.2008	1.6855	1.8863
4	152.97	90.63	.01636	120.86	1006.4	1127.3	939.3	.2198	1.6427	1.8625
5	162.24	73.52	.01640	130.13	1001.0	1131.1	933.0	.2347	1.6094	1.8441
6	170.06	61.98	.01645	137.96	996.2	1134.2	927.5	.2472	1.5820	1.8292
7	176.85	53.64	.01649	144.76	992.1	1136.9	922.7	.2581	1.5586	1.8167
8	182.86	47.34	.01653	150.79	988.5	1139.3	918.4	.2674	1.5383	1.8057
9	188.28	42.40	.01656	156.22	985.2	1141.4	914.6	.2759	1.5203	1.7962
10	193.21	38.42	.01659	161.17	982.1	1143.3	911.1	.2835	1.5041	1.7876
11	197.75	35.14	.01662	165.73	979.3	1145.0	907.8	.2903	1.4897	1.7800
12	201.96	32.40	.01665	169.96	976.6	1146.6	904.8	.2967	1.4763	1.7730
13	205.88	30.06	.01667	173.91	974.2	1148.1	901.9	.3027	1.4638	1.7665
14	209.56	28.04	.01670	177.61	971.9	1149.5	899.3	.3083	1.4522	1.7605
14.696	212.00	26.80	.01672	180.07	970.3	1150.4	897.5	.3120	1.4446	1.7566
15	213.03	26.29	.01672	181.11	969.7	1150.8	896.7	.3135	1.4415	1.7549
16	216.32	24.75	.01674	184.42	967.6	1152.0	894.3	.3184	1.4313	1.7497
17	219.44	23.39	.01677	187.56	965.5	1153.1	892.0	.3231	1.4218	1.7449
18	222.41	22.17	.01679	190.56	963.6	1154.2	889.9	.3275	1.4128	1.7403
19	225.24	21.08	.01681	193.42	961.9	1155.3	887.8	.3317	1.4043	1.7360
20	227.96	20.089	.01683	196.16	960.1	1156.3	885.8	.3356	1.3962	1.7319
21	230.57	19.192	.01685	198.79	958.4	1157.2	883.9	.3395	1.3885	1.7280
22	233.07	18.375	.01687	201.33	956.8	1158.1	882.0	.3431	1.3811	1.7242
23	235.49	17.627	.01689	203.78	955.2	1159.0	880.2	.3466	1.3740	1.7206
24	237.82	16.938	.01691	206.14	953.7	1159.8	878.5	.3500	1.3672	1.7172
25	240.07	16.303	.01692	208.42	952.1	1160.6	876.8	.3533	1.3606	1.7139
26	242.25	15.715	.01694	210.62	950.7	1161.3	875.2	.3564	1.3544	1.7108
27	244.36	15.170	.01696	212.75	949.3	1162.0	873.6	.3594	1.3484	1.7078
28	246.41	14.663	.01698	214.83	947.9	1162.7	872.1	.3623	1.3425	1.7048
29	248.40	14.189	.01699	216.86	946.5	1163.4	870.5	.3652	1.3368	1.7020
30	250.33	13.746	.01701	218.82	945.3	1164.1	869.1	.3680	1.3313	1.6993
31	252.22	13.330	.01702	220.73	944.0	1164.7	867.7	.3707	1.3260	1.6967
32	254.05	12.940	.01704	222.59	942.8	1165.4	866.3	.3733	1.3209	1.6941
33	255.84	12.572	.01705	224.41	941.6	1166.0	864.9	.3758	1.3159	1.6917
34	257.58	12.226	.01707	226.18	940.3	1166.5	863.5	.3783	1.3110	1.6893
35	259.28	11.898	.01708	227.91	939.2	1167.1	862.3	.3807	1.3063	1.6870
36	260.95	11.588	.01709	229.60	938.0	1167.6	861.0	.3831	1.3017	1.6848
37	262.57	11.294	.01711	231.26	936.9	1168.2	859.8	.3854	1.2972	1.6826
38	264.16	11.015	.01712	232.89	935.8	1168.7	858.5	.3876	1.2929	1.6805
39	265.72	10.750	.01714	234.48	934.7	1169.2	857.2	.3898	1.2886	1.6784
40	267.25	10.498	.01715	236.03	933.7	1169.7	856.1	.3919	1.2844	1.6763
41	268.74	10.258	.01716	237.55	932.6	1170.2	855.0	.3940	1.2803	1.6743
42	270.21	10.029	.01717	239.04	931.6	1170.7	853.8	.3960	1.2764	1.6724
43	271.64	9.810	.01719	240.51	930.6	1171.1	852.7	.3980	1.2726	1.6706
44	273.05	9.601	.01720	241.95	929.6	1171.6	851.6	.4000	1.2687	1.6687

---

[1] Abstracted from "Thermodynamic Properties of Steam," by J. H. Keenan and F. G. Keyes, 1936 ed., by permission of the authors and publishers, John Wiley & Sons, Inc.

Table 1. Properties of Saturated Steam—(*Continued*)

Abs. press., lb. per sq. in.	Temp., °F.	Sp. vol., cu. ft. per lb.		Heat of the liquid, B.t.u.	Latent heat of evap., B.t.u.	Total heat of steam, B.t.u.	Internal energy of evap., B.t.u.	Entropy		
		Steam	Water					Water	Increase during evap.	Steam
p	t	v_g	v_f	h_f	h_{fg}	h_g	u_{fg}	s_f	h_{fg}/T	s_g
45	274.44	9.401	0.01721	243.36	928.6	1172.0	850.5	0.4019	1.2650	1.6669
46	275.80	9.209	.01722	244.75	927.7	1172.4	849.5	.4038	1.2613	1.6652
47	277.13	9.025	.01723	246.12	926.7	1172.9	848.4	.4057	1.2577	1.6634
48	278.45	8.848	.01725	247.47	925.8	1173.3	847.4	.4075	1.2542	1.6617
49	279.74	8.678	.01726	248.79	924.9	1173.7	846.4	.4093	1.2508	1.6601
50	281.01	8.515	.01727	250.09	924.0	1174.1	845.4	.4110	1.2474	1.6585
51	282.26	8.359	.01728	251.37	923.0	1174.4	844.3	.4127	1.2442	1.6569
52	283.49	8.208	.01729	252.63	922.2	1174.8	843.3	.4144	1.2409	1.6553
53	284.70	8.062	.01730	253.87	921.3	1175.2	842.4	.4161	1.2377	1.6538
54	285.90	7.922	.01731	255.09	920.5	1175.6	841.5	.4177	1.2346	1.6523
55	287.07	7.787	.01732	256.30	919.6	1175.9	840.6	.4193	1.2316	1.6509
56	288.23	7.656	.01733	257.50	918.8	1176.3	839.7	.4209	1.2285	1.6494
57	289.37	7.529	.01734	258.67	917.9	1176.6	838.7	.4225	1.2255	1.6480
58	290.50	7.407	.01736	259.82	917.1	1176.9	837.8	.4240	1.2226	1.6466
59	291.61	7.289	.01737	260.96	916.3	1177.3	836.9	.4255	1.2197	1.6452
60	292.71	7.175	.01738	262.09	915.5	1177.6	836.0	.4270	1.2168	1.6438
61	293.79	7.064	.01739	263.20	914.7	1177.9	835.2	.4285	1.2140	1.6425
62	294.85	6.957	.01740	264.30	913.9	1178.2	834.3	.4300	1.2112	1.6412
63	295.90	6.853	.01741	265.38	913.1	1178.5	833.4	.4314	1.2085	1.6399
64	296.94	6.752	.01742	266.45	912.3	1178.8	832.6	.4328	1.2059	1.6387
65	297.97	6.655	.01743	267.50	911.6	1179.1	831.8	.4342	1.2032	1.6374
66	298.99	6.560	.01744	268.55	910.8	1179.4	831.0	.4356	1.2006	1.6362
67	299.99	6.468	.01745	269.58	910.1	1179.7	830.2	.4369	1.1981	1.6350
68	300.98	6.378	.01746	270.60	909.4	1180.0	829.4	.4383	1.1955	1.6338
69	301.96	6.291	.01747	271.61	908.7	1180.3	828.6	.4396	1.1930	1.6326
70	302.92	6.206	.01748	272.61	907.9	1180.6	827.8	.4409	1.1906	1.6315
71	303.88	6.124	.01749	273.60	907.2	1180.8	827.0	.4422	1.1881	1.6303
72	304.83	6.044	.01750	274.57	906.5	1181.1	826.3	.4435	1.1857	1.6292
73	305.76	5.966	.01751	275.54	905.8	1181.3	825.5	.4447	1.1834	1.6281
74	306.68	5.890	.01752	276.49	905.1	1181.6	824.7	.4460	1.1810	1.6270
75	307.60	5.816	.01753	277.43	904.5	1181.9	824.0	.4472	1.1787	1.6259
76	308.50	5.743	.01754	278.37	903.7	1182.1	823.3	.4484	1.1764	1.6248
77	309.40	5.673	.01754	279.30	903.1	1182.4	822.5	.4496	1.1742	1.6238
78	310.29	5.604	.01755	280.21	902.4	1182.6	821.7	.4508	1.1720	1.6228
79	311.16	5.537	.01756	281.12	901.7	1182.8	821.0	.4520	1.1698	1.6217
80	312.03	5.472	.01757	282.02	901.1	1183.1	820.3	.4531	1.1676	1.6207
81	312.89	5.408	.01758	282.91	900.4	1183.3	819.6	.4543	1.1654	1.6197
82	313.74	5.346	.01759	283.79	899.7	1183.5	818.9	.4554	1.1633	1.6187
83	314.59	5.285	.01760	284.66	899.1	1183.8	818.2	.4565	1.1612	1.6177
84	315.42	5.226	.01761	285.53	898.5	1184.0	817.5	.4576	1.1592	1.6168
85	316.25	5.168	.01761	286.39	897.8	1184.2	816.8	.4587	1.1571	1.6158
86	317.07	5.111	.01762	287.24	897.2	1184.4	816.1	.4598	1.1551	1.6149
87	317.88	5.055	.01763	288.08	896.5	1184.6	815.4	.4609	1.1530	1.6139
88	318.68	5.001	.01764	288.91	895.9	1184.8	814.8	.4620	1.1510	1.6130
89	319.48	4.948	.01765	289.74	895.3	1185.1	814.1	.4630	1.1491	1.6121
90	320.27	4.896	.01766	290.56	894.7	1185.3	813.4	.4641	1.1471	1.6112
91	321.06	4.845	.01767	291.38	894.1	1185.5	812.8	.4651	1.1452	1.6103
92	321.83	4.796	.01768	292.18	893.5	1185.7	812.2	.4661	1.1433	1.6094
93	322.60	4.747	.01768	292.98	892.9	1185.9	811.5	.4672	1.1413	1.6085
94	323.36	4.699	.01769	293.78	892.3	1186.1	810.9	.4682	1.1394	1.6076

Table 1. Properties of Saturated Steam—(*Continued*)

Abs. press., lb. per sq. in.	Temp., °F.	Sp. vol., cu. ft. per lb.		Heat of the liquid, B.t.u.	Latent heat of evap., B.t.u.	Total heat of steam, B.t.u.	Internal energy of evap., B.t.u.	Entropy		
		Steam	Water					Water	Increase during evap.	Steam
p	t	v_g	v_f	h_f	h_{fg}	h_g	u_{fg}	s_f	h_{fg}/T	s_g
95	324.12	4.652	0.01770	294.56	891.7	1186.2	810.2	0.4692	1.1376	1.6068
96	324.87	4.606	.01771	295.34	891.1	1186.4	809.6	.4702	1.1358	1.6060
97	325.61	4.561	.01772	296.12	890.5	1186.6	808.9	.4711	1.1340	1.6051
98	326.35	4.517	.01772	296.89	889.9	1186.8	808.3	.4721	1.1322	1.6043
99	327.08	4.474	.01773	297.65	889.4	1187.0	807.7	.4731	1.1304	1.6035
100	327.81	4.432	.01774	298.40	888.8	1187.2	807.1	.4740	1.1286	1.6026
102	329.25	4.350	.01775	299.90	887.6	1187.5	805.9	.4759	1.1251	1.6010
104	330.66	4.271	.01777	301.37	886.5	1187.9	804.7	.4778	1.1216	1.5994
106	332.05	4.194	.01778	302.82	885.4	1188.2	803.5	.4796	1.1182	1.5978
108	333.42	4.120	.01780	304.26	884.3	1188.6	802.4	.4814	1.1149	1.5963
110	334.77	4.049	.01782	305.66	883.2	1188.9	801.2	.4832	1.1117	1.5948
112	336.11	3.981	.01783	307.06	882.1	1189.2	800.0	.4849	1.1085	1.5934
114	337.42	3.914	.01784	308.43	881.1	1189.5	798.9	.4866	1.1053	1.5919
116	338.72	3.850	.01786	309.79	880.0	1189.8	797.8	.4883	1.1022	1.5905
118	339.99	3.788	.01787	311.12	879.0	1190.1	796.7	.4900	1.0992	1.5891
120	341.25	3.728	.01789	312.44	877.9	1190.4	795.6	.4916	1.0962	1.5878
122	342.50	3.670	.01791	313.75	876.9	1190.7	794.5	.4932	1.0933	1.5865
124	343.72	3.614	.01792	315.04	875.9	1190.9	793.4	.4948	1.0903	1.5851
126	344.94	3.560	.01793	316.31	874.9	1191.2	792.3	.4964	1.0874	1.5838
128	346.13	3.507	.01794	317.57	873.9	1191.5	791.3	.4980	1.0845	1.5825
130	347.32	3.455	.01796	318.81	872.9	1191.7	790.2	.4995	1.0817	1.5812
132	348.48	3.405	.01797	320.04	872.0	1192.0	789.2	.5010	1.0790	1.5800
134	349.64	3.357	.01799	321.25	871.0	1192.2	788.2	.5025	1.0762	1.5787
136	350.78	3.310	.01800	322.45	870.1	1192.5	787.2	.5040	1.0735	1.5775
138	351.91	3.264	.01801	323.64	869.1	1192.7	786.2	.5054	1.0709	1.5763
140	353.02	3.220	.01802	324.82	868.2	1193.0	785.2	.5069	1.0682	1.5751
142	354.12	3.177	.01804	325.98	867.2	1193.2	784.3	.5083	1.0657	1.5740
144	355.21	3.134	.01805	327.13	866.3	1193.4	783.3	.5097	1.0631	1.5728
146	356.29	3.094	.01806	328.27	865.3	1193.6	782.3	.5111	1.0605	1.5716
148	357.36	3.054	.01808	329.39	864.5	1193.9	781.4	.5124	1.0580	1.5705
150	358.42	3.015	.01809	330.51	863.6	1194.1	780.5	.5138	1.0556	1.5694
152	359.46	2.977	.01810	331.61	862.7	1194.3	779.5	.5151	1.0532	1.5683
154	360.49	2.940	.01812	332.70	861.8	1194.5	778.5	.5165	1.0507	1.5672
156	361.52	2.904	.01813	333.79	860.9	1194.7	777.6	.5178	1.0483	1.5661
158	362.53	2.869	.01814	334.86	860.0	1194.9	776.8	.5191	1.0459	1.5650
160	363.53	2.834	.01815	335.93	859.2	1195.1	775.8	.5204	1.0436	1.5640
162	364.53	2.801	.01817	336.98	858.3	1195.3	775.0	.5216	1.0414	1.5630
164	365.51	2.768	.01818	338.02	857.5	1195.5	774.1	.5229	1.0391	1.5620
166	366.48	2.736	.01819	339.05	856.6	1195.7	773.2	.5241	1.0369	1.5610
168	367.45	2.705	.01820	340.07	855.7	1195.8	772.3	.5254	1.0346	1.5600
170	368.41	2.675	.01822	341.09	854.9	1196.0	771.4	.5266	1.0324	1.5590
172	369.35	2.645	.01823	342.10	854.1	1196.2	770.5	.5278	1.0302	1.5580
174	370.29	2.616	.01824	343.10	853.3	1196.4	769.7	.5290	1.0280	1.5570
176	371.22	2.587	.01825	344.09	852.4	1196.5	768.8	.5302	1.0259	1.5561
178	372.14	2.559	.01826	345.06	851.6	1196.7	767.9	.5313	1.0238	1.5551
180	373.06	2.532	.01827	346.03	850.8	1196.9	767.1	.5325	1.0217	1.5542
182	373.96	2.505	.01829	347.00	850.0	1197.0	766.2	.5336	1.0196	1.5532
184	374.86	2.479	.01830	347.96	849.2	1197.2	765.4	.5348	1.0175	1.5523
186	375.75	2.454	.01831	348.92	848.4	1197.3	764.6	.5359	1.0155	1.5514
188	376.64	2.429	.01832	349.86	847.6	1197.5	763.8	.5370	1.0136	1.5506

Table 1. Properties of Saturated Steam—*(Concluded)*

Abs. press., lb. per sq. in.	Temp., °F.	Sp. vol., cu. ft. per lb.		Heat of the liquid, B.t.u.	Latent heat of evap., B.t.u.	Total heat of steam, B.t.u.	Internal energy of evap., B.t.u.	Entropy		
		Steam	Water					Water	Increase during evap.	Steam
p	t	v_g	v_f	h_f	h_{fg}	h_g	u_{fg}	s_f	h_{fg}/T	s_g
190	377.51	2.404	0.01833	350.79	846.8	1197.6	763.0	0.5381	1.0116	1.5497
192	378.38	2.380	.01834	351.72	846.1	1197.8	762.1	.5392	1.0096	1.5488
194	379.24	2.356	.01835	352.64	845.3	1197.9	761.3	.5403	1.0076	1.5479
196	380.10	2.333	.01836	353.55	844.5	1198.1	760.6	.5414	1.0056	1.5470
198	380.95	2.310	.01838	354.46	843.7	1198.2	759.8	.5425	1.0037	1.5462
200	381.79	2.288	.01839	355.36	843.0	1198.4	759.0	.5435	1.0018	1.5453
205	383.86	2.234	.01842	357.58	841.1	1198.7	757.1	.5461	0.9971	1.5432
210	385.90	2.183	.01844	359.77	839.2	1199.0	755.2	.5487	.9925	1.5412
215	387.89	2.134	.01847	361.91	837.4	1199.3	753.2	.5512	.9880	1.5392
220	389.86	2.087	.01850	364.02	835.6	1199.6	751.3	.5537	.9835	1.5372
225	391.79	2.0422	.01852	366.09	833.8	1199.9	749.5	.5561	.9792	1.5353
230	393.68	1.9992	.01854	368.13	832.0	1200.1	747.7	.5585	.9750	1.5334
235	395.54	1.9579	.01857	370.14	830.3	1200.4	745.9	.5608	.9708	1.5316
240	397.37	1.9183	.01860	372.12	828.5	1200.6	744.1	.5631	.9667	1.5298
245	399.18	1.8803	.01863	374.08	826.8	1200.9	742.4	.5653	.9627	1.5280
250	400.95	1.8438	.01865	376.00	825.1	1201.1	740.7	.5675	.9588	1.5263
260	404.42	1.7748	.01870	379.76	821.8	1201.5	737.3	.5719	.9510	1.5229
270	407.78	1.7107	.01875	383.42	818.5	1201.9	733.9	.5760	.9436	1.5196
280	411.05	1.6511	.01880	386.98	815.3	1202.3	730.7	.5801	.9363	1.5164
290	414.23	1.5954	.01885	390.46	812.1	1202.6	727.5	.5841	.9292	1.5133
300	417.33	1.5433	.01890	393.84	809.0	1202.8	724.3	.5879	.9225	1.5104
350	431.72	1.3260	.01913	409.69	794.2	1203.9	709.6	.6056	.8910	1.4966
400	444.59	1.1613	.0193	424.0	780.5	1204.5	695.9	.6214	.8630	1.4844
450	456.28	1.0320	.0195	437.2	767.4	1204.6	683.2	.6356	.8378	1.4734
500	467.01	0.9278	.0197	449.4	755.0	1204.4	671.0	.6487	.8147	1.4634
600	486.21	.7698	.0201	471.6	731.6	1203.2	648.3	.6720	.7734	1.4454
700	503.10	.6554	.0205	491.5	709.7	1201.2	627.5	.6925	.7371	1.4296
800	518.23	.5687	.0209	509.7	688.9	1198.6	607.8	.7108	.7045	1.4153
900	531.98	.5006	.0212	526.6	668.8	1195.4	589.0	.7275	.6744	1.4020
1000	544.61	.4456	.0216	542.4	649.4	1191.8	571.0	.7430	.6467	1.3897
1100	556.31	.4001	.0220	557.4	630.4	1187.8	553.5	.7575	.6205	1.3780
1200	567.22	.3619	.0223	571.7	611.7	1183.4	536.3	.7711	.5956	1.3667
1300	577.46	.3293	.0227	585.4	593.2	1178.6	519.4	.7840	.5719	1.3559
1400	587.10	.3012	.0231	598.7	574.7	1173.4	502.7	.7963	.5491	1.3454
1500	596.23	.2765	.0235	611.6	556.3	1167.9	486.1	.8082	.5269	1.3351
1600	604.90	.2548	.0239	624.1	538.0	1162.1	469.7	.8196	.5053	1.3249
1700	613.15	.2354	.0243	636.3	519.6	1155.9	453.1	.8306	.4843	1.3149
1800	621.03	.2179	.0247	648.3	501.1	1149.4	436.7	.8412	.4637	1.3049
1900	628.58	.2021	.0252	660.1	482.4	1142.4	420.2	.8516	.4433	1.2949
2000	635.82	.1878	.0257	671.7	463.4	1135.1	403.4	.8619	.4230	1.2849
2200	649.46	.1625	.0268	694.8	424.4	1119.2	369.2	.8820	.3826	1.2646
2400	662.12	.1407	.0280	718.4	382.7	1101.1	332.6	.9023	.3411	1.2434
2600	673.94	.1213	.0295	743.0	337.2	1080.2	293.1	.9232	.2973	1.2205
2800	684.99	.1035	.0315	770.1	284.7	1054.8	247.4	.9459	.2487	1.1946
3000	695.36	.0858	.0346	802.5	217.8	1020.3	189.3	.9731	.1885	1.1615
3200	705.11	.0580	.0444	872.4	62.0	934.4	52.4	1.0320	.0532	1.0852
3206.2	705.40	.0503	.0503	902.7	0	902.7	0.0	1.0580	0	1.0580

Table 2. Properties of Superheated Steam[1]

v = specific volume, cu. ft. per lb.; h = total heat, B.t.u. per lb.; s = entropy

Pressure, lb. per sq. in. abs. (saturation temp., °F.)		Temperature of steam, °F.											
		400	500	540	560	600	650	700	750		800	850	900
20	v	25.43	28.46	......	......	31.47	32.97	34.47	....		37.46	......	40.45
(227.96)	h	1239.2	1286.6	......	......	1334.4	1358.6	382.9	....		1432.1	......	1482.1
	s	1.8396	1.8918	......	......	1.9392	1.9614	1.9829	....		2.0235	......	2.0618
40	v	12.628	14.168	14.778	15.082	15.688	16.444	17.198	17.951		18.702	......	20.20
(267.25)	h	1236.5	1284.8	1304.1	1313.8	1333.1	1357.4	1381.9	1406.5		1431.3	......	1481.4
	s	1.7608	1.8140	1.8337	1.8432	1.8619	1.8843	1.9058	1.9266		1.9467	......	1.9850
50	v	10.065	11.309	11.800	12.044	12.532	13.139	13.744	14.348		14.950	......	16.152
(281.01)	h	1235.1	1283.9	1303.3	1313.0	1332.5	1356.9	1381.4	1406.1		1430.9	......	1481.1
	s	1.7349	1.7887	1.8085	1.8181	1.8368	1.8593	1.8809	1.9017		1.9219	......	1.9602
60	v	8.357	9.403	9.814	10.019	10.427	10.935	11.441	11.945		12.449	12.951	13.452
(292.71)	h	1233.6	1283.0	1302.5	1312.3	1331.8	1356.3	1380.9	1405.6		1430.5	1455.6	1480.8
	s	1.7135	1.7678	1.7878	1.7975	1.8162	1.8388	1.8605	1.8814		1.9015	1.9210	1.9400
80	v	6.220	7.020	7.332	7.488	7.797	8.180	8.562	8.942		9.322	9.700	10.079
(312.03)	h	1230.7	1281.1	1300.9	1310.7	1330.5	1355.1	1379.9	1404.7		1429.7	1454.8	1480.1
	s	1.6791	1.7346	1.7549	1.7646	1.7836	1.8063	1.8281	1.8491		1.8694	1.8889	1.9079
100	v	4.937	5.589	5.843	5.968	6.218	6.527	6.835	7.141		7.446	7.749	8.052
(327.81)	h	1227.6	1279.1	1299.2	1309.2	1329.1	1354.0	1378.9	1403.8		1428.9	1454.1	1479.5
	s	1.6518	1.7085	1.7290	1.7389	1.7581	1.7810	1.8029	1.8240		1.8443	1.8639	1.8829
120	v	4.081	4.636	4.849	4.955	5.165	......	5.683	5.940		6.195	6.449	6.702
(341.25)	h	1224.4	1277.2	1297.5	1307.6	1327.7	......	1377.8	1402.9		1428.1	1453.4	1478.8
	s	1.6287	1.6869	1.7077	1.7177	1.7370	......	1.7822	1.8033		1.8237	1.8434	1.8625
140	v	3.468	3.954	4.140	4.232	4.413	......	4.861	5.082		5.301	5.520	5.738
(353.02)	h	1221.1	1275.2	1295.8	1306.0	1326.4	......	1376.8	1402.0		1427.3	1452.6	1478.2
	s	1.6087	1.6683	1.6894	1.6995	1.7190	......	1.7645	1.7858		1.8063	1.8260	1.8451
160	v	3.008	3.443	3.607	3.689	3.849	......	4.244	4.438		4.631	4.823	5.015
(363.53)	h	1217.6	1273.1	1294.1	1304.4	1325.0	......	1375.7	1401.1		1426.4	1451.9	1477.5
	s	1.5908	1.6519	1.6733	1.6836	1.7033	......	1.7491	1.7705		1.7911	1.8109	1.8301
180	v	2.649	3.044	3.193	3.266	3.411	......	3.764	3.937		4.110	4.281	4.452
(373.06)	h	1214.0	1271.0	1292.3	1302.8	1323.5	......	1374.7	1400.2		1425.6	1451.2	1476.8
	s	1.5745	1.6373	1.6590	1.6694	1.6894	......	1.7355	1.7570		1.7776	1.7975	1.8167
200	v	2.361	2.726	2.862	2.928	3.060	......	3.380	3.537		3.693	3.848	4.002
(381.79)	h	1210.3	1268.9	1290.5	1301.1	1322.1	......	1373.6	1399.2		1424.8	1450.4	1476.2
	s	1.5594	1.6240	1.6460	1.6566	1.6767	......	1.7232	1.7448		1.7655	1.7855	1.8048
									740	760			
220	v	2.125	2.465	2.590	2.652	2.772	......	3.066	3.181	3.238	3.352	3.493	3.634
(389.86)	h	1206.5	1266.7	1288.7	1299.5	1320.7	......	1372.6	1393.1	1403.4	1424.0	1449.7	1475.5
	s	1.5453	1.6117	1.6342	1.6448	1.6652	......	1.7120	1.7295	1.7380	1.7545	1.7746	1.7939
240	v	1.9276	2.247	2.364	2.421	2.533	......	2.804	2.910	2.963	3.068	3.198	3.327
(397.37)	h	1202.5	1264.5	1286.9	1297.8	1319.2	......	1371.5	1392.2	1402.5	1423.2	1449.0	1474.8
	s	1.5319	1.6003	1.6232	1.6339	1.6546	......	1.7017	1.7193	1.7278	1.7444	1.7645	1.7839

---

[1] Abstracted from "Thermodynamic Properties of Steam," by J. H. Keenan and F. G. Keyes, 1936 ed., by permission of the authors and publishers, John Wiley & Sons, Inc.

Table 2. Properties of Superheated Steam—(*Continued*)

Pressure, lb. per sq. in. abs. (saturation temp., °F.)		Temperature of steam, °F.											
		400	500	540	560	600	650	700	740	760	800	850	900
250 (400.95)	*v*	......	2.151	2.264	2.319	2.427	......	2.688	2.791	2.841	2.942	3.068	3.192
	h	......	1263.4	1285.9	1296.9	1318.5	......	1371.0	1391.7	1402.1	1422.7	1448.6	1474.5
	s	......	1.5949	1.6179	1.6288	1.6495	......	1.6969	1.7145	1.7230	1.7397	1.7598	1.7793
260 (404.42)	*v*	......	2.063	2.172	2.226	2.330	......	2.582	2.681	2.729	2.827	2.947	3.067
	h	......	1262.3	1285.0	1296.0	1317.7	......	1370.4	1391.2	1401.6	1422.3	1448.2	1474.2
	s	......	1.5897	1.6129	1.6238	1.6447	......	1.6922	1.7099	1.7185	1.7352	1.7553	1.7748
280 (411.05)	*v*	......	1.9047	2.008	2.058	2.156	......	2.392	2.484	2.530	2.621	2.733	2.845
	h	......	1260.0	1283.1	1294.3	1316.2	......	1369.4	1390.3	1400.7	1421.5	1447.5	1473.5
	s	......	1.5796	1.6032	1.6143	1.6354	......	1.6834	1.7011	1.7097	1.7265	1.7467	1.7662
300 (417.33)	*v*	......	1.7675	1.8654	1.9128	2.005	......	2.227	2.314	2.357	2.442	2.547	2.652
	h	......	1257.6	1281.1	1292.5	1314.7	......	1368.3	1389.3	1420.6	1420.6	1446.7	1472.8
	s	......	1.5701	1.5941	1.6054	1.6268	......	1.6751	1.6929	1.7184	1.7184	1.7387	1.7582
350 (431.72)	*v*	......	1.4923	......	......	1.7036	......	1.8980	1.9732	2.010	2.084	2.176	2.266
	h	......	1251.5	......	......	1310.9	......	1365.5	1386.8	1397.4	1418.5	1444.8	1471.1
	s	......	1.5481	......	......	1.6070	......	1.6563	1.6744	1.6831	1.7002	1.7206	1.7403
400 (444.59)	*v*	......	1.2851	......	......	1.4770	......	1.6508	1.7177	1.9372	1.8161	1.8968	1.9767
	h	......	1245.1	......	......	1306.9	......	1362.7	1384.3	1386.8	1416.4	1442.9	1469.4
	s	......	1.5281	......	......	1.5894	......	1.6398	1.6581	1.6744	1.6842	1.7049	1.7247
450 (456.28)	*v*	......	1.1231	......	......	1.3005	......	1.4584	1.5188	1.5486	1.6074	1.6800	1.7516
	h	......	1238.4	......	......	1302.8	......	1359.9	1381.8	1392.7	1414.3	1441.0	1467.7
	s	......	1.5095	......	......	1.5735	......	1.6250	1.6436	1.6525	1.6699	1.6908	1.7108
500 (467.01)	*v*	......	0.9927	......	......	1.1591	......	1.3044	1.3596	1.3868	1.4405	1.5065	1.5715
	h	......	1231.3	......	......	1298.6	......	1357.0	1379.3	1390.3	1412.1	1439.1	1466.0
	s	......	1.4919	......	......	1.5588	......	1.6115	1.6304	1.6395	1.6571	1.6781	1.6982
550 (476.94)	*v*	......	0.8852	......	......	1.0431	......	1.1783	1.2293	1.2544	1.3038	1.3644	1.4241
	h	......	1223.7	......	......	1294.3	......	1354.0	1376.7	1387.8	1409.9	1437.2	1464.3
	s	......	1.4751	......	......	1.5451	......	1.5991	1.6182	1.6274	1.6452	1.6665	1.6868
600 (486.21)	*v*	......	0.7947	......	......	0.9463	......	1.0732	1.1207	1.1440	1.1899	1.2460	1.3013
	h	......	1215.7	......	......	1289.9	......	1351.1	1374.0	1358.3	1407.7	1435.2	1462.5
	s	......	1.4586	......	......	1.5323	......	1.5875	1.6070	1.6163	1.6343	1.6558	1.6762
700 (503.10)	*v*	......	......	......	......	0.7934	......	0.9077	0.9498	0.9704	1.0108	1.0600	1.1082
	h	......	......	......	......	1280.6	......	1345.0	1368.7	1380.3	1403.2	1431.3	1459.0
	s	......	......	......	......	1.5084	......	1.5665	1.5866	1.5962	1.6147	1.6366	1.6573
800 (518.23)	*v*	......	......	......	......	0.6779	......	0.7833	0.8215	0.8400	0.8763	......	0.9633
	h	......	......	......	......	1270.7	......	1338.6	1363.2	1375.2	1398.6	......	1455.4
	s	......	......	......	......	1.4863	......	1.5476	1.5684	1.5783	1.5972	......	1.6407
900 (531.98)	*v*	......	......	......	......	0.5873	......	0.6863	0.7215	0.7385	0.7716	......	0.8506
	h	......	......	......	......	1260.1	......	1332.1	1357.5	1369.9	1393.9	......	1451.8
	s	......	......	......	......	1.4653	......	1.5303	1.5519	1.5620	1.5814	......	1.6257
1000 (544.61)	*v*	......	......	......	......	0.5140	......	0.6084	0.6413	0.6571	0.6878	......	0.7604
	h	......	......	......	......	1248.8	......	1325.3	1351.7	1364.4	1389.2	......	1448.2
	s	......	......	......	......	1.4450	......	1.5141	1.5365	1.5470	1.5670	......	1.6121

Table 2. Properties of Superheated Steam—(*Concluded*)

Pressure, lb. per sq. in. abs. (saturation temp., °F.)		Temperature of steam, °F.											
		400	500	540	560	600	650	700	740	760	800	850	900
1100 (556.31)	*v*	......	......	......	......	0.4532	......	0.5445	0.5469	0.5613	0.6191	......	0.6866
	h	......	......	......	......	1236.7	......	1318.3	1342.7	1356.1	1384.3	......	1444.5
	s	......	......	......	......	1.4251	......	1.4989	1.5153	1.5263	1.5535	......	1.5995
1200 (567.22)	*v*	......	......	......	......	0.4016	......	0.4909	0.5206	0.5347	0.5617	......	0.6250
	h	......	......	......	......	1223.5	......	1311.0	1339.6	1353.2	1379.3	......	1440.7
	s	......	......	......	......	1.4052	......	1.4843	1.5086	1.5918	1.5409	......	1.5879
1400 (587.10)	*v*	......	......	......	......	0.3174	......	0.4062	0.4338	0.4468	0.4714	......	0.5281
	h	......	......	......	......	1193.0	......	1295.5	1326.7	1341.3	1369.1	......	1433.1
	s	......	......	......	......	1.3639	......	1.4567	1.4832	1.4953	1.5177	......	1.5666
1600 (604.90)	*v*	......	......	......	......	......	......	0.3417	0.3682	0.3804	0.4034	......	0.4553
	h	......	......	......	......	......	......	1278.7	1313.0	1328.8	1358.4	......	1425.3
	s	......	......	......	......	......	......	1.4303	1.4595	1.4725	1.4964	......	1.5476
1800 (621.03)	*v*	......	......	......	......	......	......	0.2907	0.3116	0.3284	0.3502	......	0.3986
	h	......	......	......	......	......	......	1260.3	1298.4	1315.5	1347.2	......	1417.4
	s	......	......	......	......	......	......	1.4044	1.4367	1.4509	1.4765	......	1.5301

pressures. In the fire-tube boiler, gas passes inside the tubes and the water surrounds them. The horizontal-return-tubular (h.r.t.) boiler is the most common of this type. The Scotch marine and the locomotive types fall in this class, and there are also vertical-fire-tube boilers.

Water-tube boilers include the following types: straight inclined tubes with longitudinal or cross drum; bent tubes with single or multiple steam drums; vertical and semivertical tubes with vertical or cross drum; various combinations of the foregoing; and boilers with forced circulation. Water-tube boilers are adapted to higher pressures and are quicker steaming. There is little difference in efficiency attainable as between types, other things being equal. Fire-tube boilers may be supported by the side walls of the setting by means of lugs riveted to the shell and resting on rollers, but the preferable type of suspension is from an overhead beam by means of buckstays. Water-tube boilers are usually suspended from overhead steel work and columns.

Water Walls. The furnaces of many boilers are protected by water walls consisting of closely spaced tubes supplied with water from the boiler circulation. The steam generated in them is delivered to the boiler steam drum. One or all walls of the furnace may be protected with water walls depending on the firing method.

The purpose of water walls is to permit higher rates of heat release without excessive furnace maintenance. They also lower the temperature of the gas leaving the furnace and so permit higher ratings before trouble occurs from fly ash slagging on the boiler tubes.

Water walls form an extremely active part of the boiler heating surface, often evaporating 100 lb. of steam per square foot per hour. There are no standards for tube spacing, which may vary from 3.5-in. centers with 3¼-in.

tubes to wide spacing. Arrangement of the tubes vary with different types of boilers and firing methods.

The tubes forming the walls may be bare (plain or with fins), or protected with refractory or cast-iron blocks. In another arrangement studs are spot-welded to the tubes on the fire side and the tubes and studs covered with plastic chrome.

Boiler Rating. Boilers were formerly rated on a horsepower basis, a *boiler horsepower* being defined as equivalent to the evaporation of 34.5 lb. of water per hour from and at 212°F. On this basis, with the existing practice of operating boilers, 10 sq. ft. heating surface was considered equivalent to 1 boiler h.p. The practice in later years of employing higher evaporative rates, and the inclusion of furnace water walls, economizers, and air preheaters in the boiler unit, have rendered the term boiler horsepower meaningless. Boilers are now commonly rated on the thousands of pounds of steam they are designed to produce per hour. A more exact unit of evaporation has been defined by the "A.S.M.E. Power Test Codes" in terms of the total heat transferred through the heating surfaces per hour. This is taken as 1000 B.t.u. transferred per hour and is called the *kilo B.t.u.* (kb.) or, more conveniently, for large boilers, 1,000,000 B.t.u. which is called the *mega B.t.u.* (mb.).

Some of the largest single central-station boilers have evaporated 1,250,000 lb. steam per hour. These boilers operate at 400 lb. per sq. in. pressure. The largest boilers installed in an industrial plant to date (1932) are in the Rouge plant of the Ford Motor Co. at Detroit. These are each rated at 900,000 lb. steam per hour but have generated over 1,000,000 lb. per hr. The largest high-pressure boiler is at the Logan station of the Appalachian Electric Power Co. and is rated at 100,000 lb. steam per hour at 1425 lb. per sq. in. pressure and 925°F.

The trend in both central-station and industrial-plant practice is toward fewer and larger boilers. The number of units and their capacities are governed by load conditions. Inasmuch as boiler efficiency decreases when forcing considerably above rating, it is well to plan the number and size such that they will be run for the greater part of the time near the point of maximum efficiency. It is usually more economical to handle large peak loads of short duration by forcing boiler units above their most efficient rate than to incur the fixed charges on additional boilers. As a rule industrial-plant boilers are not forced to the extent that central-station boilers are; 250 per cent is fair practice.

Boiler Efficiency. This is the ratio of the heat absorbed by the boiler per pound of fuel fired to the heat contained in 1 lb. of fuel and is usually applied to the unit as a whole. The heat absorbed by the boiler is equal to the heat in the steam delivered minus the heat in the feed water supplied. If the efficiency of the boiler alone is desired, this is the ratio of the heat absorbed by the boiler per pound of fuel fired to the heat actually developed in the furnace per pound of fuel. Full instructions for conducting a boiler test and computation of results are to be found in the "A.S.M.E. Power Test Codes."

Boilers and furnaces are now designed so that the efficiency curve is relatively flat over a considerable range in output.

Efficiencies of 90 per cent for a combined boiler unit, including economizer and air preheater, have been attained in some of the larger central-station installations, and sustained efficiencies of 85 to 88 per cent for monthly periods are common. For industrial-plant boilers, sustained efficiencies of about 80 per cent are considered excellent performances owing to the use of smaller units and less favorable load conditions. When burning natural gas, the efficiency will be from 2 to 4 per cent lower than that with coal, owing to the hydrogen of the gas forming water vapor.

Waste-heat Boilers. These are not to be confused with direct-fired boilers burning waste fuels such as hog fuel, sawdust, and refinery wastes, all of which, with certain modifications in furnace design, follow the practice in coal or oil-fired boilers. Waste-heat boilers, as here discussed, refer to those utilizing waste gas from industrial processes such as steel mills, cement mills, glass works, and metal refineries.

In contrast to the fuel-fired boiler the waste-heat boiler utilizes the heat of convection only, there being no radiant heat. Moreover, the entrance-gas temperatures are lower. Because of these facts the waste-heat boiler must be designed to handle large volumes of relatively low-temperature gas and the rate of heat transfer through the tubes must be much greater than in the direct-fired type. This requires higher gas velocities and greater draft loss. Induced draft is usually employed and economizers are common. The exit-gas temperatures ordinarily are reduced to 400° or 500°F. A waste-heat boiler is seldom operated above rating. The illustration (Fig. 6) shows a typical waste-heat boiler for a cement mill. Owing to the fact that the gas is likely to contain much dust, the boiler must be arranged for easy cleaning.

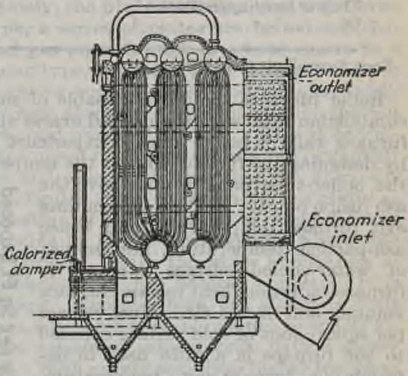


FIG. 6.—Vertical-type waste-heat boiler.

Waste-gas temperatures usually range from 1000° to 2000°F. the lower part of the range applying to gas from steel mills, cement kilns, and glass works and the higher temperatures to gas from metal refining. In cement mills the gas temperatures are higher with the dry process than with the wet process of manufacture. In steel mills the waste heat available for power generation averages about 200 kw.-hr. per ton of ingots or about two-thirds of the total power required. In cement mills, about 24 kw.-hr. are producible from each barrel of clinker burned per day.

Standards for the construction of stationary steam boilers are defined by the "A.S.M.E. Boiler Code." At the end of 1940, this code had been officially adopted by the states of New York, New Jersey, Pennsylvania, Delaware, Maryland, Michigan, Ohio, Indiana, Wisconsin, Minnesota, Missouri, Washington, Oregon, California, Utah, Oklahoma, Arkansas, Rhode Island, and the District of Columbia; also Hawaii and the Canal Zone, Maine, Massachusetts, North Carolina, Texas, and West Virginia.

Boiler Furnaces. For low and moderate rates of firing, solid refractory furnaces will give satisfactory results. For higher rates of firing, air-cooled walls or water walls are used to decrease maintenance. Heat release per cubic foot of furnace volume may be used as a rough guide in determining proper furnace size. The following table shows the range in heat release values now in use with various methods of firing and fuels used.

Furnace Heat Release

Method of firing	Heat release, B.t.u. per cu. ft. per hr.	
	All-refractory furnace	Complete water-wall furnace
Pulverized coal, 15% CO_2 at furnace exit:		
Illinois coal	12,000–15,000	20,000–35,000
Eastern bituminous coal	20,000–22,000	25,000–35,000
Underfeed stoker	25,000–35,000	30,000–42,000
Chain or traveling grate stoker	15,000–25,000	30,000–45,000
Oil	20,000–30,000	30,000–50,000

From *Power*, September, 1935.

Boiler furnaces should be capable of sustaining the maximum continuous combustion rate, with the desired excess air, without excessively slagging the furnace walls or boiler tubes. Experience indicates this can be accomplished by designing the furnace so that the temperature of the ash particles entering the boiler-tube bank is not above the ash fusion point. Factors determining exit furnace temperature, and hence ash-particle temperature, are amount of water-cooled surface exposed to furnace radiation, heat release, furnace volume, and excess air. Heat release per square foot of cold surface exposed to the furnace is a ratio used in determining furnace-exit temperature. Curves, such as in Fig. 7, use this ratio together with excess air, as indicated by CO_2, to find exit-furnace temperature, and are very helpful in furnace design.

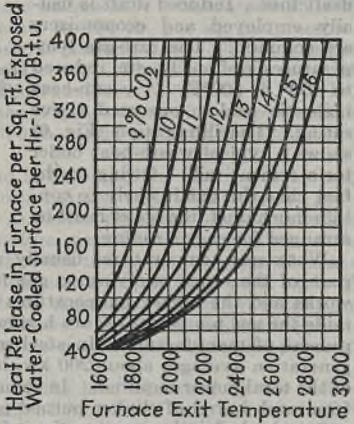


FIG. 7.—Curves for finding approximate temperature of gas leaving the boiler furnace. (*From Power, September*, 1935.)

Economizers

An economizer is essentially a feed-water heater placed just beyond the boiler to extract heat from the flue gases and thus reduce the stack loss. It consists of headers and groups of tubes, through which the boiler feed water is forced and over which the hot gas passes. The economizers may be integral with the boiler and placed inside the boiler setting, or they may be separate units, each with its own casing. Formerly they were invariably constructed of cast iron to resist corrosion better, but the advent of higher pressures has brought about the use of steel. They are of advantage in plants where an insufficient quantity of exhaust steam is available for feed-water heating as when the auxiliaries are motor driven; also where fuel costs are high so that saving with them will more than offset the additional investment charges. The economizer surface costs less than high-pressure boiler surface and hence is generally used to reduce the flue-gas temperature to a desirable value.

For every 10° to 11°F. increase in feed-water temperature obtained by decreasing the flue-gas temperature, there results an increase of about 1 per cent in boiler efficiency. The economizer surface employed varies from 0.2 to 2 sq. ft. per sq. ft. of boiler surface. In the larger units sufficient heat is absorbed to generate steam in the economizer. Generally, induced draft is necessary where economizers are employed, to permit high ratings without incurring excessive furnace pressure from the forced draft.

Air Preheaters

These are placed beyond the boilers to extract heat from the flue gases to preheat air for combustion. They may be used with or without economizers; in the former case they are placed after the economizer. Three different types of air preheaters are in use, namely, the **plate, tubular,** and **regenerative** types. In the first-named type, a series of plates are bolted together with proper spacing, so that air and gas pass through alternate spaces in a counterflow direction. In the tubular type, the gases pass through, and the air over, the tubes. In the regenerative type, the gas and air alternately pass over the same plates, so that the heat absorbed by the plates from the hot gases is given up to the air in the second part of the cycle. This is accomplished in one form of preheater by the use of rotating elements and in another type by a series of valves that open and close, first admitting gas, then air. Where a high-sulfur coal is used, difficulty from corrosion of the plates is sometimes encountered if the temperature of the flue gases is reduced below the dewpoint. This is often noticeable in stoker-fired boilers, where the boilers are banked for considerable periods. With the wider application of pulverized-coal firing, the use of preheated air has become popular. It is also used to some extent with stokers. Except where the stoker is built especially for high preheat, it is seldom advisable to employ much over 350°F., although over 500° has recently been employed satisfactorily with a stoker designed to handle such temperatures. With pulverizers these higher temperatures are frequently employed. Roughly, for every 35° to 40°F. drop in flue-gas temperature caused by heating air for combustion, there is a gain of 1 per cent in efficiency. In addition, preheated air improves combustion. As with economizers, air preheaters usually require induced draft fans, and also forced-draft fans are required.

Draft

The term **draft** as applied to boiler practice is the pressure difference which produces the flow of air and gas through the boiler. It is measured in inches of water.

Natural draft is that produced by the chimney or stack alone and is produced by the difference in weights of the column of hot gas within the stack and an equal column of outside air. An equation for determining the height and diameter of stack with natural draft, as given by Mingle (*Power*, Mar. 3, 1931), is as follows:

$$H = \frac{D_r}{2.96B_0\left(\frac{W_0}{T_0} - \frac{W_c}{T_c}\right) - \frac{0.184f.W_cB_0V^2}{T_cD}}$$

and

$$D = 0.228\sqrt{\frac{WT_c}{B_0W_cV}}$$

where H = height of stack above the grate, ft.
D = minimum internal diameter, ft.
D_r = required draft or sum of various losses of draft (including loss due to velocity) throughout the boiler setting exclusive of the stack itself, in. water.
B_0 = atmospheric pressure corresponding to altitude, in. Hg.
W_0 = density of atmosphere at °F. and sea level, lb. per cu. ft.
W_c = density of chimney gases at °F. and sea level, lb. per cu. ft.
T_0 = atmospheric temperature, °F. abs.
T_c = chimney gas temperature, °F. abs.
f = coefficient of friction
W = quantity of gas flowing, lb. per sec.
V = gas velocity, ft. per sec.

Table 3 by the same author is for determining economical sizes of natural draft chimneys for various quantities of gas handled and the required draft intensities at the breeching entrance based on assumed operating factors of 62°F. atmospheric temperature, 500°F. stack-gas temperature, sea-level atmospheric conditions, coefficient of friction of 0.016, and flue-gas density of 0.09 lb. per cu. ft. at 0°F. and sea level.

Forced draft is the creation, by means of a fan or blower, of a positive pressure greater than that of the atmosphere, under the fuel bed. With pulverized coal, the primary, and both the primary and the secondary air if preheaters are used, are supplied under forced draft, but sufficient induced draft is usually provided to maintain a slightly negative pressure within the furnace.

In small installations, steam-jet blowers are sometimes employed to force the air through the fuel bed. Steam jets are sometimes employed above the fire to produce turbulence and assist in smoke abatement.

Induced draft is the creation of a suction or negative pressure within the boiler by a fan located in the path of the flue gas. The induced-draft fan, because of the temperature of the gas handled and its subsequent greater volume, must be larger than the forced-draft fan. (For fan characteristics see section on Forced and Induced Draft Fans under Auxiliaries, pp. 2485 and 2486.)

Where boilers are operated at relatively low rates, the stack will usually provide sufficient draft, but where high rates of combustion are desired or when economizers or air preheaters are used, or the stack is of restricted height, it becomes necessary to employ either forced or induced draft, or both, to pass the necessary air through the fuel bed and remove the products of combustion. Where the smaller sizes of anthracite are burned, or where a thick fire with bituminous coal is carried on a stoker, forced draft is necessary. It is the usual practice, when forced draft is used, so to regulate the pressure that there is a balanced pressure in the furnace or a slight suction above the fuel bed.

Several systems of combustion control are now in successful use in which the supply of fuel, air, and boiler feed water is automatically controlled to meet the steam demands upon the boiler.

Stokers

Stokers may be classified as follows: hand or semiautomatic; traveling or chain grate, with natural or forced draft; overfeed of the front or side-feed and spreader types; underfeed single- or multiple-retort type.

Table 3. Table of Economical Sizes for Natural-draft Chimneys

Based on: 62°F. atmospheric temperature, 500°F. chimney gas temperature, 0.09 chimney gas density, 0.016 coefficient of friction, sea level elevation.

Amount of gases flowing, lb. per sec.	Required draft at breeching entrance, in. water												
	0.4	0.5	0.6	0.7	0.8	0.9	1.0	1.1	1.2	1.3	1.4	1.5	1.6
	Economical size of chimney height and diameter, ft.												
5	75 3.5	95 3.5	115 3.5	135 3.5									
10	75 4.0	95 4.0	115 4.0	135 4.0	150 4.0								
15	75 4.5	95 4.5	115 4.5	135 4.5	150 4.5	170 4.5	190 4.5						
20	75 5.0	95 5.0	115 5.0	135 5.0	150 5.0	170 5.0	190 5.0						
25	75 5.5	95 5.5	115 5.5	135 5.5	150 5.5	170 5.5	190 5.5						
30	...	95 5.5	115 5.5	135 5.5	150 5.5	170 5.5	190 5.5						
35	...	95 6.0	115 6.0	135 6.0	150 6.0	170 6.0	190 6.0	210 6.0					
40	...	95 6.5	115 6.5	135 6.5	150 6.5	170 6.5	190 6.5	210 6.5					
45	...	95 6.5	115 6.5	135 6.5	150 6.5	170 6.5	190 6.5	210 6.5					
50	...	...	115 7.0	135 7.0	150 7.0	170 7.0	190 7.0	210 7.0	230 7.0				
60	...	...	115 7.5	135 7.5	150 7.5	170 7.5	190 7.5	210 7.5	230 7.5				
70	...	...	...	...	150 8.0	170 8.0	190 8.0	210 8.0	230 8.0	250 8.0	270 8.0	290 8.0	310 8.0
80	...	...	...	...	150 8.5	170 8.5	190 8.5	210 8.5	230 8.5	250 8.5	270 8.5	290 8.5	310 8.5
90	...	...	...	...	...	170 9.0	190 9.0	210 9.0	230 9.0	250 9.0	270 9.0	290 9.0	310 9.0
100	...	...	...	...	...	170 9.5	190 9.5	210 9.5	230 9.5	250 9.5	270 9.5	290 9.5	310 9.5
110	...	...	...	...	...	170 9.5	190 9.5	210 9.5	230 9.5	250 9.5	270 9.5	290 9.5	310 9.5
120	...	...	...	...	...	170 10.0	190 10.0	210 10.0	230 10.0	250 10.0	270 10.0	290 10.0	310 10.0
130	...	...	...	...	...	170 10.5	190 10.5	210 10.5	230 10.5	250 10.5	270 10.5	290 10.5	310 10.5
140	...	...	...	...	...	170 10.5	190 10.5	210 10.5	230 10.5	250 10.5	270 10.5	290 10.5	310 10.5
150	...	...	...	...	...	170 11.0	190 11.0	210 11.0	230 11.0	250 11.0	270 11.0	290 11.0	310 11.0
160	...	...	...	...	...	170 11.0	190 11.0	210 11.0	230 11.0	250 11.0	270 11.0	290 11.0	310 11.0
170	...	...	...	...	...	170 11.5	190 11.5	210 11.5	230 11.5	250 11.5	270 11.5	290 11.5	310 11.5
180	...	...	...	...	...	170 12.0	190 12.0	210 12.0	230 12.0	250 12.0	270 12.0	290 12.0	310 12.0
190	...	...	...	...	...	...	190 12.0	210 12.0	230 12.0	250 12.0	270 12.0	290 12.0	310 12.0
200	...	...	...	...	...	...	190 12.5	210 12.5	230 12.5	250 12.5	270 12.5	290 12.5	310 12.5

Traveling or chain-grate stokers, with natural or forced draft, depending upon load conditions, are extensively used to burn Midwestern bituminous coals. These coals are free burning and generally high in ash, which has a low fusing temperature. The fuel bed must not be agitated. It is possible to burn Eastern bituminous coal of the coking variety on these stokers, if they are provided with agitating plates. Anthracite and coke breeze usually require the traveling grate stoker with forced draft, whereas this type, with or without forced draft, may be used to burn subbituminous coal or lignite.

Overfeed spreader stokers are used more extensively than formerly in some sections and are adapted to a wide variety of bituminous coals. They operate to best advantage when fired with 1 to 3/8 in. screenings or slack. They require ignition arches to ignite and coke the coal. They are suitable for combustion rates up to 35 lb. per sq. ft. grate area per hour. One design of inclined overfeed stoker has been used to burn anthracite. They are seldom employed under boilers of over 6000 sq. ft. of heating surface, as they cannot be forced to such high rates of combustion as the underfeed type, and when forced are likely to produce objectionable smoke and also cause excessive sifting into the ashpit.

Underfeed stokers are used more than any other type in burning Eastern bituminous coal of the low-ash coking variety, inasmuch as the underfeed mechanism keeps the fuel bed broken up. With forced draft they are capable of high overload and do not produce excessive smoke. This type of stoker has more lately been employed to some extent with high-ash coals and with mixtures of bituminous coal and coke breeze or culm. However, where the ash runs to 20 per cent or more, the traveling grate must be used. Large stokers of the underfeed type are usually provided with a clinker grinder at the rear end, and recent designs have also been provided with ashpit and clinker grinders submerged. Combustion rates as high as 75 lb. of coal per square foot of grate surface per hour have been obtained when the air is automatically metered to the various zones of the stoker, but 40 to 50 lb. is generally the practice except on peak loads for short periods.

Pulverized Coal*

Two systems of pulverized-coal burning are in vogue, namely: (1) the storage system, in which the coal, after being crushed, is dried, pulverized, and stored in a bin to be conveyed to the burners as required; and (2) the unit or direct-firing system in which the drying and pulverizing are accomplished in the mill, which feeds directly to the furnace. The storage system has been used only in large plants, and even there has given way to the unit system.

Mills used for pulverizing coal include roller, bowl, ball-ring, impact, and ball-mill types.† Fineness of grinding is important and is determined by running a sample through a series of screens and obtaining the percentage going through each. Screen sizes usually employed include 200, 100, 50, or 40 mesh. Low-volatile coal must be pulverized to a finer degree than high-volatile coal. Thus, a good fineness for low-volatile coal is 80 per cent through 200-mesh, 95 per cent through 100-mesh, and 99.5 per cent through 50-mesh screen, and for high-volatile coal 65 per cent through 200-mesh, 90 per cent through 100-mesh, and 99 per cent through 50-mesh screen. It is more important that the percentage on a 50-mesh screen be low than to have a high percentage through a 200-mesh screen.

---

* See Fuels section, pp. 2323–2430.

† See Sec. 16.

The power consumed in crushing, pulverizing, and conveying the coal to the burner will range from 15 to 35 kw.-hr. per ton, depending on the system, type of mill, and character of the coal.

Although pulverizing is confined largely to bituminous coal, it is applied successfully to lignite, and, in a few instances, to anthracite in favored localities where the low cost of culm offsets the increased pulverizing and maintenance costs. About 35 kw.-hr. per ton is consumed in pulverizing anthracite compared with 5 to 7 kw.-hr. with bituminous.

Relative Advantages of Stokers and Pulverized Coal. It is impossible to make any general statement as to which method of firing is the better, as local conditions will govern in every case. The efficiencies obtainable with either are comparable. Certain coals will give better results with stokers, whereas pulverizers offer greater flexibility in operation. The banking losses are likely to be less with pulverized coal, but the power consumption is somewhat greater. The pulverized coal can be burned with less excess air, but in sections where the fly ash escaping from the stack is likely to become a nuisance, it may be necessary to go to the expense of installing ash-recovery equipment. On the other hand, the maintenance on stokers is usually higher than on pulverized-coal equipment. Higher heat releases per cubic foot of furnace volume are obtainable with stokers than with pulverized coal, but while high preheat has been employed with stokers, it is more general with pulverized coal.

Data collected yearly by *Power* on utility and industrial plants show that from September, 1937, to December, 1938, 81 utility plants were installed with pulverized-coal firing, 26 with stoker firing, 14 with oil firing, 6 with gas firing, 11 combined pulverized coal, oil, and gas, and 9 combined oil and gas firing. During the same period 66 industrial plants installed pulverized-coal equipment, 48 installed stokers, 22 installed oil burners, 4 gas burners, 12 combined pulverized coal and oil, 5 installed combination pulverized-coal and gas equipment, and 13 combined oil and gas.

Oil Burning*

Because of the excess production of crude oil and improvement in oil-burning equipment, fuel oil has become a strong competitor of coal at low prices in many sections of the country. Many of the plants that have changed over to oil have done so without removal of the coal-burning equipment, thus making it possible to take full advantage of periodic fluctuations in fuel prices and providing the owner with the means of bargaining in the fuel market. Furnaces designed for burning pulverized coal are well adapted to burning oil; in fact the two fuels may be burned simultaneously or the change-over made while the boiler is under steam. To change from stoker to oil firing it is necessary either to remove parts of the stoker or to floor over the operating mechanism.

Combination burners, capable of burning pulverized coal or coke, gas, and oil, either separately or together, are now being extensively used.

Three types of oil burners are now in common use, namely, the steam-atomizing, mechanical-atomizing, and rotary-cup types. The former is employed generally with the heavier fuel oils and with refinery wastes. About 2 per cent of the steam output of the boiler is required for atomization. Therefore, where it is necessary to keep the make-up water to a minimum, mechanical atomization has some advantage. With the latter the steam

---

* See p. 2352.

required for heating and pumping the oil will, under favorable conditions, be about 1 per cent of the boiler output.

The temperature to which the oil must be heated will depend upon its viscosity. Heavy oils will generally require heating up to 100° or 110°F. for pumping and then to 150° to 220°F. in closed heaters for mechanical atomization. The oil should be heated no higher than is necessary to reduce the viscosity to about 15 sec. Saybolt-Furol. With mechanical atomization the pressure may range anywhere up to 300 lb. per sq. in., depending upon the oil.

The heating value of fuel oil averages about 18,500 B.t.u. per lb., and in stationary-plant practice heat releases of from 35,000 to 50,000 B.t.u. per cu. ft. of furnace volume are practical where water-cooled walls are employed.

No general comparisons can be made between oil and coal as many factors, influenced by local conditions, determine the ultimate economies. On a heat-content basis, a very rough rule that is sometimes employed is to consider that the price of oil, in cents per gallon, should be one-half that of coal in dollars per ton, to make the two equivalent. However, the cost of fuel handling, investment costs, banking losses, cost of handling ashes with coal, the problem of fly ash with pulverized coal in certain localities, and the steam required for heating, pumping, and atomizing with oil are all factors that must be considered in the final analysis. Boiler efficiencies with coal and oil are comparable, the loss due to the latent heat of the water vapor in the products of combustion resulting from the hydrogen content of the oil being offset by ashpit or stack losses with coal. Oil can be burned with low excess air (from 15 to 20 per cent), and because of the ease of control a close regulation of superheat is possible.

Boiler Feed-water Treatment

No definite rules can be given for the treatment of boiler feed water, as the analysis of the water and the operating conditions will dictate the treatment in the individual case.

Feed-water treatment may be broadly considered in two classes, namely, internal and external. In the former, chemicals are introduced into the boiler to inhibit corrosion, prevent scale formation, or, where sodium salts are employed, to convert calcium and magnesium sulfates into soluble sodium sulfates and precipitate the calcium and magnesium in a form that can be easily removed by blowing down.

In the external treatment of feed water, chemical reagents are applied in a softening tank which may be of the continuous or intermittent type. Continuous softeners are of either the cold- or the hot-process type. The latter combines the function of feed-water heater and softener in which the scale-forming solids are removed partly through chemical reaction and partly as a result of heat.

Another external method of treating feed water is by the use of zeolites, which have the property of removing calcium and magnesium from the water as it is passes through the bed of zeolite and replacing with sodium. The zeolite is regenerated, after it has been depleted of sodium, by treating with a solution of common salt.

The adoption of high steam pressures and high rates of forcing boilers has been responsible for an increased use of evaporators to provide pure feed water. These may be of the submerged-tube, film, or flash types and designed for single-, double-, triple-, or quadruple-effect operation. They may be supplied with live, bled, or exhaust steam.

Recognition of the fact that gases dissolved in water will cause corrosion has led to the frequent use of deaerators. These may be of the mechanical or the chemical type. In one design of the mechanical type, preheated water is partly flashed into steam as it enters a vacuum chamber or condenser and the gases are thus released from the water. In the chemical type, oxygen is removed from the water by a reagent.

Embrittlement is a term applied to intercrystalline cracking of boiler steel below the water line and is produced by solutions which attack the grain boundaries. Much controversy has existed as to its cause and it has been the subject of much research. It was long believed that embrittlement was caused by the action of caustic soda (sodium hydroxide) solutions in the presence of sodium silicate on boiler steel in which initial stresses had been set up due to processing or due to distortion at operating temperatures. These salts build up to the necessary concentration, which is many times that found in boiler waters, in drum seams, and other crevices. Early investigations indicated that embrittlement was inhibited by maintaining specific sulfate-carbonate ratios. As a result the American Society of Mechanical Engineers recommended that the following relation between sodium sulfate and total alkalinity be maintained in the boiler water for various steam pressures:

Steam pressure, lb. per sq. in.	Sodium sulfate	Total sodium hydroxide and carbonate alkalinity calculated to equivalent sodium carbonate
0–150	1	1
150–250	2	1
250 and over	3	1

More recent investigations, however, have shown that sulfates cannot be entirely depended upon to inhibit embrittlement in high-pressure boilers, that sodium nitrate or dilute nitric acid solutions have an action similar to sodium hydroxide, and that embrittlement may be produced by sodium hydroxide in solutions containing other substances than silica.

Work by the Bureau of Mines Experiment Station at the University of Maryland (reported in papers by Schroeder, Berk, and O'Brien in *Trans. Am. Soc. Mech. Engrs.*) disclosed among other things that concentrated sulfite waste liquors, lignin sulfonate, Philippine cutch, and quebracho did prevent embrittlement even when a very high stress was applied at high temperatures and in the presence of high sodium silicate or high sodium hydroxide concentrations. It was also found that sodium phosphate does not prevent embrittlement unless the sodium silicate-sodium hydroxide ratio is well below that existing in most feed waters. The Bureau of Mines has developed an embrittlement detector which indicates whether the nature of the boiler water is likely to cause embrittlement.

Welded drums or drums having seams caulked on the inside only give less trouble from embrittlement as the only places for high concentrations of salts to build up is where the boiler tubes are rolled into the drum and at other connections.

Priming of a boiler may be defined as the discharge of slugs of water with the steam caused by the violent bursting of steam bubbles. It is usually caused by insufficient steam-disengaging space in the drum at high rates of evaporation, by carrying too high a water level, or by cutting in a boiler too rapidly. High concentrations of salts and suspended matter are also believed to exert an influence.

Foaming is the existence of foam or froth on the surface of the water. It is generally conceded to result from high concentrations of soluble salts and finely divided suspended matter, and is related to the surface tension. Where foaming is noted, treatment of the water should receive consideration.

Under certain conditions both priming and foaming may occur. They may be controlled by giving attention to the amount of suspended matter and soluble salts in the boiler water, by mechanical purification of the steam, by the use of antifoaming compounds, and by controlling the water level and the rate of evaporation.

Carry-over of solids with the steam to turbines has been troublesome in many high-pressure plants, because the solids accumulate on the turbine blading and reduce ability to carry load. Where the accumulations are soft they may be washed off by passing wet steam through the turbine but where they are hard they must be removed by mechanical means.

To prevent carry-over of solids, various devices, placed in the boiler drum, have been developed for removing moisture from the steam before it leaves the boiler. A more recent development is the steam washer. This device washes the steam with feed water that has a lower concentration of solids than the water in the boiler drum. Washing dilutes the concentration of solids in the water entrained by the steam. The washed steam is then dried by various means. Steam washers are said to reduce the solids delivered by the steam to ½ p.p.m. and moisture to ¼ per cent.

PRIME MOVERS

For the industrial power plant, there is available a wide variety of prime movers. These may be broadly classified under steam engines, steam turbines, internal-combustion engines, and water turbines. Which of these broad types may be selected for the industrial plant depends largely on such basic considerations as the location of the plant with respect to water power or fuel supply, whether or not steam is required in a manufacturing process, and the size of the plant under consideration.

Steam Engines

To the casual observer, it may appear that the steam engine as a prime mover has been replaced by the steam turbine. This is largely true as far as utility power plants are concerned where size of the unit makes the use of a steam engine impractical. In the industrial plant, and for buildings, however, where the units are smaller, the steam engine still is an attractive prime mover.

Steam engines are classified according to the number and arrangement of the cylinders, the type of valve gear, and the speed at which they operate. In the larger sizes, the engine is usually compounded, *i.e.*, the expansion of the steam is made to take place in two or more cylinders. There are three more common arrangements of doing this, namely, the **tandem-compound, cross-compound,** and **vertical types.** In a **tandem-compound engine,** the cylinders are arranged in a straight line, the piston rod being common to both cylinders and the total force transmitted through one connecting rod. The **cross-compound engine** has cylinders, arranged side by side, each cylinder having a separate piston rod, connecting rod, and crank.

There are many types of valve mechanisms, but most of them will fall under one of the following more important types: (*a*) **slide valve,** (*b*) **Corliss,** (*c*) **non-releasing Corliss,** and (*d*) **Unaflow.**

a. Slide-valve Engine. This is the most elementary and simplest type of steam engine. A single eccentric-driven valve is arranged so that it controls the admission of steam into the cylinder, closes to stop the admission of steam during expansion, opens again to allow the steam to exhaust from the cylinder, and closes during compression. This type of valve has various features that make its use undesirable for prime-mover engines. It is, however, used on engines for driving auxiliaries. One of the main faults of this slide-valve engine is that the same port is used for live steam entering the cylinder, after it has been cooled by exhaust steam leaving the cylinder.

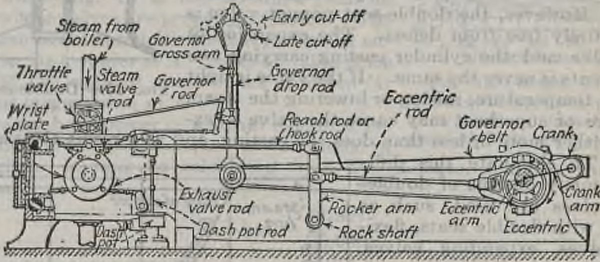


FIG. 8.—General assembly of releasing Corliss valve gear.

b. Corliss Engine. This is one of the early types to adopt the use of separate valves for controlling live and exhaust steam, and is illustrated in Fig. 8. This type of engine has four cylindrical valves that are oscillated in seats or chambers. The two upper valves on top of the cylinder control the admission of live steam, and the two valves at the bottom of the cylinder control exhaust and compression. The eccentric which is keyed to the crank shaft moves the eccentric rod back and forth as the crank shaft rotates, and this in turn oscillates the rocker arm. This rocker arm is pivoted through a shaft or pin on the same pedestal. At the upper end of the rocker arm is connected the reach rod which moves back and forth, and being connected to the wrist plate at the other end causes the latter, which is pivoted to a pin, to oscillate through a considerable angle. To the wristplate are fastened four links, two actuating the exhaust valve and the other two moving the admission valves.

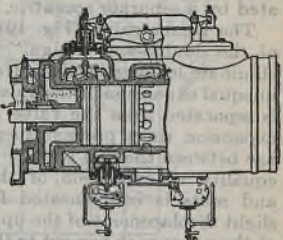


FIG. 9.—Cross section of Unaflow engine cylinder.

The valves fit into bored recesses at the ends of the cylinders. As they rock back and forth in their circular seats, the two ends of the cylinder are alternately placed in communication with the steam supply and the exhaust pipe. Placing the valves at the corner of the cylinders permits short passages to be used, and by giving separate passageways for the live steam and exhaust steam, cylinder condensation is reduced. The use of four oscillating valves, and the method of detaching the steam valves allowing them to close rapidly by the use of dashpots, constitute the chief feature of the Corliss engine.

d. The Unaflow Engine. The main feature of the Unaflow engine is that exhaust takes place through special ports placed around the middle of the cylinder. The piston acts as an exhaust valve uncovering the port as it nears the end of the stroke. With this arrangement the flow of steam is not reversed; decreased cylinder condensation results. It has been almost universal practice to use double-seat poppet valves on Unaflow engines. The poppet valve is necessary if highly superheated steam is used, for slide and other type valves have a tendency to warp at high temperatures. However, the double-seat poppet valve is not entirely free from defect. The expansion of the valve and the cylinder casting carrying two valve seats is never the same. If the valve is tight at one temperature, raising or lowering the steam pressure or superheat may cause the valve to expand either more or less than does the casting, and as a result the valve will leak. To eliminate this difficulty, many designs of double-seat valves are used, such as valves with flexible seats, flexible valves, expanding valves, etc.

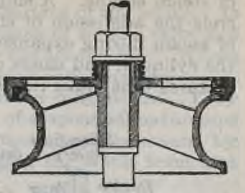


FIG. 10.—Double-seat poppet valve.

Figure 9 is a cross section of a Unaflow engine showing the location of the exhaust port and the valve-operating mechanism. This particular engine is designed for non-condensing operation and is equipped with auxiliary exhaust valves operated by a separate eccentric.

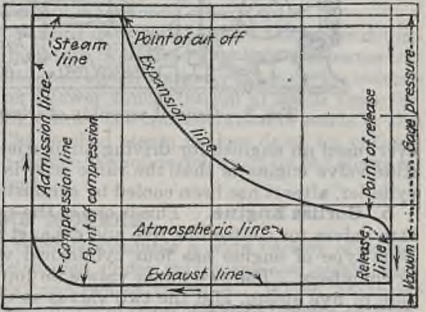


FIG. 11.—Ideal or perfect indicator card of a condensing steam engine.

The steam valve (Fig. 10) is of the double-seat design. To eliminate leakage resulting from unequal expansion of the valve and cylinder heads, the top portion of the valve is separated from the valve body and is held down by spring tension. The expansion rings prevent steam leakage between the two parts. Any inequality of expansion of the valve and seats is compensated for by a slight displacement of the upper part of the valve with respect to the valve body. The steam valves are opened by a bell crank, which carries a roller contacting with the cam (Fig. 9) in the rocker. This rocker is moved through a reach rod and rocker by the governor-controlled eccentric.

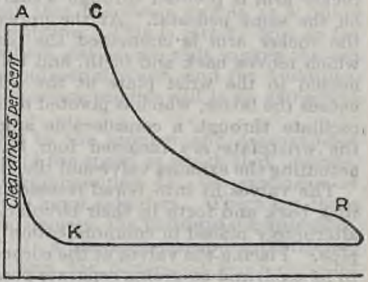


FIG. 12.—Indicator card from a non-condensing steam engine.

Steam-engine Performance Indicator Diagrams. Much can be told about the performance of a steam engine from a study of diagrams made by a steam-engine indicator, a device that records the pressure existing in the cylinder of an engine for all positions of the piston.

The indicator diagram, as it is called, will show whether the engine valves have been properly adjusted; and from various data obtained from it and the engine, at the time the diagram is made, indicated horsepower can be calculated and an estimate of steam consumption made.

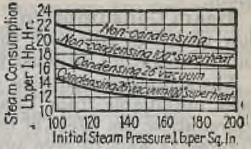


Fig. 13.—Steam consumption of 400 i.h.p. horizontal Unaflow engine. (*Marks, Mechanical Engineers' Handbook.*)

Figure 11 is the ideal or perfect indicator diagram, while Fig. 12 is one obtained from an engine whose valves are properly adjusted. The four valve events are indicated: (1) steam is admitted to the cylinder at A, (2) cut-off takes place at C, (3) the exhaust valve opens at R and (4) closes for compression at K.

Indicated horsepower of a steam engine is the power delivered by the steam to the piston of the engine. It may be calculated from the indicator diagram from the equation:

$$\text{I.hp.} = \frac{PLAN}{33{,}000}$$

where P is the mean effective pressure in pounds per square inch, L the stroke in feet, A the area of the piston in square inches, N the number of strokes per minute and 33,000 is, by definition, the foot-pounds per minute in 1 hp.

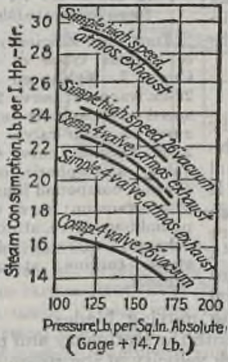


Fig. 14.—Steam consumption of high-speed and four-valve engines, simple and compound.

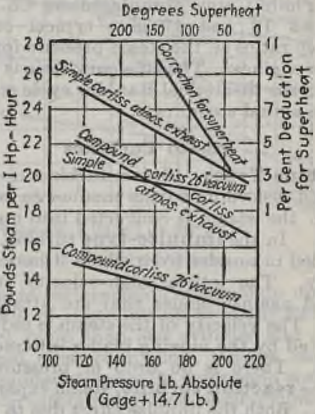


Fig. 15.—Steam consumption of Corliss engines.

The mean effective pressure is obtained from the indicator diagram. Its area is first found by means of a planimeter. By dividing the area by the length of the diagram the average height is found. This, multiplied by the scale of the spring with which the diagram was taken, will give the mean effective pressure.

Although the steam consumption can be estimated from the indicator diagram, it is not recommended, as the result is apt to be in error by 15 to

20 per cent. Steam consumption is best determined either by measuring or weighing the condensed exhaust, or by using a flow meter in the steam-supply main.

Steam Consumption. The power obtainable from a given rate of steam flow varies greatly with the size and type of engine, initial steam pressure and temperature, exhaust pressure, and ratio of expansion.

The steam-consumption curves (Fig. 13) may be taken as typical of steam rates obtainable from Unaflow engines of about 400 hp. capacity. These curves show the effect of both initial pressure and superheat on steam rates. Figures 14 and 15 give typical steam-consumption curves for simple, compound, and Corliss engines.

Ideally, the increase in engine efficiency is nearly proportional to the amount of superheat. In practice, superheat is justified because of reduction in cylinder condensation rather than on thermodynamic grounds. Cylinder condensation is practically eliminated when steam is kept dry at the point of cut-off, which ordinarily is obtained with 150°F. superheat.

The present temperature limit is about 700°F. although two five-cylinder, three-crank, triple-expansion vertical engines of 6060 hp. designed to operate with steam at 1840 lb. per sq. in. pressure and at 820°F. are installed in the power plant of the Philip Carey Manufacturing Co.

Figures 16 and 17 give typical curves of efficiency ratios *vs.* the steam pressure for various types of engines. The efficiency ratio is the ratio between the theoretical Rankine cycle steam rate and the actual steam rate.

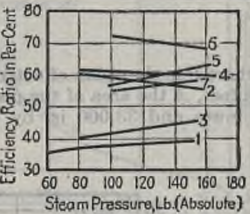


FIG. 16.—Efficiency ratio of various types of engine. Curve 1, high-speed engine, 27-in. vacuum; curve 2, high-speed atmospheric exhaust; curve 3, four-valve engine, 27-in. vacuum; curve 4, four-valve, atmospheric exhaust; curve 5, compound four-valve, 27-in. vacuum; curve 6, compound four-valve, atmospheric exhaust; curve 7, 500-kw. steam turbine, atmospheric exhaust.

Steam Turbines

Turbine Types. Steam turbines are broadly classified according to the manner in which the heat in the steam is converted into mechanical energy. In the **impulse-type** turbine, steam is expanded in nozzles from which it issues at high velocity. The jets of high-velocity steam are directed against blades that are attached to a rotor. The velocity of the steam is reduced and converted by the moving blades into mechanical energy. There is no drop of pressure in the moving blades.

In the **reaction turbine,** steam expands in both the stationary and moving blades. Both the impulse effect due to the high velocity of the steam issuing from the stationary blades and the reaction of the steam expanding in the moving blades are used to produce rotation.

Generally the available pressure drop is subdivided among two or more successive sets of moving blades. This is called *pressure staging.* In a single pressure stage of an impulse turbine there may be two or more successive sets of rotating blades with guide vanes between them. The velocity decrease is divided among them and is called *velocity staging.* Pressure staging is secured in a reaction turbine by alternating stationary and moving blades, each set of blades taking a small drop in pressure. By sufficient subdivision of the total pressure drop, the steam velocities produced can be efficiently converted into mechanical energy by the moving blades of a turbine.

Flow of steam through a turbine is generally in an axial direction. Economical turbines are built, however, in which the steam flow is radial, and another type in which the flow is tangential.

Turbines are high-speed prime movers and for this reason are well adapted for direct connection to alternating-current generators. When used to drive direct-current generators, reduction gearing must generally be resorted to.

Both impulse and reaction turbines can be operated with high-pressure and high-temperature steam either condensing or against back pressure. They may be operated condensing using low-pressure exhaust from other noncondensing prime movers; or as mixed-pressure turbines utilizing both high- and low-pressure steam; or as extraction turbines from which steam at several intermediate pressures may be bled.

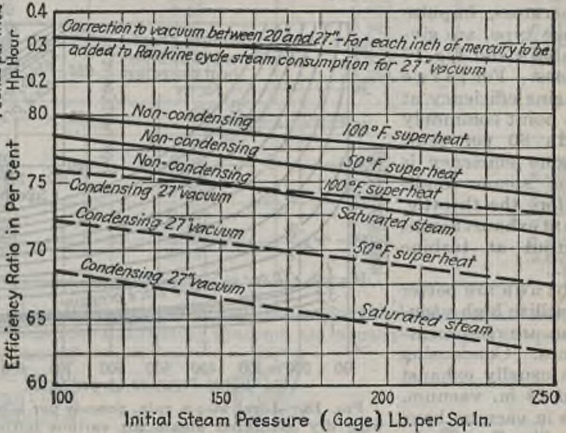


FIG. 17.—Efficiency ratio of Unaflow engines.

Turbines are commercially available for operation with steam up to 2500 lb. per sq. in. pressure and 950°F. and of capacities up to about 200,000 kw. In the larger sizes, over 75,000 kw., more than one cylinder and rotor are often used. These may be arranged cross compound, tandem compound, or vertical compound. European builders frequently compound small units, but American engineers and manufacturers prefer the single-cylinder unit.

Steam Consumption. The steam rate of turbine-generators is universally given as the pounds of steam used per kilowatt-hour output of the generator. The theoretical steam rate of a turbine is determined on the assumption that expansion takes place adiabatically (at constant entropy). The Rankine cycle is employed as the basis of comparing turbine performance. The theoretical steam rate is determined from the following formula:

$$\text{Steam rate, pounds per kilowatt hour} = \frac{3413}{h_1 - h_2}$$

where h_1 = enthalpy in B.t.u. per lb. steam at initial pressure and temperature.

h_2 = enthalpy in B.t.u. per lb. steam after adiabatic expansion to exhaust pressure.

3413 = B.t.u. equivalent to 1 kw.-hr.

The curves of Fig. 18 show the theoretical steam rates for dry-saturated steam at various initial and exhaust pressures. When the steam is initially superheated, the values read from the curves are to be corrected by multiplying by the superheat factors given by the inserted curves (Fig. 18). The factors are average values for the range of back pressures indicated.

The wide variation in theoretical steam rate for various steam conditions makes a tabulation of actual steam rates meaningless. However, actual steam rates can be determined with sufficient accuracy for plant-design purposes by dividing the ideal steam rate by engine and generator efficiencies. Engine efficiency varies with the size, type, and design of the turbine. It also varies with the load. Approximate full-load efficiencies for turbines and generators, impulse multistage type, are given in Table 4 and Fig. 17 for engines. For large units engine efficiency at the best point commonly exceeds 80 per cent. The engine efficiency is based on steam conditions before the throttle, pressure at exhaust flange, and output at turbine coupling.

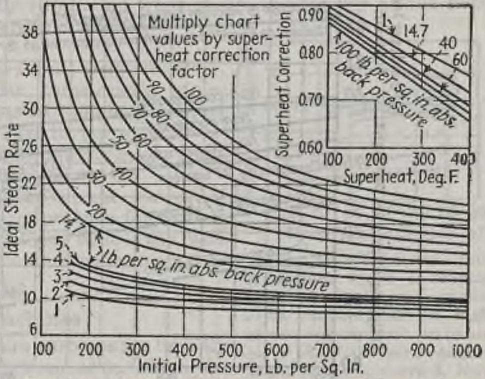


FIG. 18.—Ideal steam rate, pounds per kilowatt-hour, with dry saturated steam for various initial pressures and back pressures.

Turbines are better able to utilize high vacuums than are reciprocating engines. Condensing turbines usually exhaust at 27 to 29 in. vacuum. Changes in vacuum have considerable effect on the turbine steam consumption. For each inch of vacuum secured between 25 and 29 in., the steam rate decreases from 5 to 7 per cent.

High superheats can also be better utilized in steam turbines, as there are no rubbing parts in contact with the steam. Every 7° to 13° increase in superheat improves the steam consumption about 1 per cent. In addition, high superheat eliminates the erosion due to moisture in the low-pressure stages, which may become serious at about 10 per cent.

Figure 19 shows the longitudinal assembly section of a non-condensing steam turbine. Figures 20 and 21 give the total steam flow to a bleeder turbine as a function of the amount of steam extracted.

Lubrication. Bearings of large turbines are lubricated by a forced-circulation system. An oil pump, either mounted on the end of the turbine shaft, or gear driven from the shaft, pumps oil from a storage tank, usually built into the base at the front end of the turbine, through oil coolers to the bearings to be lubricated. Oil from the bearings returns to the storage tank. This same pump also may deliver oil for operating the relays and pistons of the valves under governor control. Bearings of small turbines are often ring-oiled.

Table 4. Approximate Full-load Efficiencies Multistage Impulse Steam Turbine—Generators—Geared Type

Rated capacity at full load, kw.	Brake potential efficiency ratio of turbine θ_t (1)	Generator efficiency at full load θ_g (2)	Combined efficiency at full load $\theta_t \times \theta_g$ (3)
Non-condensing Turbines			
100	0.47		
200	0.48		
300	0.61		
400	0.63		
500	0.64	0.940	0.600
600	0.64	0.942	0.603
750	0.65	0.944	0.614
1000	0.67	0.945	0.633
1250	0.67	0.947	0.634
1500	0.67	0.950	0.637
2000	0.68	0.954	0.649
Condensing Turbines			
500	0.68	0.940	0.639
600	0.69	0.942	0.650
750	0.70	0.944	0.661
1000	0.70	0.945	0.662
1250	0.70	0.947	0.663
1500	0.70	0.950	0.665
2000	0.70	0.954	0.668

Steam-turbine lubrication demands the use of lubricating oils of the highest quality, because the lubricating systems are usually arranged to permit the continuous use of the same oil for an indefinite period with the addition of make-up oil.

While oil is circulating in the oiling system, decomposition occurs which tends to change the properties of the oil. It has been found that the speed at which this aging process, which is attributed to oxidation, takes place varies with different kinds of oil according to the sources from which they are obtained and the degree to which they have been refined. Thus

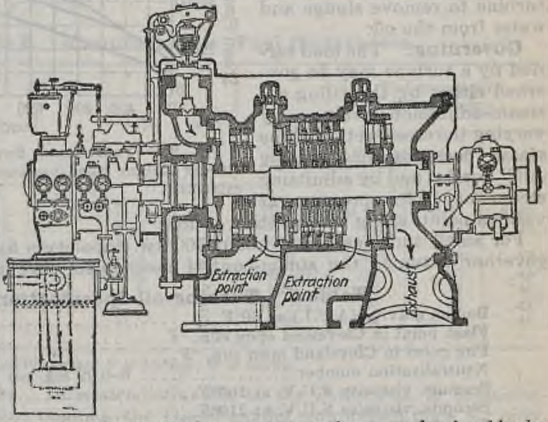


FIG. 19.—Longitudinal cross section of non-condensing bleeder turbine.

the extent to which oil can change during service varies considerably. Likewise it is important to prevent moisture from entering the oiling system, as direct contact of oil with moisture augments oxidation.

The important qualities of turbine oil are indicated by viscosity, demulsibility, sludge content, and neutralization number. A turbine-oil specification of standard refinery practice is given in Table 5. Suitability of a turbine oil can be determined only by its performance in service and not by specifications.

Where a turbine oil shows a steam emulsion (A.S.T.M. method) of over 600, a high neutralization number, and sediment, the turbine needs close watching for oil trouble.

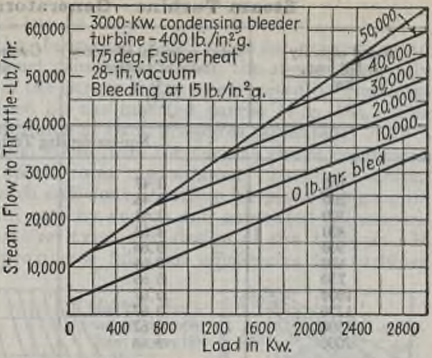


Fig. 20.—Total steam flow to a condensing bleeder turbine when various amounts of steam are extracted.

The quantity of oil in the lubricating system ranges from 0.15 gal. per kw. for small turbines to 0.10 gal. per kw. for large units. Generally some means, such as centrifugal oil purifiers or batch oil filters, should be installed with the turbine to remove sludge and water from the oil.

Governing. The load carried by a turbine may be governed either by throttling the steam-admission valve, by varying the cross section of the steam passages by "cutting out" nozzles and by admitting steam at boiler pressure at various points along the direction of flow.

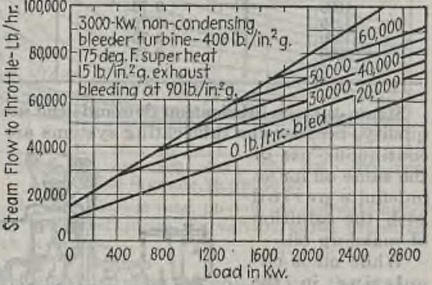


Fig. 21.—Total steam flow to a non-condensing bleeder turbine when various amounts of steam are extracted.

For small turbines, below about 500 kw. capacity a flyball or inertia-type governor actuates the steam-control mechanism mechanically. In larger turbines the governor operates a pilot valve which controls the flow of oil to

Table 5. Turbine-oil Specification

Baumé gravity (A.P.I.) at 60°F.	26–31
Flash point in Cleveland open cup, °F.	380
Fire point in Cleveland open cup, °F.	430
Neutralization number	0–0.02 mg. per KOH per gram
Seconds, viscosity S.U.V. at 100°F.	145–155
Seconds, viscosity S.U.V. at 210°F.	42–44
Demulsibility number, 1620 at 130°F. Herschel or 30–60 steam-emulsion number	

cylinders that operate the steam-control valve. A recently developed governor utilizes the change in oil pressure with change in speed that is developed by a specially designed pump.

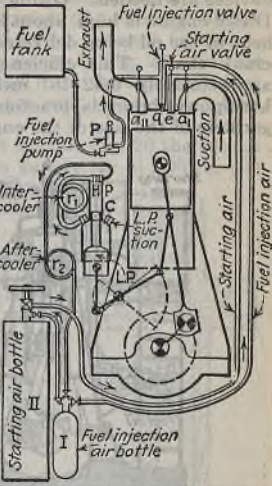


Fig. 22.—Schematic outline of a four-cycle air-injection Diesel.

Most industrial turbines bleed or exhaust steam to process. Many special types of governors are available for controlling the bled- or exhaust-steam pressure. Steam turbine governors are available which permit bleeding varying amounts of steam at constant pressures over a considerable range of turbine load without any practical variation in speed. Back-pressure units can be controlled so that both exhaust- and bleed-steam pressures are maintained constant. When so controlled, the power produced depends upon the amount of steam passed. Governors for mixed-pressure turbines maintain constant speed and load while allowing varying amounts of available low-pressure steam to pass through the turbines.

Diesel Engines

Where process steam is not needed Diesel oil engines may be the most economical power source, and as process and power demands are not always in synchronism, a combined Diesel- and steam-engine plant, with the steam-engine generating power up to the requirements for exhaust steam for processes, is both flexible and economical. American-made Diesels of practically any capacity are available from 5 to 7500 hp.

Table 6. Manufacturers' Fuel Guarantees

Make	Type	Cycle	Size	Fuel consumption, lb. per hp-hr.		
				Full load	¾ load	½ load
National Supply Co.	*V, S*	4	70-420	0.40	0.41	0.44
De La Vergne	*V, A*	4	100-300	.42	.435	.52
Fulton	*V, M, A*	4	50-100	.50		
Bush-Sulzer	*V, A*	2	500	.425	.44	.52
Bush-Sulzer	*V, S*	2	1800-2400	.39	.40	.43
American Locomotive	*V, S*	2	1800	.40	.41	.44
Nordberg	*V, S*	2	1800-2250	.39	.41	.43
Hamilton M.A.N.	*V, S*	2	2250	.39	.40	.43

A = air injection; *S* = solid injection; *V* = vertical; *M* = marine.

The Diesel Cycle. In contradistinction to the Otto-cycle, or explosion-type, gas and gasoline engines, the Diesel engine compresses pure air only, and the fuel is introduced at the end of the compression stroke rather than during the suction stroke. Compression of the air to 400 to 500 lb. per sq. in. pressure is sufficient to raise its temperature to from 900° to 1200°F., depending upon the suction temperature. Fuel, injected into this heated air, is ignited solely by this temperature, and no spark plugs or other ignition devices are needed. Owing to the high-compression ratio, the efficiency of a Diesel engine is high, about 32 per cent of the heat in the oil being delivered as power at the crank shaft. This efficiency, equivalent to approximately 0.42 lb. fuel oil per brake-horsepower-hour, is practically constant regardless of the size of the engine.

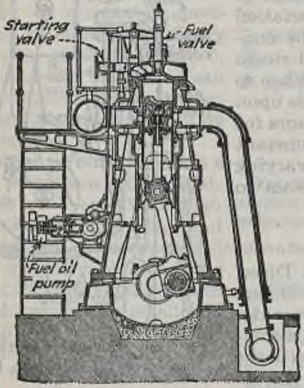


Fig. 23.—Typical two-stroke-cycle air-injection Diesel.

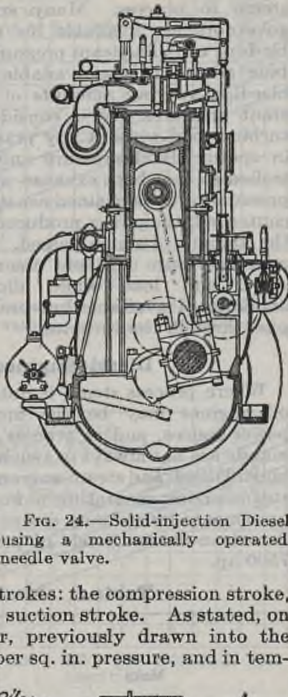


Fig. 24.—Solid-injection Diesel using a mechanically operated needle valve.

The original Diesel cycle is made up of four strokes: the compression stroke, the power stroke, the exhaust stroke, and the suction stroke. As stated, on the compression stroke a charge of fresh air, previously drawn into the cylinder, is raised from atmospheric to 500 lb. per sq. in. pressure, and in temperature from 120° to 1100°F. Just before the beginning of the power stroke, fuel is injected through a spray valve in a finely atomized condition and at a rate such that the cylinder pressure is held at 500 lb. per sq. in. until the piston has traveled about 10 per cent of its stroke. At this point the fuel injection ceases, and the burning gases expand behind the moving piston. Shortly before the end of the stroke is reached an exhaust valve opens relieving the cylinder of its gases, and, as the piston reverses and makes the in-stroke again, the remaining burned gases are expelled. On the next out-stroke, fresh air is inducted through an admission valve and at the end of this stroke the piston starts the cycle again by compressing the new air charge.

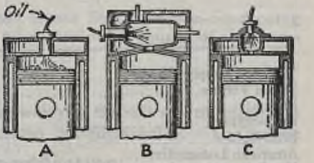


Fig. 25.—Combustion chambers of solid-injection Diesels.

A diagrammatic outline of a four-stroke-cycle air-injection Diesel engine is shown in Fig. 22. Originally all Diesel engines operated on the four-stroke cycle, but many of the engines installed now operate on the two-stroke cycle.

In these engines, at the end of the power stroke, the burned gases are expelled through a set of ports in the cylinder barrel which are uncovered by the piston. At the same time, air under about 3 lb. per sq. in. pressure blows into the cylinder through a second row of ports and scavenges the cylinder of the burned gases as well as supplying a charge of fresh air for the next (compression) stroke. A typical two-stroke-cycle engine is shown in Fig. 23. In this way, almost twice the amount of power is obtained from the same cylinder dimensions, but the fuel consumption is about 5 per cent greater than that of the four-stroke cycle.

Types of Diesel Engines. It will be observed that in Fig. 20 the engine is supplied with a high-pressure air compressor to supply air at 900 lb. per sq. in. pressure which is used to blow the fuel charge into the cylinder when a needle valve opens. This was the original method patented by Dr. Diesel.

In late years, designers have evolved engines in which the fuel is injected by direct pump pressure. In all other respects the engine mechanism remains unchanged. (Figure 24 illustrates this type.) As the work of the pump is less than the work needed to compress the injection air, the airless, or solid-injection Diesel engine, has a slightly better fuel consumption. At present about 80 per cent of the Diesel engines built in this country use solid injections. In general, air-injection Diesels are installed only when high-viscosity, poor-quality fuel is to be burned.

Types of Solid-injection Diesels. Some solid-injection Diesel engines employ a needle valve, lifted mechanically by a rocker arm and cam shaft. The greater number, however, employ some type of spring-loaded needle valve or a simple check valve.

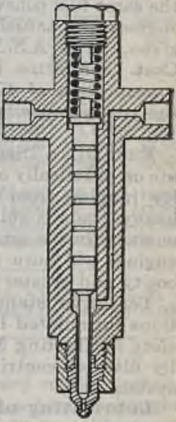


FIG. 26.—Differential-needle spray valve.

There are nearly as many types of combustion chambers as there are makes of Diesels. A few of these are shown in Fig. 25. In the design shown at *A*, the oil is sprayed directly into the engine cylinder exactly as with the air-injection Diesel engine. It is necessary to obtain a fine spray, so the spray valve is heavily spring loaded as in Fig. 26. In Fig. 25*B*, all the cylinder air is forced into a combustion chamber in the cylinder head, and the combustion of the fuel occurs in this cavity. The spray need not be so fine, as the rush of air into the cavity ensures good mixing. Therefore, usually, a simple nozzle with a check to prevent the back flow of combustion gases is employed. In the so-called *precombustion* Diesel engines (Fig. 25*C*), a small part of the cylinder air enters the small chamber in the head. The oil sprays toward the throat opening, thereby meeting the air. As there is a deficiency of air in the chamber, only a small amount of fuel is burned there, and the consequent increase in pressure blows the remainder into the cylinder where it burns. The oil spray is generally coarse and a check-valve orifice, as in Fig. 27, is the prevailing fuel-spray nozzle.

Solid-injection engines do not as a rule have constant-pressure combustion. In some, where the compression pressure is carried from 350 to 400 lb. per sq. in. (ample for ignition as there is no refrigerating effect from expanding high-pressure injection air), combustion at constant volume causes a pressure rise up to about 500 lb. per sq. in., followed by combustion at a falling pressure. Indicator diagrams show a sharp peak at the start of the power stroke.

Engine Weights and Speeds. Up until about 1920, Diesels operated at slow speeds, 164 r.p.m. being the standard maximum. Weights were high, ranging from 300 to 400 lb. per hp. Of late years, speeds have been increased, and engines for power-plant service now operate at from 200 to 600 r.p.m. and weigh from 100 to 300 lb. per hp. Engines of 150 hp. are operating in power service at 1200 r.p.m. and in automatic service at 1800 to 2000 r.p.m. Light-weight, high-speed engines weigh from 19 to 60 lb. per hp.

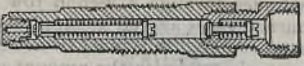


Fig. 27.—Solid-injection spray valve.

Fuel Consumption. Diesel engines of all sizes and makes have about the same fuel consumption. In Table 6 are given a number of guarantees on engines; the actual consumption will be about 5 per cent less than the value given. The A.S.M.E. Diesel Power Cost Committee found that the fuel consumption of 47 plants varied with the yearly capacity factor as shown in Fig. 28.

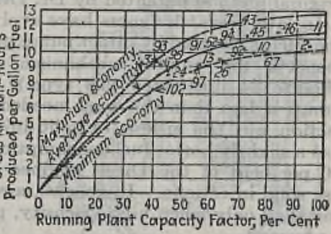


Fig. 28.—Effect of capacity factor on fuel consumption.

Fuel Oil. Diesel engines will operate on practically any fuel oil of a gravity ranging from 16° to 36°Bé. The heavy grade is suitable for slow-speed engines, but most air-injection Diesel engines will burn an oil of greater viscosity and greater carbon residue.

Tentative standard fuel specifications formulated by the American Society for Testing Materials are given on p. 2345. All fuel should be cleaned by filter or centrifuge, or both, before it is delivered to the engine injection system.

Lubricating-oil Consumption. From reports obtained of 47 Diesel plants the A.S.M.E. Diesel Power Cost Committee found that the consumption of lubricants at various yearly load factors falls within the area bounded by the maximum and minimum curves in Fig. 29. Lubricating oil consumption is greatly affected by the type of oil filter or reclaimer employed.

Table 7. Waste-heat Boiler Performance

Lb. of steam per hr., full load; pressure 5 lb. per sq. in.			Sq. ft. of heating surface	
Hp.	4 cycle	2 cycle	4 cycle	2 cycle
75	77	63	72	72
100	92	78	72	72
200	154	165	72	144
300	212	229	72	144
400	327	300	144	192
500	392	370	144	240
600	483	450	192	288
700	550	525	192	336
800	638	600	240	384
900	700	675	240	432
1000	780	750	288	480

Cooling Water. A Diesel plant should be provided with scale-free water for the cylinder-jacket cooling. If the water is of this character the simplest system is made up of a cooling tower over which the warm water from the jackets is run. The cooled water drops into a sump and is forced back through the jacket by a centrifugal or plunger pump.

Table 8. Waste-heat Water-heater Performance—Heat Recovered, B.t.u. per Hr.

Gas temperatures	Diesel engines		Exhaust gas, lb. per hr.	Draft loss, in. water gage	Heating 60°F. Water to 160°F.			Heating 130°F. Water to 150°F.		
	4 cycle, hp.	2 cycle, hp.			700°F. gas	500°F. gas	300°F. gas	700°F. gas	500°F. gas	300°F. gas
Full load:										
4 cycle	75	45	900	0.5	92,000	60,800	29,600	87,400	56,100	25,000
700°F.	100	60	1200	0.8	118,000	78,000	38,000	111,500	71,750	31,900
2 cycle	200	120	2400	2.9	212,000	140,000	68,100	201,000	129,500	57,500
500°F.	300	180	3600	2.9	318,000	210,000	102,500	302,000	194,000	86,500
Half load:										
4 cycle	400	240	4800	3.1	427,500	282,500	137,700	405,000	260,500	116,000
500°F.	500	300	6000	3.1	543,000	359,000	175,000	515,000	331,000	147,000
2 cycle	600	360	7200	2.9	660,000	436,000	212,500	628,000	404,000	179,500
300°F.	650	390	7800	3.0	717,000	474,000	231,000	680,000	437,000	194,000

To obtain pounds of water per hour:	Divide B.t.u. per hr. by 100.	Divide B.t.u. per hr. by 20.
To obtain gallons of water per minute:	Divide B.t.u. per hr. by 50,000.	Divide B.t.u. per hr. by 10,000.

From Foster-Wheeler Corp.

Where the water is bad, it is advisable to install a closed cooling system. Water is circulated through the jacket and then through a heat exchanger, where the heat is removed by raw water, which is then run to a cooling tower or spray pond to be cooled for use again.

It is considered the best practice to design the cooling system so that the flow of water through the engine jackets and the discharge temperature are both held constant for all loads. This may be done by controlling the water by-passed around the cooling system from the jacket discharge to the inlet of the jacket-water pump.

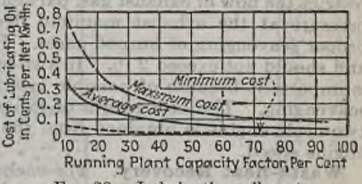


Fig. 29.—Lubricating-oil costs.

From 2000 to 3500 B.t.u. must be removed by the cooling water per brake horsepower per hour. A rise in cooling-water temperature of from 12° to 40°F. is permissible. General practice holds the temperature rise to about 20°. Discharge temperatures over 180° are not recommended, 160° representing average practice. It is well to design the pump to handle about 20 gal. of water per horsepower per hour.

Air Intake. Diesels require about 3 cu. ft. of air per minute per horsepower. Present practice is to filter all air taken in by the engine. Where noise is objectionable air-intake silencers are provided. Pressure drop in the air-intake system should not exceed about 6 in. of water.

Engine Starting. With the exception of engines operating at or above 500 r.p.m., all Diesels are started by compressed air. This air is usually stored in receivers at 250 lb. pressure. A general rule is to provide receiver capacity equal to thirty times the stroke volume of one cylinder.

Exhaust. Diesel engines exhaust about 7.5 cu. ft. of gas per minute per horsepower. The temperature of the exhaust at full load varies from about 500°F. for two-cycle, crankcase-scavenging engines to 700°F. for four-cycle, mechanical-injection engines. Small high-speed Diesels have exhausts as high as 1000°F.

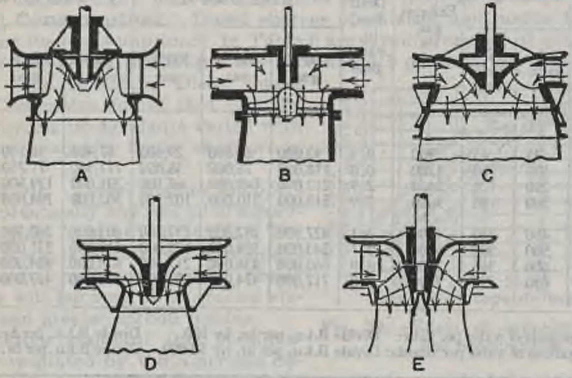


FIG. 30.—Types of water turbine. *A*, Francis; *B*, Kaplan; *C*, Dubs (*Escher-Wyss Co.*); *D*, Nagler; *E*, Moody. (*Barrows*, "*Water Power Engineering*.")

To reduce exhaust noise, silencers should be provided. Silencers can be made to eliminate practically all exhaust noise. Care must be taken not to restrict the flow of exhaust gas, particularly on two-cycle engines, as excessive pressure at the exhaust manifold decreases engine capacity and prevents proper scavenging of burned gases. Exhaust pressure should be held to 1 lb. and should not exceed 2 lb. In two-cycle engines the length of exhaust pipe is important because of possible interference of pressure waves with proper scavenging. The best length of exhaust pipe serving a single cylinder of a two-cycle engine is obtained from the equation $L = 25 \times 200/n$, where n is engine speed.

Waste-heat Recovery. For each pound of oil burned there is approximately a pound of water vapor formed, which passes out with the other gases. The latent heat in this water vapor, amounting to about 500 B.t.u. per h.p.-hr. cannot be recovered in a water-heat boiler, for condensation would cause corrosion. Of the remainder, the sensible heat in the gases, enough may be recovered to bring the gas temperature to about 300°F. Of the heat released by the combustion of fuel about 32 per cent develops power, about 32 per cent is in the exhaust gas, about 32 per cent is in the jacket water, and the remainder is lost to friction and radiation. Waste heat may be used for heating water or for generating steam. Usually waste-heat boilers are of the tubular type, but recently several water-tube boilers have been developed for this work. Table 7 shows the pounds of 5-lb.-pressure steam that may be generated per hour and Table 8 the amount of heat recoverable by heating water.

Water Turbines

The power available from the fall of water is equal to the product of the weight of water flowing per unit of time into the total head. The theoretical horsepower available may be expressed by the equation,

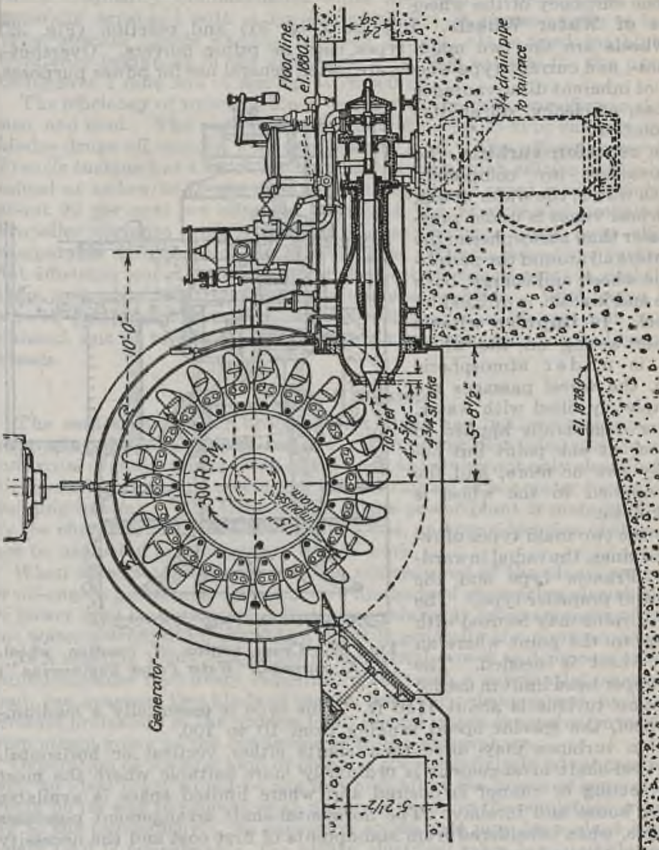


Fig. 31.—Impulse-type water-turbine with governor-operated needle valve. (*W. P. Creager and J. D. Justin, "Hydroelectric Handbook."*)

$$\text{Horsepower} = \frac{Q \times D \times H}{550}$$

where Q is the cubic feet of water discharged per second, D the density of water equal to 62.36 lb. per cu. ft. at 60°F., and H the total head or difference in height between the surface of the water in the forebay when the turbine is operating, and the surface of the water in the tailrace.

All this power is not available to the water turbine, nor can the turbine itself utilize all the energy of the water that is made available to it. The ratio of the power developed by the wheel to the power theoretically available to it is the efficiency of the wheel.

Types of Water Wheels. Impulse (Fig. 31) and reaction (Fig. 32) water wheels are the two main types used as prime movers. Overshot-, undershot-, and current-type wheels are not in general use for power purposes, because of inherent disadvantages as to size, efficiency, durability, and regulation.

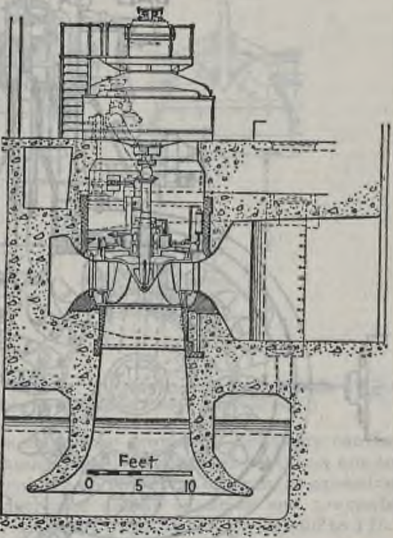


Fig. 32.—Cross section of reaction wheel. (*H. K. Barrows, "Water Power Engineering."*)

In the **reaction turbine,** the wheel passages are completely filled with water; the water acting on the wheel vanes is under pressure greater than atmospheric; the water enters all around the periphery of the wheel; and energy, both pressure and kinetic, is utilized by the wheel. In **impulse wheels** the water acting on the wheel buckets is under atmospheric pressure, the wheel passages are not completely filled with water, the water is generally applied to the wheel at one point but occasionally two or more, and the energy applied to the wheel is wholly kinetic.

There are two main types of reaction turbines, the radial inward-flow or Francis type and the axial-flow or propeller type. The reaction turbine may be used with heads up to the point where an impulse wheel is needed. The present upper head limit in use for the reaction turbine is about 1150 ft. This type is essentially a medium-speed wheel, the specific speed ranging from 10 to 100.

Reaction turbines may have their shafts either vertical or horizontal. The vertical-shaft arrangement is ordinarily more suitable where the most efficient setting or runner is desired and where limited space is available for power house and forebay. The horizontal-shaft arrangement possesses advantages, when considered from standpoints of first cost and the necessity of expensive thrust bearings.

The propeller-type wheel is of high speed, used for specific speeds varying from 110 to 225. It is adapted for use under heads up to about 100 ft. While the high speeds permit the use of wheels and generators of small diameters for a given power output and hence tend to lessen cost, the best efficiencies of reaction wheels are obtained when the Francis medium-speed types are used.

The **impulse wheel** consists of a disk upon the periphery of which are mounted buckets. The buckets, as generally built today, are of a doublelobe form, the two parts coming together in a common splitter. A high-velocity jet is directed tangentially against the runner and imparts its kinetic energy to the buckets. The jet is diverted laterally by the bucket, approximately 180 deg. from its initial absolute direction. To obtain higher speeds and greater power from a single wheel, more than one jet is sometimes used. More frequently double-runner units are used. The impulse turbine is generally arranged with a horizontal shaft. It is essentially a low-speed type, specific speeds varying between 3 to 6. It is used for high heads, ordinarily beginning in the neighborhood of 500 to 1000 ft., the upper limit being over 1 mile and in size up to 70,000 hp.

The efficiency of water turbines varies with the type, specific speed, setting, size, and load. The maximum efficiency of propeller-type runners with fixed blades drops off rapidly with decreases in load. The efficiency curve of the Francis turbine has a much flatter characteristic, high efficiencies being maintained at as low as 70 per cent of rated load. At rated loads, efficiencies of about 92 per cent are obtained with both the Francis and propeller types. Propeller turbines with adjustable blades (Kaplan type) have efficiencies comparable with those of the Francis type. The impulse-type turbine has a flat efficiency curve, but in general the best efficiency point is slightly lower than that obtained from the reaction-type turbines. With some impulse wheels, and under certain conditions, efficiencies of 90 per cent have been claimed, but 87 to 88 per cent is a more usual efficiency for well-constructed wheels.

Heat Balance

The central-station or utility power plant has but one purpose, that of generating electrical energy for sale. The power plant designed to serve an industrial organization is often called upon to furnish not only electrical energy for operating the plant machinery but also steam or hot water for process and building heating. The basic design of the power plant is materially affected by the character of these secondary services, and consideration of them should not be neglected in preliminary design work.

When these secondary services are not required of the plant, water power or oil-engine power may be attractive methods of generating electrical energy or power may be purchased. But when relatively large amounts of steam or hot water are required, the steam plant is generally the more economical.

As a starting point in a study of industrial power and process steam, one should consider that every industrial plant, with a substantial process-steam load, can generate two kinds of power: first, very cheap power, up to the by-product limitation of the process load; second, more expensive power derived from steam going to the condenser or to the atmosphere.

Generation of the first kind of power is always definitely advantageous. In fact, an ideal plant would be one in which all of the power is generated from steam on its way to process, with no steam going to the condenser or exhausting to the atmosphere. An exception to the rule may be a plant where the power load is considerable, the utility electrical rates low, and the process-steam requirements so small as to justify but simple boiler-room equipment.

The second kind of power generally costs more to produce than the generating cost of central-station power and will frequently cost more than the selling price of the latter. This is due to the poorer economy of the smaller units and their greater first cost per unit of capacity. If an installation is to be economically successful, the engineer must select the generating equipment and determine the heat-balance layout, so that as much as possible of the power generated will be of the cheap by-product type.

Assuming that the amount of power and the quantity and pressure of the process steam required have been determined from surveys and estimates, the boiler pressure or initial pressure at the turbine throttle should be decided upon. This pressure may be selected so that the average process-steam demands will generate the average power load as its pressure is reduced in the prime mover from initial pressure to that required for process.

With the boiler pressure selected in this way, there will be times when a high power load and a low process load exist, under which condition the deficiency in power must be made up by electricity, either purchased, or generated from steam going to the condenser or to the atmosphere. It may also be that at times there is a low power load and a high demand for process steam. Under these conditions the process steam will have to be supplied from separate low-pressure boilers or through a reducing valve, the latter being the more common practice. The desirable result always should be a maximum amount of by-product power with a minimum amount of steam going to the condenser, and a minimum amount of live steam going through reducing valves to the process, unless the condenser cooling water can be used to meet process hot-water demands.

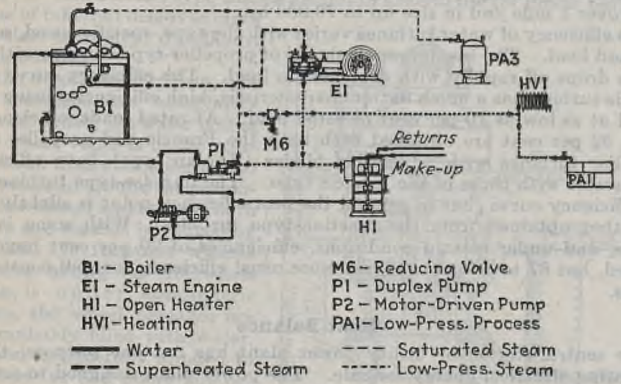


FIG. 33.—Engine exhausting steam to heating and process.

Obviously, in any particular power plant there may be certain conditions of power load and process demands that prevent the application of this broad general rule; for example, when the process condensate returns to the boiler, and when condensing water is not available, it may be advisable to choose a boiler of such pressure that the minimum process steam will generate the maximum power demand, the remainder of the process steam coming direct from the boiler.

A common method of power-plant analysis includes the construction of typical 24-hr. power-load and process-steam-demand curves. By using the steam-rate curve of the prospective prime mover, the hourly generation of steam for power can be set up. The difference between that curve and the process-steam curve is the excess or deficiency in the process requirements. By constructing such curves to cover several boiler pressures, which would result in differences in steam consumption of the generating units, the best boiler pressure for a particular condition can be ascertained with some certainty.

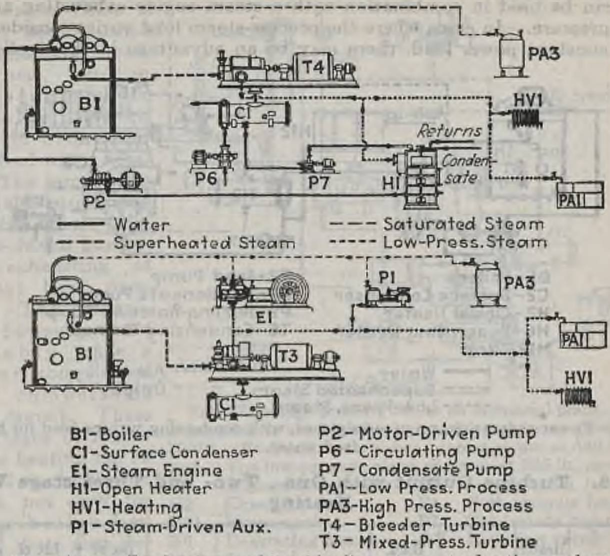


FIG. 34.—Above: Condensing turbine bleeding steam to heating and process. Below: Engine exhausting to mixed-pressure condensing turbine and to process.

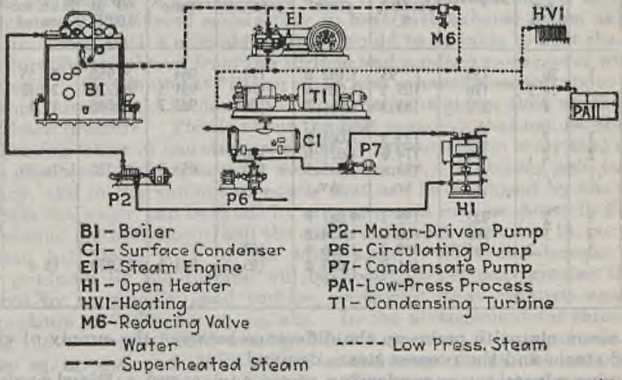


FIG. 35.—Engine exhausting to process and low-pressure condensing turbine.

The selection of a prime mover for use in an industrial plant depends largely upon the peculiarities of the industrial process that is to be served. For example, one may use a back-pressure turbine or engine, or a combination of a condensing turbine and a back-pressure engine. Then there are also various combinations of bleeder turbines and mixed-pressure turbines, many of which can be used in combination with a steam engine exhausting against a back pressure. In cases where the process-steam load varies considerably, with a constant power load, there may be an advantage in the application of an accumulator to make up the difference between the supply of exhaust or bled steam and the process-steam demand.

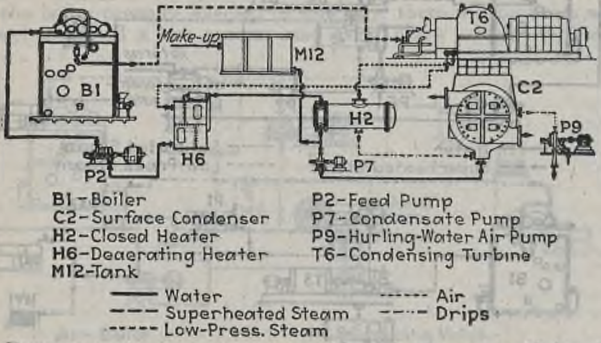


FIG. 36.—Power generation, no process steam, with condensing turbine bled for heating feed water.

Table 9. Turbine Output with One-, Two- and Three-stage Water Heating

Number of stages	Heater	Heater pressure, lb. abs. per sq. in.	B.t.u. available for power	B.t.u. converted to power (at 60% eff.)	B.t.u. remaining per lb. steam	Assumed condensate temperature	B.t.u. to water per lb. steam	Lb. of steam to heat 500 g.p.m. 60° to 160°F.	Lb. of steam per boiler h.p. equiv. to B.t.u. converted	Developed b.h.p.
1	1	20	155	93	1,102.0	170	964.	25,950	27.35	950
		14.7	176	105.5	1,089.5	170	951.5	26,270	24.12	1,088
		7	222	133.3	1,061.7	170	923.7	27,080	19.1	1,417
2	1	7	222	133.3	1,061.7					
	2	2	290	174.0	1,021.0	120				
	..	....	...	153.6 Av.	1,041.3 Av.	120	953.3	26,220	16.55	1,585
3	1	7	222	133.3	1,061.7					
	2	3	268	160.8	1,034.2					
	3	1.2	315	189.0	1,006.0	103				
	..	....	...	161.0 Av.	1,034.0 Av.	103	963.3	25,970	15.8	1,644

In other plants a non-condensing steam engine and a Diesel engine will work out best, especially in the paper industry where the power is always beyond that generated from the process steam.

Flow diagrams showing various possible arrangements for prime movers are shown in Figs. 33 to 38 (*Power*, **74**, No. 21, Nov. 25, 1930).

Multistage Heating. Central stations have been able to realize very considerable improvement in thermal economy by multistage heating of the feed water. This is usually accomplished by extracting steam from one, two, or three stages of the turbine, and using this steam in closed or open feed-water heaters. The same result might also be obtained by using steam from engine-driven auxiliaries, exhausting at different back pressures. This arrangement, however, would probably make a rather complicated and cumbersome plant layout. These advantages of regenerative heating can be utilized in industrial plants, not only for heating boiler feed water, but also for heating water used in manufacturing processes.

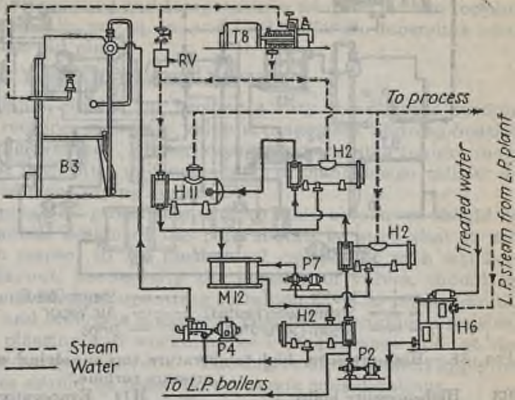


FIG. 37.—High-pressure top to an industrial plant. Steam is supplied to the high-back-pressure turbine at 1250 lb. and 750°F. Evaporator furnishes process steam at 180 lb. pressure. The low-pressure plant operates at 225 lb. pressure.

B3	High-pressure boiler	P2	Low-pressure feed pump
H2	Closed heater	P4	High-pressure feed pump
H11	Evaporator	P7	Condensate pump
H6	Deaerating heater	RV	Reducing valve
M12	Tank		

If the temperature of a quantity of water is to be raised from 60° to 160°F., it is commonly considered satisfactory to heat with exhaust steam at 20 lb. per sq. in. abs. With a suitable heater it would be possible to heat the water with atmospheric exhaust from the turbine and produce more power with the same steam. By running at 7 lb. per sq. in. abs. exhaust pressure, about 43 per cent more power could be developed with the same steam than at 20 lb. per sq. in. back pressure. This is about the best economy that can be obtained when heating water in one stage, whether by exhaust steam or by bled steam.

It is possible, however, to work more efficiently. A decided gain in cycle efficiency, and in the amount of power that can be produced by the steam that heats the water, can be made by arrangements such as shown in Fig. 39. In the second method shown, half the steam does work down to 7 lb. per sq. in. abs., and half down to 2 lb. per sq.in. abs., or 25.85 in. vacuum. The power produced by the turbine will be about 67 per cent greater than is produced by an equally good turbine, or engine, that exhausts against a back pressure of 20 lb. per sq. in. abs. In the arrangement for three-stage heating the turbine exhausts one-third of its steam at 7 lb. per sq. in. abs., 3 lb. per sq. in. abs., and 1.2 lb. per sq. in. abs., respectively, the last pressure being equal to about 27.55 in. vacuum. The power produced represents a 73 per cent gain over the turbine at 20 lb. per sq. in. abs. back pressure.

Table 9 gives approximate figures based on 150 lb. initial pressure, no superheat, and the use of various arrangements for feed-water heating. Turbine efficiencies are all taken at the conservative value of 60 per cent.

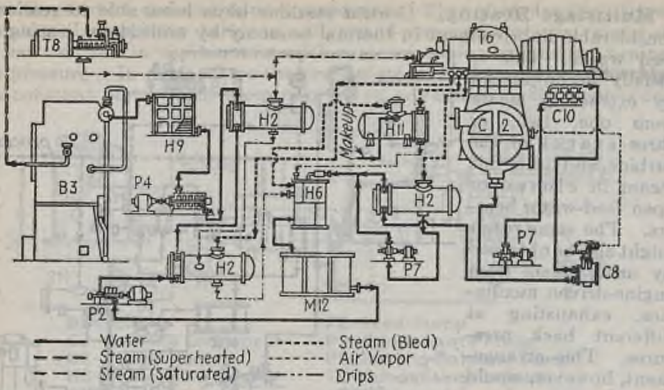


Fig. 38.—High-pressure, high-temperature top supplying steam to an existing low-pressure turbine.

B3	High-pressure boiler	H11	Evaporator
C2	Surface condenser	M12	Tank
C8	Steam-jet vacuum pump	P2	Feed pump
C10	Generator air cooler	P4	High-pressure feed pump
H2	Closed heater	P7	Condensate pump
H6	Deaerating heater	T6	Condensing turbine
H9	Economizer	T8	High-pressure turbine

The heaters are all of the closed type, operating at 15°F. terminal difference. For the single-stage heater the condensate temperature is shown 10°F. above the leaving-water temperature. In multistage heating, the condensate is passed through succeeding heaters and is assumed to leave at 10°F. above the exit-water temperature from the heater at the lowest pressure.

As shown by Fig. 1, many utilities have materially improved the efficiency of their old plants, while adding capacity, by "topping" with a high-pressure unit. This involves the addition of a high-pressure boiler and a high-pressure turbine that will exhaust at the existing boiler pressure and so supply steam to the existing turbine units. Industrial plants may, under proper circumstances, economically adopt this method of adding plant capacity and at the same time improve the efficiency of the entire plant owing to the inher-

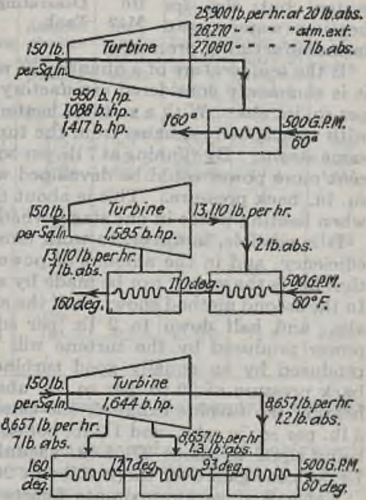


Fig. 39.—Turbines arranged for one-, two-, and three-stage water heating.

ently higher efficiency possible with high-pressure and high-temperature steam. Exhaust steam may be used for process as in Fig. 38 or in low-pressure turbines. The pressures and temperatures adopted for the topping unit range from 600 to 1400 lb. per sq. in. and 750° to 950°F., depending upon the steam pressure of the old plant.

POWER-PLANT PIPING

In modern power plants, piping has become a very complicated system. Continuity of service requires elaborate valve arrangement and duplication of portions of the piping system. Higher steam temperatures require more elaborate provisions for flexibility, and refinement in plant design call for an increasing number of different piping systems.

In laying out the piping of a power plant, three main objectives should be observed. The mechanical design of the pipe should be such that it will function properly with respect to the mechanical equipment with which it serves. The general layout, particularly the location of valves, should be such that it is convenient from an operating standpoint. The piping should be arranged in as neat and orderly a manner as possible to produce a finished assembly that will be pleasing and workmanlike in appearance and, at the same time, accessible and easy to operate. The degree of refinement applying to these main objectives should be based on economic considerations.

Pipe Size. After the basic piping diagrams have been decided upon and before any detail drawings can be attempted, it is necessary to decide upon the sizes of the various pipe lines that make up the system. Pipe sizes should be determined on the basis of reasonable velocities and friction losses, together with a consideration of investment cost. From the standpoint of investment cost, it is desirable to keep the velocity as high as possible without exceeding the maximum allowable velocity. The maximum allowable velocity of the fluid in a pipe line is that which corresponds to a permissible pressure drop or friction loss from the point of supply to the point of consumption. Values given in Table 10 are reasonable. They represent average values used in power-plant practice and may be used as a guide in many cases where actual pressure drops are not computed. The lower velocity should be used for small pipes, and the upper limits for larger ones.

Table 10. Velocities of Steam and Water in Power-plant Piping[1]

Fluid	Pressure, lb. per sq. in. gage	Use	Reasonable velocity, ft. per min.
Water	25–40	City water	120–300
Water	50–150	General service	300–600
Water	150 up	Boiler feed	600
Saturated steam	0–15	Heating	4,000– 6,000
Saturated steam	50 up	Miscellaneous	6,000–10,000
Superheated steam	200 up	Large turbine and boiler leads	10,000–15,000

[1] From Walker and Crocker, "Piping Handbook," McGraw-Hill, New York, 1930.

Kind of Pipe. For general use around power plants, wrought-steel pipe is used more than any other kind. It should be used for all steam lines, boiler-feed high-pressure lines, and condensate lines. Cast-iron piping is used for underground floor drains, sewers, wet soot removal, and ash removal. Galvanized-steel piping is frequently used for compressed-air lines and for the smaller sizes of city water and general service use. Principal standard pipe specifications are listed on the following page:

PRINCIPAL STANDARD PIPE SPECIFICATIONS

Steel:

Standard Specifications for Welded and Seamless Steel Pipe (B36.1-1934) (A.S.T.M., A53-33)
Tentative Standard Specifications for Lap-welded and Seamless Steel Pipe for High-temperature Service (B36.3-1934) (A.S.T.M., A106-34T)
Tentative Standard Specifications for Electrical Fusion-welded Steel Pipe, 30 in. and over (B36.4-1934) (A.S.T.M., A134-32T)
Tentative Standard Specifications for Electrical Resistance-welded Steel Pipe (B36.5-1934) (A.S.T.M., A135-32T)
Tentative Standard Specifications for Forge-welded Steel Pipe (B36.6-1934) (A.S.T.M., A136-34)
Tentative Standard Specifications for Lock-bar Steel Pipe (B36.7-1934) (A.S.T.M., A137-34)
Tentative Standard Specifications for Riveted Steel and Wrought-iron pipe (B36.8-1934) (A.S.T.M., A138-34)
Tentative Standard Specifications for Electrical Fusion-welded Steel Pipe, 8 to, but not inc., 30 in. (B36.9-1934) (A.S.T.M., A139-34T)
A.S.T.M. Specifications for Steel Suitable for Welding A151-33
A.P.I. Line Pipe Specification Standard No. 5-L
A.P.I. Pipe Specification Standard No. 5A

Wrought Iron:

A.S.M.E. Boiler Code Specifications for Welded Wrought-iron Pipe (Des. S-19)
A.S.A. Specifications for Welded Wrought-iron Pipe (B36.2-1934) (A.S.T.M., A72-33)
Federal Specifications for Pipe; Wrought-iron, Welded, Black and Galvanized (WW-P-441)
U. S. Navy Dept. Specifications for Pipe, Iron, Wrought (44-P-11c)
Am. Ry. Assoc. Mech. Div., Specifications for Welded W.I. Pipe
A.P.I. Standards for Line Pipe (No. 5-L)
A.P.I. Standards for Casing, Drill Pipe, and Tubing (No. 5-A)
Am. Standards Assoc. Specifications for Riveted Steel and Wrought-iron Pipe (B36.8-1934) (A.S.T.M., A138-34)

Flanges:

American Standards for Cast-iron Flanged Fittings (B16b-2 for 25-lb., B16a for 125 lb. B166 for 250 lb.)
American Standards for Steel Flanged Fittings and Companion Flanges (B16e-1932)
American Standards for 800-lb (Hydraulic) Cast-iron Flanged Fittings

Bolting Materials:

A.S.T.M. Standard Specification A96-27
American Tentative Standards for Wrench-head Bolts and Nuts and Wrench Openings

Pipe Dimensions. See Flow of Fluids (Sec. 6), p. 869.

Pipe Joints. See Sec. 6, p. 922.

Expansion of Pipe. One of the most difficult problems in the design of piping systems is proper provision for expansion and contraction owing to temperature changes. If a pipe is under no stress when cold, and the temperature is increased, it will increase in length an amount that cannot be neglected. A table indicating the expansion per 100 ft. pipe of various materials and for different temperature ranges is given in Sec. 6, p. 870.

FIG. 40.—Typical butt weld made by either electric arc or acetylene process.

For low-pressure work, expansion may be taken care of by inserting expansion joints in the piping system. For high-pressure work, and, in general, the main steam lines and boiler feed lines, expansion is provided for in the flexibility of the piping system itself. This flexibility may be obtained by using standard types of expansion bends, or by providing loops made out of standard fittings.

The calculation of piping-system flexibility and of the stress encountered due to pipe expansion is beyond the scope of this handbook. This subject has been treated, at length, in various A.S.M.E. papers and by Walker and Crocker ("Piping Handbook," McGraw-Hill, New York, 1931).

FIG. 41.—V-type chill sleeve. Sleeve inside joint prevents "icicles" from welding but reduces internal diameter.

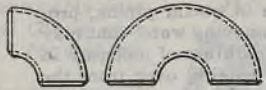


FIG. 42.—Fittings for use with welded pipe.

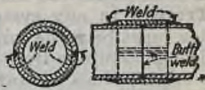


FIG. 43.—Sleeve line weld made by first butt-welding the pipe ends and then reinforcing with a split sleeve, which minimizes the stress on the butt weld.

Valves*

Valve Materials.† Valves for working pressures up to 250 lb. per sq. in. are commonly made of cast iron. Valves of this material should not be used with temperatures exceeding about 450°F. For pressures up to 1350 lb. per sq. in., valves are made with steel bodies, both cast and forged. Some manufacturers use cast chrome-nickel steel for valves used with high-temperature high-pressure steam. In power-plant practice, bronze or brass valves are used only in the small sizes and then mostly for instrument piping.

Various alloy steels are available for valve parts, such as seats, disks, and stems, which are subject to corrosion and erosion. The problem in seating materials is fivefold: (*a*) resistance to corrosion from salts in steam or water and from included air or CO_2; (*b*) resistance to erosion, especially for the flow of wet and dirty steam; (*c*) good seating-metal qualities, to avoid galling when seat and disk ride over each other; (*d*) maintenance of high strength at high temperature; (*e*) avoidance of distortion.

Acceleration tests on seats and disks, set 0.003 in. apart and blown with 250-lb. per sq. in. steam for about 350 hr., showed that the following metals resisted erosion in about the order given beginning with the most resistant:

1. Nickel-chromium alloy (20 per cent nickel, 75 per cent chromium).
2. Stainless steel (14 per cent chromium).
3. Monel and forged Everbright (30 per cent nickel).
4. Cast nickel; copper alloys (25 to 30 per cent nickel).
5. High-tin bronze (12 per cent tin).

Bronzes give satisfactory strength and resistance to corrosion for temperatures up to 550°F., especially the nickel bronze, but show high erosion and low strength for higher temperatures. Monel metal had for a number of years a monopoly for high-temperature work; but while it has good strength at high temperatures, and excellent resistance to corrosion, it is not satisfactory in resistance to erosion, and it galls badly.

Distortion has much to do with seat tightness and durability. It is a function almost entirely of valve design, compensated by the fact that all

---

* See Sec. 6, p. 935.

† See pp. 2091 *ff*.

present materials, monel, stainless steel, and nickel-chromium alloys, have coefficients of expansion exceeding those of cast or forged low-carbon steel by 25 to 45 per cent. Consequently some distortion is bound to occur with a valve that is assembled cold and heated to 600° to 700°F., aside from the additional distortion that may come from pressure effects on the valve body.

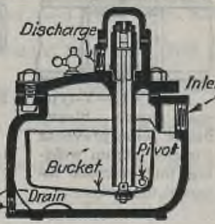


FIG. 44.—Bucket trap.

Traps. In any system of steam piping, provision must be made for removing water that may collect in the pipe from separation of moisture in the steam, slugs of water coming over from the boiler, or condensation due to heat loss through the pipe covering. This applies equally to superheated as well as saturated steam lines, as it is entirely possible for water to flow along the bottom of a pipe in the presence of superheated steam.

Open blows are generally provided to remove the condensate that occurs in a cold pipe line when being warmed up prior to cutting in for regular operation. Steam traps are usually used to remove the condensate that forms when the line is in service.

Traps used for this service are usually of the bucket or ball-float type. Figure 44 is a cross section of a bucket trap. When the trap becomes full of water the bucket drops, thereby opening a valve which allows the water to be discharged. When the trap becomes empty the bucket lifts, thus closing the valve and preventing the escape of steam. By-passes should be provided around traps so that the pipe line may be drained while the trap is being repaired. Discharge from traps is generally collected in an open feed-water heater or in a receiver which in turn drains to a heater or water-storage sump.

The steam loop is another method of draining a piping system. In this system the drips are collected in a receiver, or "pig," forced by steam pressure to a tank above the boilers and then into the boiler drums. In high-pressure plants the trend is to eliminate traps and employ an orifice system which discharges to a feed-water heater.

AUXILIARIES

Condensers. The economies to be gained by reducing the exhaust pressure of steam engines and turbines have been indicated in the part on Prime Movers.[*] Condensers designed for service with steam engines generally do not contemplate vacuums higher than 27 in. referred to 30 in. barometer. Those for service with turbines are generally designed for a vacuum of 29 in. wherever the temperature and quantity of cooling water permit. Under favorable conditions, back pressures of less than 0.5 lb. abs. have been obtained.

Types. Condensers may be classified under two main headings: **surface condensers** in which the cooling water and condensed steam are not allowed to mingle, and **jet condensers** in which steam is condensed by direct contact with cooling water.

Surface condensers (Fig. 45) consist of a cast-iron or steel shell in which are placed tubes of $\frac{5}{8}$ to 1 in. in diameter. Cooling water is circulated through the tubes. The engine or turbine exhaust enters the condenser shell at the top, and the steam is condensed on the tube surfaces.

The tubes are held in tube sheets by screwed ferrules and packing of corset lace, fiber, or metallic packing. They may be rolled into the tube sheet at

---

[*] P. 2460.

one end and sometimes at both ends, expansion being provided for either by bowing the tubes or by flexible arrangement of tube sheet and water box. Tubes are composed of various alloys: muntz metal (60 per cent Cu, 40 per cent Zn) for reasonably pure water; admiralty metal (70 per cent Cu, 29 per cent Zn, 1 per cent Sn) for sea water.

In single-pass condensers there is a single water box at each end of the tubes. In a two-pass condenser the water boxes are divided into two compartments so that the cooling water passes through the lower bank of tubes and returns through the upper.

Heat transfer between steam and water depends upon velocity of circulating water and cleanliness of tubes, temperature difference, amount of air on steam side of tube, and extent of water blanketing. Actually the heat transfer varies considerably in various parts of the condenser. For the condenser as a whole the heat-transfer coefficient may be taken as from 270 to 500 B.t.u. per sq. ft. per mean deg. F. temp. diff. per hr. at water velocities of from 4 to 10 ft. per sec.

The square feet of surface required per kilowatt of prime mover served will vary with the steam rate of the prime mover. In general, for small units of 500 to 1000 kw., the surface required is about 3.5 to 2.75 sq. ft. per kw. In larger sizes, condensers have been built with only 1 sq. ft. of surface per kilowatt turbine capacity.

The quantity of cooling water required per pound of steam condensed varies with the vacuum and the temperature rise of the cooling water. For small units a temperature rise of 12° to 15°F. is usual; for larger units, when a higher

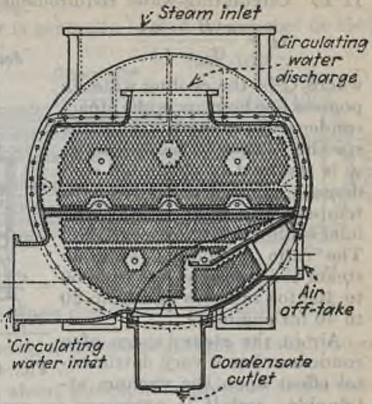


Fig. 45.—Arrangement of downflow surface condenser.

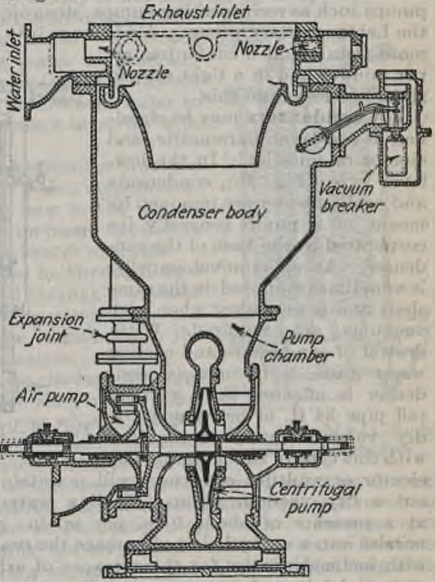


Fig. 46.—Multijet high-vacuum condenser.

vacuum is desired, the temperature rise of cooling water is reduced to 10° or 11°F. Circulating-water requirements may be calculated from the equation

$$Q = \frac{w(H - q_c)}{(t_o - t_i)}$$

where Q is the cooling water in pounds per hour, w is the steam condensed per hour, H is the specific heat of exhaust steam, q_c is the heat of the liquid condensate, and t_o and t_i are the temperatures of the outlet and inlet cooling water, respectively. The ratio of cooling water to steam condensed varies from 50 to 100 for turbines and from 20 to 40 for engines.

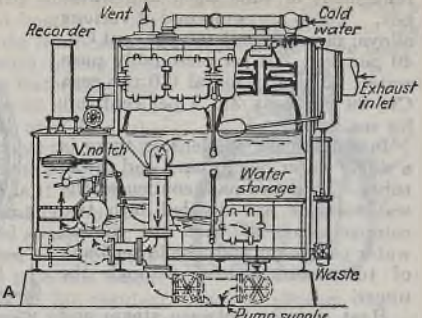


FIG. 47.—Open heater with integral flow meter.

Air in the steam space of a condenser has a very detrimental effect upon the vacuum attainable, and it is necessary to provide means for continually removing it. Air enters the condenser with the steam and leaks in through shaft seals and other joints. It is removed by various types of dry vacuum pumps such as reciprocating pumps, steam-jet air pumps and, to a less extent, the Leblanc rotary pump. Air leakage into a large condenser should not be more than about 5 cu. ft. free air per minute and in a tight system should be less than this.

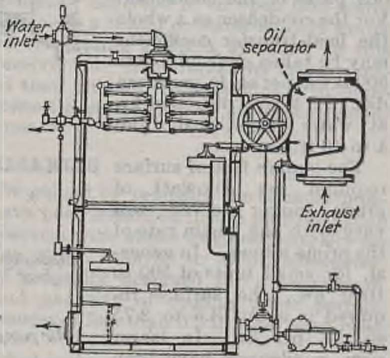


FIG. 48.—Front sectional elevation of induced-flow open feed-water heater.

Jet condensers may be classified as low-level, barometric, and ejector or multijet. In the low-level type (Fig. 46), condensate and cooling water are removed by means of a pump generally incorporated in the base of the condenser. An air-removal impeller is sometimes mounted on the same shaft and is necessary when high vacuums are required. Withdrawal of condensate and cooling water from a barometric condenser is effected by a vertical tail pipe 34 ft. or more long. A dry vacuum pump is required with this type of condenser. The ejector or multijet condenser will maintain high vacuum, 29 in. Hg, without a dry vacuum pump. Cooling water is supplied to this condenser at a pressure of about 9 lb. per sq. in. gage and is discharged through nozzles into a venturi-shaped passage the central portion of which is provided with inclined nozzles for the entrance of exhaust steam. The shape of the water inlet is such as to convert the static pressure into velocity, and the high-velocity jets force the entering exhaust steam and entrained air into a diverging tail pipe which converts the velocity into pressure in order to overcome the atmospheric pressure. Multijet condensers are suitable for prime movers up to about 10,000 kw. capacity.

Feed-water Heaters. Feed water is generally heated by exhaust or bled steam from main or auxiliary engines or by flue gas, the latter method having been discussed under Economizers.* There are two types of steam feed-water heaters—open heaters and closed heaters. In open heaters steam and water come in direct contact with each other, but in closed heaters the steam and water do not mingle, the water flowing through tubes and the steam around the tubes.

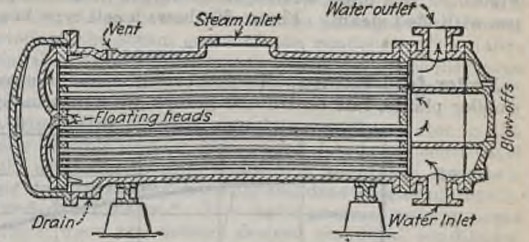


Fig. 49.—Multipass closed heater with floating heads.

Open heaters generally operate at about atmospheric pressure and a vent is provided to allow the escape of non-condensable gas. The removal of these gases constitutes one of the advantages of this type heater. Also, in open heaters, when the steam supply is not limited, no temperature head is lost as the water is heated to the saturation temperature of the steam in the heater. In closed heaters, however, the leaving water is generally from 2° to 12°F. below the temperature of the steam in the heater. Closed heaters permit feed water to be pumped through several heaters in series with but one pump.

Deaerating open heaters are frequently provided with a small condenser through which non-condensable and condensable vapors flow to the atmosphere or are removed by a vacuum pump depending upon the pressure at which the heater operates.

Open heaters may be classified as **through flow** in which all of the steam is forced through the heater irrespective of whether it will be condensed; and **induced flow** in which the flow of steam into the heater is induced by its condensation.

Figure 47 is an example of a *through-flow-type heater*. It consists of a cast-iron shell in the upper part of which are trays or pans over which the water flows in finely divided streams. Steam entering through an oil separator is directed across the falling streams of water. That part of the steam not condensed exhausts through the outlet at the top. The bottom part of the heater serves as a reservoir. Figure 48 is a cross section of an *induced-flow-type heater*. In some open heaters, nozzles take the place of the trays and break the water flow into finely divided sprays. Metering equipment and filter bed are frequently made integral with open heaters.

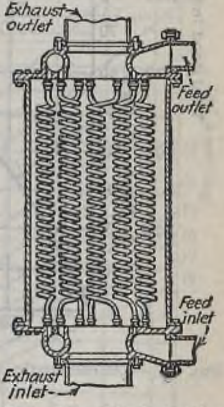


Fig. 50.—Multitube coiled closed heater.

---

* P. 2446.

Closed heaters may be either **water tube** or **steam tube,** but for power-plant service the former is in more general use. The water tubes may be straight, hairpin, or arranged in coils. They may be single- or multipass. Figure 49 is a cross section of a straight-tube four-pass heater suitable for use with bled steam. Figure 50 shows a coil-type heater.

Pumps*

Boiler-feed Pumps. Reciprocating steam pumps are often used in the smaller plants, but centrifugal pumps are now usually used for this service particularly in larger plants. The advantage of the latter type lies in the fact that it can be driven by either electric motor or steam turbine. The centrifugal boiler-feed pump is generally built in two or more stages. Total pressure is from 50 to 100 lb. per sq. in. above boiler pressure depending upon the piping and equipment, such as feed-water heaters and economizers that are installed between the boiler and the pump. The pumps usually have relatively flat characteristic curves.

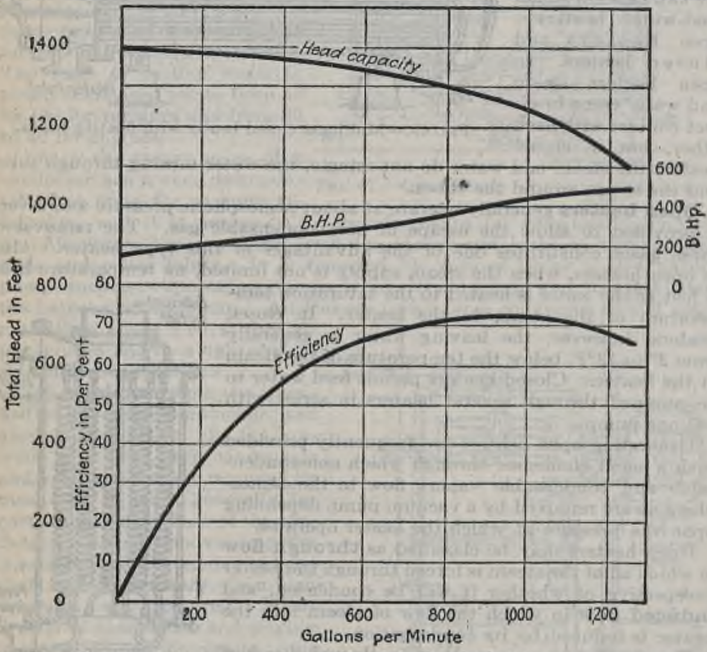


FIG. 51.—Characteristic curves of centrifugal boiler feed-water pumps—six stage, 1775 r.p.m., based on 150°F. water.

Figure 51 shows the performance curves of a typical centrifugal boiler feed-water pump.

---

* See Sec. 20.

Condensate Pumps. Centrifugal pumps are used to remove the condensed steam from the condenser hot well. They should be installed so that the water will flow to the pump under a head of 3 to 4 ft. in order to prevent cavitation. Condensate pumps usually have horizontal inlets and vertical discharge to prevent air pockets. The suction passages of the pump are frequently connected to the steam space of the condenser to equalize the pressure and so assist the flow of condensate to the pump. The discharge of the pump should be provided with a check valve to prevent water from flowing back to the condenser.

Condenser circulating water pumps, supplying cooling water to the condenser, are generally double-suction slow-speed low-head centrifugal pumps. It is frequently necessary to provide priming apparatus, such as a steam jet which removes air from the pump casing and draws water up into it.

Forced- and Induced-draft Fans.* The static-pressure requirements of the fans depend largely upon the ratings desired and local conditions. Normally, in industrial plants, the forced-draft fan will not be required to produce a static pressure of over 2.5 to 3 in. water. The pressure requirements of induced-draft fans are influenced by the heat economy equipment through which the flue gas must pass. With economizers and air preheaters the induced-draft fan may be required to develop as much as 6 to 8 in. static pressure.

From the chart (Fig. 52) can be found the weight of flue gas that an induced-draft fan will be required to handle per pound of fuel burned and for various combustion conditions. The weight of gas can easily be converted to cubic feet at the temperature of the exit gas from Fig. 53. In selecting fan capacity, an additional allowance of about 5 per cent should be allowed for air infiltration. The same chart can be used to obtain an estimate of the forced-draft-fan capacity by assuming 1 lb. less air per pound of coal than indicated in Fig. 52.

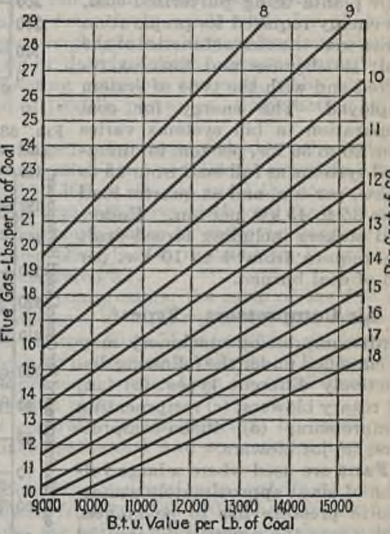


FIG. 52.—Weight of flue gas per pound of coal for various B.t.u. and CO_2 values. (*From "Dust Problems," by Dust Recovery, Inc.*)

Horsepower requirements for fan drives may be calculated from the equation

$$\text{Horsepower} = \frac{5.2Qh}{(33{,}000 \times E)}$$

where Q is cubic feet of gas per minute; h, static pressure developed by the fan in inches of water; and E, efficiency of fan based on static pressure. Fan efficiencies vary from 50 to 65 per cent.

Fan Types. Fan rotors are available with forward- or backward-curved blades, and with radial or straight blades. The effect of blade curvature on

---

* P. 2448.

fan characteristics is shown in Fig. 54. Generally a fan having a rising horsepower characteristic such as obtained with forward-curved blades is not suitable for forced-draft service but can be used for induced draft. Fans may be obtained with single or double inlets and with discharges arranged vertically up or down, or horizontal.

Power for Auxiliaries. The total energy required to operate power-plant auxiliaries varies from about 3 to 8 per cent of the output, depending to some extent on the operating pressure, size of plant, type of auxiliaries, method of coal handling, and type of combustion employed.

In plants using pulverized coal, the energy required for preparation varies with the characteristics of the coal: its dryness and fineness required and with the type of system employed. The energy for coal preparation in bin systems varies from 20 to 30 kw. per ton, in direct-fired systems at full load from 15 to 20 kw. per ton, and at quarter load from 35 to 45 kw. per ton. Underfeed stokers including forced-draft fan require from 4 to 10 kw. per ton of coal burned.

Air Compressors—Types*

Air-compressing machinery may be classified under the following distinctively different types: (*a*) fan; (*b*) rotary blowers; (*c*) reciprocating compressors; (*d*) turbo-compressors; (*e*) jet blowers.

Fans are used where a large volume of air at approximately atmospheric pressure has to be moved, and where the pressure differential is small, seldom exceeding 15 to 16 in. of water. Fan characteristics have been covered previously in this section.

Rotary blowers are designed to deliver air in the pressure range of 5 oz. to 12 lb. per sq. in. They

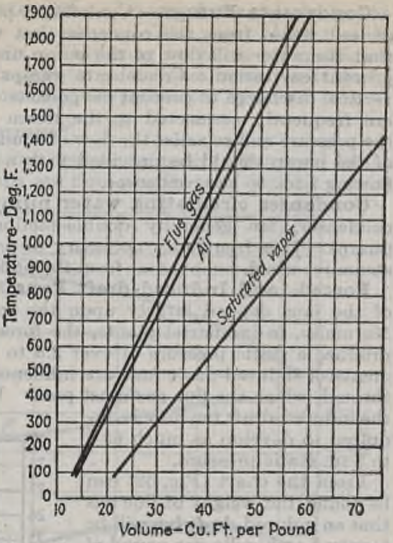


FIG. 53.—Specific volume of flue gas, air, and water vapor at various temperatures.

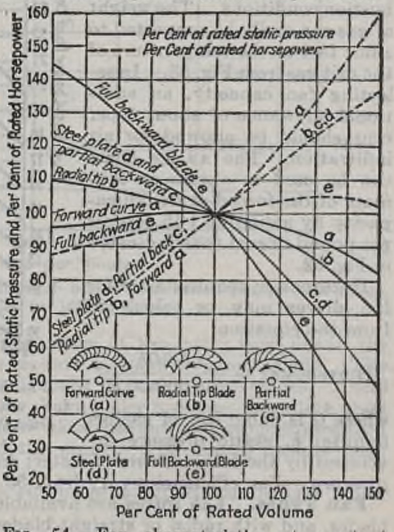


FIG. 54.—Fan characteristics at constant speeds. (*Laurence, Trans. A.S.M.E.*, 1922.)

---

* See Sec. 20.

range in capacity from 5000 to 15,000 cu. ft. per min., and in speed from 375 to 800 r.p.m. for the smaller sizes down to 160 and 220 r.p.m. for the larger sizes.

Blowers of this type are usually preferred to a centrifugal fan for cupola service, because their positive displacement action maintains the air supply, even if the cupola becomes clogged. They may be arranged to develop constant-pressure or constant-volume characteristics. Efficiencies vary between 80 and 90 per cent.

Rotary blowers differ greatly in design, but they generally embody two blades, impellers, or displacers driven by gears and rotating in opposite directions. These impellers are housed in a casing. In rotating they trap between them and the casing a volume of air which is carried from the inlet to the discharge side of the casing.

Reciprocating compressors are used for pressures ranging from 10 lb. per sq. in. up to 2000 lb. per sq. in. Compressor capacities generally range from a few cubic feet to 5000 cu. ft. per min., though some very large compressors for blast-furnace service have been built. Turbine compressors are now very largely supplanting reciprocating-type compressors in this service.

Reciprocating compressors are constructed very similar to steam engines, excepting that the valves are generally of the automatic type as distinguished from mechanically operated valves. For small compressors, poppet valves of the single-beat-cup type are most extensively used, while for all other services some one of the various light-weight plate or strip valves is employed. Compressor cylinders may be single or double acting. For pressures up to 500 or 600 lb. per sq. in., double-acting cylinders are used, while for very high pressures, in the neighborhood of 1000 lb. per sq. in., single-acting cylinders give the best service.

For low pressures, the compression is generally accomplished in a single stage. For large-sized compressors, the compression ratio per stage is generally 3 to 4. For the relatively smaller size (50 to 300 cu. ft. per min.) the compression ratio may be 7 to 8. When two-stage compression is employed, the air is generally cooled between the first and second stages in order to improve the efficiency of the compressor. After-cooling is also resorted to, in order to reduce the moisture content of the delivered air.

Volumetric efficiency of piston-type compressors, the ratio of the free air volume actually taken into the compressor at intake pressure and temperature to the piston displacement, varies from 50 to 75 per cent for single-stage 100-lb. machines and 80 to 90 per cent for multistage machines. Compression efficiency (based on adiabatic compression) varies from 74 to 90 per cent, largely depending upon the compression ratio.

Reciprocating compressors are generally water-jacketed, in order that compression may normally approach isothermal and so improve the compression efficiency.

Turbo-compressors are very similar in construction to centrifugal pumps. They are designed for volumes ranging from 1000 to 100,000 cu. ft. per min. and for pressures up to about 125 lb. per sq. in. The usual size for a single-stage compressor is from 1000 to 25,000 cu. ft. per min. One of the largest turbo-compressors now in operation has a capacity of 100,000 cu. ft. per min. at a pressure of 215 lb. per sq. in. Efficiencies of 75 to 80 per cent are obtained with this type of compressor.

Jet blowers operate on exactly the same principle as feed-water injectors and are made in capacities ranging from 100 to 600 cu. ft. per min. They require steam pressures of from 75 to 90 lb. per sq. in. or water pressure of from 45 to 75 lb. per sq. in.

The horsepower required to compress air in a single-stage compressor is

$$\text{Horsepower} = \frac{144}{33{,}000} \times \frac{n}{n-1} \times P_1V_1\left[\left(\frac{P_2}{P_1}\right)^{\frac{n-1}{n}} - 1\right]$$

where V_1 represents the volume of gas compressed per minute; P_1 and P_2, the pressure in pounds per square inch absolute; n is a constant. For adiabatic compression of air n is equal to 1.4; for isothermal compression $n = 1$. Actual compression, because of jacket cooling, lies between these extremes, and n is generally taken at 1.3.

POWER-PLANT COSTS

Any figures on power-plant costs, either investment or operating, are likely to be misleading unless interpreted in the light of all local conditions.

Table 11. Distribution of Boiler-plant Costs*

Two similar boiler plants, one stoker fired, the other pulverized-coal fired and both designed by the same engineer. Both boilers were of the same make, inclined-water-tube type, and each rated at approximately 1000 hp.

Components of plant	Per cent of total cost	
	Plant 1	Plant 2
Excavation	1.46	1.32
Concrete reinforced foundation below grade	0.67	0.61
Concrete setting walls above grade	2.90	
Concrete floors, basement, and boiler room	1.35	1.22
Boiler delivered and erected, including slag screen, rear-wall protection, and front water wall	23.10	20.8
Drake armor clad, for rear-wall water tubes	1.51	
Ashpit doors	0.22	0.39
Supporting steel for boiler and air-cooled walls	4.55	6.15
Superheater	5.62	5.07
Air-cooled wall refractory and brackets	2.38	4.25
Boiler setting including all refractories except air-cooled wall	9.43	9.45
Stoker: 29 tuyères, 10 retorts, double 14-in. crusher rolls; projected area 240.8 sq. ft.; steam-turbine, stoker drive direct connected; crusher rolls driven by hydraulic motor; overspeed governor	16.70	
Non-Seg coal distributors, three	1.69	
Pulverized-coal equipment: 2 pulverizers, 6000 lb. per hr. each; 2 low-pressure coal burners; coal piping from pulverizers to burners; motors direct connected to pulverizers	....	14.50
Soot blowers	2.25	2.03
Instruments: steam flow meter, CO_2 meter, multipointer draft gage, flue-gas temperature recorders	1.80	1.62
Combustion control	1.24	1.42
Liquid-level gage	0.21	0.19
Soot-pit gates and metal hoppers	1.01	0.91
Steam purifier and deconcentrator	1.07	0.96
Feed-water regulator	0.21	0.19
Piping and insulation	5.62	5.07
Platforms and stairs	1.69	1.52
Blast fan, turbine driven, 35,000 cu. ft. per min., 6-in. w. g.	3.71	3.35
Flue	2.75	2.49
Blast-duct work, including return air from air-cooled walls		
Blast-duct work, not including return air from air-cooled walls	2.02	1.11
Total cost	$88,910	
Induced-draft fan, 47,000 cu. ft. per min., 65 b.hp., motor driven	....	2.64
Steel supports	....	0.61
Economizer	....	11.68
Extra flue	....	0.61
Total cost	....	$98,660

The economizer and necessary fan increase the guaranteed efficiency at 200 per cent of rating from 78 to 83.6 per cent, and this shows a gross saving of about $8000 per year.

* From article by Samuel M. Green, *Power*, Nov. 4, 1930.

Table 12. High-pressure Plant Costs*

Plant built in 1934

Rating of plant, kw.	75,000
Number of generators	1
Number of boilers	1
Capacity, lb. steam per hr	690,000
Pressure, lb. per sq. in	1,390
Temperature, °F	835
Fuel	Pulverized coal
Operating costs excluding fixed charges, mills per kw	2.05
First Costs Dollars per kw.	
Land and site preparation	5.24
Building and foundation	21.30
Condenser water works	1.83
Fuel unloading and storage	15.50
Ash handling	0.75
Fuel preparation	3.44
Switch gear and station wiring	8.05
Boiler plant	13.22
Draft system	1.48
Feed-water system	3.00
Piping	3.01
Heat-recovery equipment	1.79
Generator air coolers	0.24
Main condensers	2.69
Turbo-generator	17.20
Turbine auxiliaries	0.52
Turbine foundation	0.92
Total including miscellaneous	100.18

Plant has steam reheaters, air preheaters, combustion control duplex feed pumps, five-stage feed heating.

* From *Electrical World*.

Table 13. Analysis of Costs for Industrial Boiler Plant*

	Dollars per 10 Sq. Ft. of Boiler Surface
Foundations for building equipment	8–12
Building and fuel bunkers	20–40
Boilers, superheaters, soot blowers (erected)†	20–35
Economizer (erected)†	10–20
Air heaters (erected)	10–15
Settings	6–10
Coal firing and forced-draft equipment‡	12–18
Coal-handling equipment	5– 7
Ash-handling equipment	4– 5
Breeching and chimney	7–10
Feed pumps	2– 3
Piping	10–15
Water-treating plant	6– 9
Motors, wiring, and lighting	6– 8
Combustion control	1– 3

* From an article by H. Bleibtreu, *Power*, Oct. 6, 1931.

† Boiler pressures of 250 lb. and less.

‡ Including unit mills in case of powdered coal.

Table 14. Boiler-plant Costs

Purchased 1938–1939

	Number of boilers				
	1	2	3	4	1
Boiler capacity, lb. per hr. each	450,000	400,000	70,000	75,000	70,000
Pressure, lb. per sq. in	1350	900	415	400	350
Temperature, °F	905	—	600	—	—
Water walls	....	✓	✓	✓	✓
Superheater	....	✓	✓	✓	
Setting	✓		✓		
Pulverizers	✓	✓	✓		
Burners	✓	✓	✓	oil	
Stoker	....	....	....	✓	
Air preheater	✓	✓			
Economizer	✓	....	✓		
Piping	S.B.				
Misc	S.H. ducts	Fans			
Total cost installed	$435,000	$320,000	$167,000	$297,000	$47,000
Cost per lb. steam	$0.97	$0.80	$0.80	$1.00	$0.67

Check marks indicate equipment included in cost. S.B. = soot blowers, S.H. = soot hoppers.

Load factor, size of units, and price of fuel will vary operating costs over a wide range. In view of this, such figures as are given may be taken at best as only a guide.

Total investment costs in large steam central stations vary from $85 to $110 per kw. of installed capacity, depending upon the plant location, size of units, and the refinements in equipment. In a plant designed for base-load service, or to operate with high-priced fuel, the operating savings through the installation of more efficient equipment will more than offset the increased charges on the investment. Conversely a plant designed to operate on a low-load factor, or with cheap fuel, may not warrant the additional investment.

In industrial plants the unit investment costs vary over a wide range. The smaller size of units tends to increase the unit cost of individual equipment, but this in turn is offset in view of fewer refinements, especially in the electrical end, which tends to reduce the total unit costs. Moreover, the site and building costs are usually lower than in the case of central stations.

The cost of steam turbines and generators, with exciter, in sizes commonly employed in industrial plants will range from $25 to $35 per kw. for condensing and non-condensing units and slightly more for bleeder-type units. Very large turbine-generators, such as used in large central stations, will run $15 per kw., or lower.

Operating Costs. In the modern and well-managed industrial plant, steam costs should range from 22 to 35 cts. per 1000 lb., depending on the cost of fuel, size of units, load factor, and labor cost. The electrical cost per kilowatt-hour, exclusive of fixed charges, will depend upon the use of exhaust steam for process. This may vary from half a cent to two or more cents per kilowatt-hour. In the large modern central station, the cost at the switchboard may be as low as 3 mils per kw.-hr., but this is only a fraction of the average delivered cost when fixed charges, distribution, and overhead are considered. The best of such stations are operating on approximately 1 lb. of coal per kilowatt-hour and the average for the year 1932 was 1.5 lb. per kilowatt-hour, according to statistics of the U. S. Geological Survey.

Table 15. Volume Structural-steel Weights, and Cost of Boiler Houses with Coal Bunkers

Types	Cu. ft. building space per 10 sq. ft. boiler surface	Structural steel lb. per 10 sq. ft. boiler surface	Cost of building and coal bunkers above foundation	
			Per cu. ft. building space	Per 10 sq. ft. boiler space
1. Two 5120-sq. ft. straight-tube boilers, 3 unit mills, overhead bunkers	118	197	$0.22	$26.50
2. Four 6000-sq. ft. 4-drum Stirling-type boilers, traveling-grate stokers, overhead bunkers	158	280	0.25	39.30
3. Six 8140-sq. ft., 3-drum Stirling-type boilers, traveling-grate stokers and blast-furnace gas, overhead bunkers	150	270	0.16	23.55
4. Six 9120-sq. ft. boilers, economizers, 8 unit mills, 1 traveling-grate stoker, overhead bunkers	193	269	0.17	31.95

Table 16. Details of Construction Costs, Diesel Engine Plant*

(Board of Public Works, Grand Haven, Mich. Hamilton & Weeber, Engineers, Grand Rapids, Mich.)

		Total Including Labor and Material
1. Building		$38,105.92
Brick, concrete, and steel construction, face-brick exterior with stone trim, 4,271.7 sq. ft. floor area, 171,936 cu. ft. contents including basement.		
Building cost per sq. ft.	$8.925	
Building cost per cu. ft.	0.2216	
(1600 kw.) bldg. cost kw. plant capacity—present	23.81	
(2600 kw.) bldg. cost per kw. plant capacity—future	14.65	
2. Diesel Engine Units and Foundations:		
Two—1140 b.hp., 6-cylinder, 200-r.p.m. model VB, 4-cycle solid-injection, De La Vergne engines with auxiliary compressed-air starting equipment, and two 800-kw. Elliott generators with two 17-kw. direct-connected exciters:		
Equipment contract		$101,979.00
Erection		6,596.30
Foundations:		
Cement		1,131.21
Gravel		897.66
Form lumber		270.17
Labor		1,441.39
Equipment rental		280.51
Pyrometers		547.96
Insurance		350.00
		$113,494.20
3. Switchboard and Station Power:		
Westinghouse contract		$10,614.51
Wiring to generators and exciters		1,926.15
Station auxiliary power lines		2,586.35
Brown-Boveri voltage regulator		1,285.00
		$16,412.01

---

* From *Power*, Sept. 22, 1931.

Table 16. Details of Construction Costs, Diesel Engine Plant*—
(*Continued*)

	Total Including Labor and Material
4. Traveling Crane	
5-ton, 55-ft. span hand-operated	$1,681.38
(Crane erection incl. in building contract).	
5. Fuel-oil System:	
100,000-gal. steel storage tank, contract	$1,795.00
Excavation	2,059.60
Foundation and sidewalks	1,840.20
Piping	1,873.66
D-50 Hydroil purifier—contract	1,875.00
50-gal. per min. fuel unloading pump	225.00
Tank gages	307.72
	$9,976.18
6. Lubricating-oil System:	
D-50 Hydroil purifier—contract	$1,875.00
Piping	647.25
Tank gages	135.72
Two oil coolers	1,076.00
	$3,733.97
7. Air Intake:	
Two 6-unit air filters	612.50
Piping	1,571.55
	$2,184.05
8. Exhaust System:	
Two De La Vergne 5-compartment mufflers	$835.00
Exhaust piping	1,433.67
	$2,268.67
9. Cooling-water System:	
Intake and wells	$2,832.90
Equipment rental	138.38
Elevated, 15,000 gal., steel storage tank on 50-ft. tower—contract	2,975.20
Tank foundation	323.82
King-Seeley tank gage	139.65
Pumps, one 400- and one 600-gal. per min.—contract	1,734.98
Piping and miscellaneous material	1,074.53
	$9,219.46
10. Building Heating Equipment:	
One—916,000 B.t.u. unit heater	$523.43
Piping and office radiation	277.69
	$801.12
11. Substation:	
One Delta-Star substation tower with bus and switching equipment	$2,056.54
Foundations	310.30
Conductors and erection	2,529.49
Transformers, 2000-kva., 2300-volt auto and one 833-kva. 2300/6600-volt power	3,966.35
	$8,862.68
12. Engineering:	
Hamilton & Weeber, design and consulting engineers	$8,827.87
Local inspections	3.20
	$8,831.07

* From *Power*, Sept. 22, 1931.

Table 16. Details of Construction Costs, Diesel Engine Plant*—(*Concluded*)

Summary	Total Including Labor and Material
1. Building	$38,105.92
2. Diesel engine units and foundations	113,494.20
3. Switchboard and station power	16,412.01
4. Traveling crane	1,681.38
5. Fuel-oil system	9,976.18
6. Lubricating-oil system	3,733.97
7. Air intake	2,184.05
8. Exhaust system	2,268.67
9. Cooling-water system	9,219.46
10. Building heating equipment	801.12
11. Substation	8,862.68
Subtotal	$206,739.64
12. Engineering	8,831.07
Total construction cost	$215,570.71

* From *Power*, Sept. 22, 1931.

Table 17. Diesel Engine Bids for Greenville, Tex., Municipal Plant*

	Make of engine					
	American Locomotive	Bush-Sulzer	Bush-Sulzer	Hamilton M.A.N.	Nordberg	Nordberg
Horsepower	1800	1800	2400	2250	1875	2250
Type of engine	Two-cycle mech. injection	6-D-HB-27	8-D-HB-27	Two-cycle single acting	Two-cycle mech. injection	Two-cycle mech. injection
Actual capacity, k.w.	1250	1250	1690	1500	1320	1585
R.p.m.	277	240	240	240	225	225
Number of cylinders	10	6	8	6	5	6
Mean effective pressure, lb. per sq. in.	71	61.4	61.4	60	65.7	65.7
Method of scavenging	Reciprocating pump attached	Blower port	Blower port	Loop	Pump	Pump
Gal. per min. 90°F. cooling water	450	360	480	350	310	375
Intake-air filter	Yes	Yes	Yes	Yes	Yes	Yes
Cost per hp	$53.90	$57.03	$53.43	$55.11	$59.00	$57.94
Cost per kw	$77.63	$82.13	$75.87	$82.67	$83.70	$82.25
Total installed cost	$97,036	$102,664	$128,232	$124,000	$110,614	$130,337

* From *Diesel Power*, July, 1937.

MECHANICAL POWER TRANSMISSION

BY WILLIAM STANIAR

BELTING

Power belting is one medium of supplying a given amount of power at a certain cost per unit of time, which if properly selected and applied should result in the transmission of power at a low cost per horsepower per year over a long period of useful service.

There are four general classes of power belting: **leather, rubber** or Neoprene, **stitched canvas,** and **hair.** The characteristics, properties, and uses of these classes are generally applicable to all types and brands manufactured. Therefore, based on the fact that the transmission of power by belting covers such an extensive field, it is essential for economy and efficiency that the proper belt as to type, width, and thickness be employed for a given installation.

The theory of belt transmission is ably discussed in such mechanical handbooks as those of Marks and Kent, but in actual practice, type, width, and thickness are most important. Therefore, these factors will be treated in a condensed form.

Leather Belting

The best leather belting comes from the hides of 3- or 4-year old steers and is generally cut from a slab known as the *butt bend*, 44 to 48 in. from the root of the tail toward the head and 24 in. on each side of the backbone of the animal. The location of the cut in its relation to the backbone is most important. The process of building the belt is a series of operations which include cutting to required widths, matching ends for thickness, scarfing and tapering the ends of the strip, squaring of the scarfed ends, and, finally, the cementing and the inspection.

There are numerous grades and brands of leather belting, but, essentially, there are three distinct types, namely, (1) **regular oak tanned,** cemented with a non-waterproof glue if a non-waterproof belt is desired, or cemented with a waterproof pyroxylin cement if a waterproof belt is desired, (2) **mineral retanned**, and (3) the combination of **oak tanned and mineral retanned.**

Regular Oak Tanned.

Characteristics. Made in 1 ply, 2 ply, and 3 ply. Known as singles, doubles, and triples. 4 ply is very rare.

Thickness of singles, $^{10}/_{64}$ to $^{14}/_{64}$ in.; doubles, $^{1}/_{4}$ to $^{3}/_{8}$ in.; triples, $^{15}/_{32}$ to $^{36}/_{64}$ in. Widths of singles, $^{1}/_{8}$ to 10 in.; doubles, 1 to 72 in.; triples, 24 to 72 in.

Fabricated with a hot non-water-resisting glue. It does not possess flexibility when new. Ultimate tensile strength, 4000 to 6500 lb. per sq. in. It has low stretch. Coefficient of friction: new oak, 0.27 to 0.45; well-worn oak, 0.35 to 0.60. It will take any type of metallic fastener and can be made endless at the point of application if necessary.

Uses. Singles: (1) Small power applications; (2) light machine-tool driving; (3) on pulleys 3 and 4 in. diameter and over; (4) on light shifting drives; (5) where moisture or acid is not present.

Doubles: (1) medium and heavy power applications; (2) all manner of machine-tool driving; (3) if light double, on pulleys 6 in. diameter and over; (4) if regular double, on pulleys 10 in. diameter and over; (5) on shifting drives; (6) where moisture or acid is not present.

Waterproof Cemented Oak-tanned Belting.

Characteristics. This type is made in similar plies, thicknesses, and widths to, and possesses all the characteristics of, non-waterproof oak, but it has the advantage of being water- and moisture-proof. Its cost is approximately 10 per cent higher than regular oak, but this additional expenditure is insurance against premature destruction. This type is steam-proof and can be operated immersed in water if necessary.

Uses. (1) All places where regular oak is applicable; (2) in moisture, steam, and sloppy conditions; (3) not in the presence of acid.

Mineral-retanned Leather Belting.

Characteristics. This type is an entirely different leather from the regular oak-tanned variety and, therefore, has definite characteristics of its own. It is made in similar thicknesses and widths to regular oak, but its ability to transmit power is greater, ply for ply and width for width. It possesses a tensile strength of approximately 6000 lb. per sq. in. and has a coefficient of friction of 0.60 to 1.10. It possesses low "stretch." It has extreme flexibility and, therefore, is efficient on high-speed, small pulley work. It is fabricated with a waterproof cement and, therefore is impervious to the action of moisture. It is moisture-, steam-, and heat-proof and will resist the action of corrosive acids for a considerable period of time. It will take any type of metallic fastener and can be made endless at the point of application. It can be repaired, cut down, rebuilt, and salvaged in a manner similar to oak-tanned leather.

Uses. **Singles:** (1) High-speed motor drives up to $7\frac{1}{2}$ hp., 1200 to 1800 r.p.m.; (2) high-speed spindle drives of various machine tools such as internal and external grinders; (3) mule-pulley installations on small single- and multiple-drill presses; (4) on pulleys as small as $1\frac{1}{4}$ in. diameter; (5) pivoted motor-base drives when made endless.

Doubles: (1) High-speed motor drives, 10 hp. and up, 720 to 1200 r.p.m.; (2) high-speed pulleys 6 in. diameter and over; (3) high-speed spindle drives of the larger machine tools; (4) automatic idler and pivoted motor-base drives when made endless.

Combination Oak- and Mineral-retanned Leather.

Characteristics. This type of leather belting is made, as the name implies, of a combination of the regular oak-tanned and mineral-retanned types joined flesh side to flesh side with a cement insoluble in water. It is made in doubles only. It is made in similar widths to regular oak. It is specially manufactured for shifting, step-cone, and flanged-pulley drives because the oak tannage resists the transverse crumpling action on the flexible retanned type of leather caused by the action of shifter forks and pulley flanges. It is moisture- and steam-proof, but will not resist the action of corrosive acids. It will take any style of metallic fastener and can be made endless at point of application. Its repair and salvage values are similar to those of all types of leather.

Uses. (1) Heavy slow-speed loads, such as 200 to 500 ft. per min.; (2) shifting and loose pulley drives; (3) step-cone and flanged-pulley drives; (4) pulleys should not be under 8 in. diameter; (5) heavy-duty milling machines or lathe work.

Formulas for Leather Belting. Where T equals allowable effective pull; W, width of belt in inches; and S, speed of belt in feet per minute.

Single Oak Leather Belting. Heavy single, $\frac{12}{64}$ to $\frac{14}{64}$ in. thick. Allowable effective pull T in tight side per inch of width = 45 lb.

Formula:

$$\text{Horsepower} = \frac{TWS}{33{,}000}$$

Double Oak Leather Belting. Medium double, $\frac{5}{16}$ to $\frac{11}{32}$ in. thick. Allowable effective pull T in tight side per inch of width = 65 lb.

Formula:

$$\text{Horsepower} = \frac{TWS}{33{,}000}$$

Single Special Tannage Leather Belting. Heavy single, $\frac{12}{64}$ to $\frac{14}{64}$ in. thick. Allowable effective pull T in tight side per inch of width = 50 lb.

Formula:

$$\text{Horsepower} = \frac{TWS}{33{,}000}$$

Double Special Tannage and Combination of Oak and Special Tannage Leather Belting. Double, $\frac{5}{16}$ to $\frac{11}{32}$ in. thick. Allowable effective pull T in tight side per inch of width = 70 lb.

Formula:

$$\text{Horsepower} = \frac{TWS}{33{,}000}$$

Atmospheric Effects on Leather Belting. Regular Oak Non-waterproof Cemented: An atmosphere charged with steam or moisture causes separation of the lap and plies because of its action on the cement or glue. An atmosphere charged with the mists or vapors of the corrosive acids attacks the leather causing rapid deterioration.

Regular Oak Waterproof Cemented: An atmosphere charged with steam or moisture has no effect because in this type the leather is dressed and the pyroxylin cement is impervious to the action of moisture. An atmosphere charged with the mists or vapors of the corrosive acids attacks the leather regardless of the dressing causing rapid deterioration.

Special Tannage Waterproof Cemented: An atmosphere charged with steam or moisture has no effect on this type because of the special tanning of the leather and the pyroxylin cement. An atmosphere charged with the mists or vapors of the corrosive acids attacks this type but is slow in its action.

Combination of Oak and Special Tannage: This type is waterproof due to the pyroxylin cement, but the oak ply will not resist the action of the mists and vapors of the corrosive acids.

Leather-belting Dressings. There are numerous cheap and low-grade stick and semiliquid dressings on the market most of which contain resin, tar, and graphite. All such ingredients are injurious to leather and also to its power-transmitting ability; therefore caution should be exercised in their use. Belting of any type does not pull its load by being stuck to the pulleys. It functions through its own natural frictional grip.

Leather belting requires periodical lubrication or dressing because the heat generated by the transmission of power is absorbed by the belt, resulting in a drying out of the cod oil and beef tallow used in the currying process. In view of this, a dressing should contain similar ingredients so that penetration and lubrication of the fibrous structure will result. Reputable leather-belting manufacturers supply dressings containing the above currying materials which can always be employed to an advantage on oak leather belting whether waterproof or non-waterproof. Special tannage leather requires a special dressing which is also supplied by the respective manufacturers.

Under normal operating conditions leather belting should be dressed every two or three weeks, and the most effective results are obtained by applying the dressing to the outer ply. This allows the material to work its way through the leather and, therefore, prevents an accumulation on the pulley side of the belt.

The Effect of Centrifugal Force. The action of centrifugal force creates in a belt **centrifugal tension** which is in addition to any other existing tension. This centrifugal tension is caused by the action of the different portions of the belt upon the belt itself based on the speed in feet per minute. However, in the average industrial belt drive, this action is not of sufficient moment to cause anxiety because the average horsepower formula takes this factor into consideration. The following formula may be used to ascertain the centrifugal tension in a belt:

$$T_c = \frac{WS_2}{116{,}000}$$

where T_c is centrifugal tension in pounds; W, weight of belt in pounds per foot; and S = speed in feet per minute.

Horsepower Losses Due to Centrifugal Tension

Speed of Belt, Ft. per Min.	Percentage off Rated hp.
1,000	1
2,000	4
3,000	8
4,000	15
5,000	23
6,000	34
7,000	46
8,000	60
9,000	76
10,000	95

Notes: At 10,000 ft. per min. centrifugal tension almost equals effective tension or $T_1 - T_2$.

These losses are applicable to belts operating without automatic-idler or pivoted motor-base control.

Practical Belt Speeds. Belt speeds of 1000, 1500, 2000, and 3000 ft. per min. are common, but the most economical and efficient belt velocities range from 3500 to 4500 ft. per min. At 5000 ft. per min., centrifugal tension is high, causing excessive strain and wear on the belt. At 10,000 ft. per min., the centrifugal tension equalizes the normal tension of the belt which theoretically reduces the power-transmitting ability to zero.

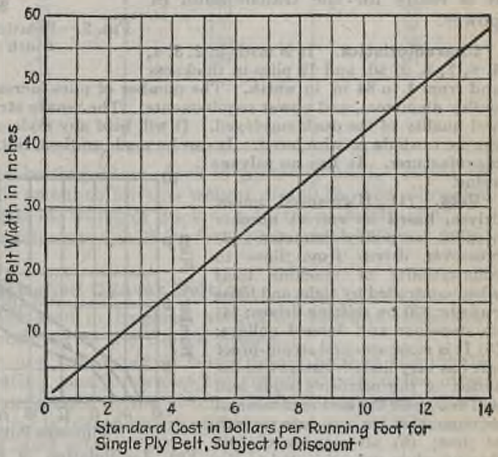


Fig. 1.

Rubber Belting

Origin. Rubber belting made its appearance approximately in the year 1844, but it did not receive much impetus until after the close of the Civil War, at which time leather became scarce and high priced. However, the industry did not reach its real period of growth until after 1880.

Construction. The average belting of this type is known as *friction surface rubber belting*, and it is composed of plies of powerful cotton duck impregnated or frictioned with a strong, tenacious, slow-aging rubber compound. Modern manufacturing methods produce a belt which is homogeneous and inseparable, possessing strength, flexibility, and durability when properly selected and applied. The rubber friction is forced into the interstices of the duck by massive iron calender rolls.

Manufacture. After the duck for rubber belting is thoroughly inspected and dried, it is impregnated or frictioned with the rubber compound. This rubber must be compounded so that the resultant material will be highly elastic and tough and be capable of retaining these particular qualities during the full life of the belt.

The actual manufacture of the belt begins with the cutting of the friction duck to the desired width, the width of the strips depending on the size of the belt desired. After cutting, the duck is then folded in such a manner as to give the width and thickness of belt required. The raw belt, constructed as described, is then placed, a section at a time, in a large horizontal vulcanizing press. The time required for vulcanizing usually averages from 20 to 40 min., depending on thickness. After this process, it is ready for the transmission of power.

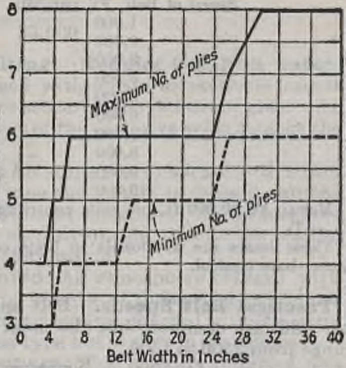


FIG. 2.—Relation of number of plies to belt width for rubber belting.

Characteristics. It is made in 2, 3, 4, 5, 6, 7, 8, 9, 10, and 12 plies in thickness and from 1 to 84 in. in width. The number of plies increases in proportion to width, pulley diameters, and power requirements. The tensile strength depends upon weight and quality of the duck employed. It will hold any style of metallic fastener and also can be rawhide or wire laced. It can be made endless, but this should be done by the manufacturer. It has no salvage value.

Uses. (1) High-speed motor drives, based on correct number of plies for pulley diameters; (2) crossover drives from line- to countershafts of machine tools when controlled by tight and loose pulleys; (3) on shifting drives; (4) on step-cone and flanged pulleys; (5) It is moisture- and steam-proof but has very little resistance to the action of the corrosive acids and will not resist the action of mineral or animal oil for any great length of time; (6) all manner of belt driving when not subjected to the atmospheric conditions given.

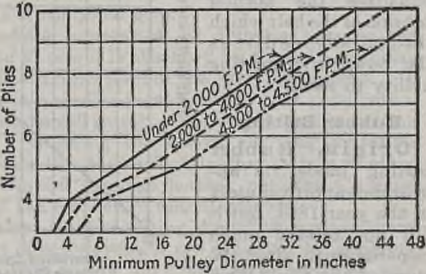


FIG. 3.—Relation of number of plies to pulley diameters and speeds for rubber belting.

Dressings for Rubber Belting. It is impossible to actually dress or lubricate the fibers of a friction surface rubber belt because the rubber friction compound between and in the duck plies cannot be penetrated. All types of dressings applied, based on this fact, remain on the surface causing endless amounts of trouble. The immediate use of vegetable castor oil or boiled linseed oil on new rubber belting will remove the bloom of the rubber and give a non-dangerous adhesiveness to the belt. These oils are beneficial if used sparingly on old or worn rubber belting.

Formula for Rubber Belting. Rubber-belting manufacturers use the following formula:

$$\text{Hp.} = \frac{WNS(T_1 - T_c)}{33{,}000}\left(1 - \frac{1}{R}\right)$$

where Hp. = horsepower capacity of belt.
W = width, in.
N = number of plies.
S = belt speed, ft. per min.
T_1 = tight-side tension, lb. per in. width.
T_c = centrifugal tension, lb. per in. width.
$\left(1 - \frac{1}{R}\right)$ = correction factor for arc of contact (see table).

Arc of Contact Correction Factors

Arc of contact, deg.	$1 - \frac{1}{R}$	Arc of contact, deg.	$1 - \frac{1}{R}$
90	0.29	190	0.51
100	.32	200	.53
110	.35	210	.55
120	.37	220	.57
130	.39	230	.58
140	.41	240	.60
150	.44		
160	.46		
170	.48		
180	.50		

For good quality rubber belting 32-oz. duck is generally used. Based on an average ultimate tensile strength for this weight of 440 lb., the allowable effective tension is 25 lb. per in. width of duck or a factor of safety of 17½. $(T_1 - T_c)$ can therefore be calculated as 25 lb. for a 32-oz. duck belt and 20 lb. for 28-oz. duck.

Stitched Canvas Belting

Origin. This type originated in England and, after meeting with considerable success, was introduced to the United States.

Construction. There are two types of construction used in the manufacture of this class of belting, one known as the round edge and the other as the folded edge. The difference in the two methods of construction is in the manner in which the duck is folded before stitching and in the building up of the various numbers of duck plies.

The round-edge construction gives a smoother surface to the action of shifter forks and prevents premature cracking of the edges of the belt. The better grades of this class of belt are usually made from 36- to 37½-oz. duck, which possesses a tensile strength of from 550 to 600 lb. per in. of width, therefore, giving the belt high tensile strength.

After folding for either the round- or folded-edge construction, the belt is stitched and is then thoroughly impregnated with certain compounds either by the immersion or the pressure process, thus rendering the belt impervious to mechanical and atmospheric actions. After the belt is thoroughly impregnated it is stretched and is then ready for the transmission of power.

Characteristics. It has a high coefficient of friction, extreme flexibility, resistance to ply separation, and is elastic. It will hold any type of metallic fastener but is difficult to make endless. It has extreme uniformity, therefore results in a true running belt. It has no salvage value.

Uses. (1) High-speed motor drives, based on correct number of plies for pulley diameters; (2) crossover drives from line- to countershafts of machine tools, preferably when clutch controlled; (3) shifting drives, preferably when the belt is of 6-ply construction; (4) it will resist the action of water, steam, heat, and mineral oil but will not resist the action of the corrosive acids; (5) all manner of slow speed, heavy driving, when not subjected to the atmospheric conditions stated.

Formula for Stitched Canvas Belting. As in rubber belting, the allowable effective pull is based on so many pounds per inch ply of the belt and varies according to the weight of the duck employed. The average good grade of stitched canvas belting is made up of 36-oz. duck, 28 lb. per in. width allowable effective pull on tight side.

The formula for 36-oz. duck at 28 lb. allowable effective pull per inch is

$$\text{Horsepower} = \frac{WNS(T_1 - T_c)\left(1 - \frac{1}{R}\right)}{33{,}000}$$

Use 28 lb. for the value of $(T_1 - T_c)$ in the above formula.

Hair Belting

Origin. Hair belting originated in Manchester, England, in 1873. In the late eighties, a company was formed in the United States for the production of this type of belting, and the brand manufactured was known as *camel's hair*.

Construction. Belting of this type is made of hair yarn of various kinds and mixtures of hair and wool.

Hair yarn must possess high tensile strength, elasticity, and no excessive stretch. Belting of this type made of pure camel's hair which has been properly selected and prepared is considered the best. The best camel's hair comes from the long-haired, two-humped Bactrian camel of northern China, Mongolia, and Siberia. The cotton yarn used in the best types of this class of belting should be of long fiber and the very best quality. This type is of the solid-woven variety and, prior to weaving the hair, is cleaned, and the longest fibers made up into yarn.

In weaving, the hair yarn becomes the warp of the belt while the cotton yarn is used as the filler and also a binder to hold the several layers of hair together in a compact and homogeneous fabric. The warp or lengthwise hair strands form the portion of the belt which gives the traction and takes the load pull. The cotton yarn which binds the several layers of camel's-hair yarn together acts as a check on the hair and restrains its stretch.

In some types of hair belting the edges are made of cotton or leather known as friction edges. These edges are designed to protect the belt against the action of shifter forks.

After weaving, the belt is impregnated with a combination of heavy oils which preserve the entire fabric, lubricate it, and aid the frictional qualities.

Characteristics. It is made in the following weights and thicknesses:

Weight	Thickness, In.
Light	3/16
Light, double or single	1/4
Double	9/32 to 3/8
Extra heavy	9/16

It is highly elastic and, therefore, should be placed on the pulleys extremely tight. It has no laps or plies. It will remain soft and pliable during its life. It possesses an average ultimate tensile strength of 6300 lb. per sq. in. It has extreme flexibility and therefore can be run on high-speed small pulleys. It will hold any style of metallic fastener but cannot be made endless. It has no salvage value.

Uses. (1) High-speed motor drives, based on correct thickness for pulley diameters; (2) intermittent loads where shocks must be absorbed by the belt; (3) shifting and cross drives; (4) heavy hard-pull drives; (5) on crossover drives from line to counter of machine tools, whether clutch or tight- and loose-pulley control; (6) it is not affected by mineral oils unless the amount is excessive. It will stand heat to 300°F. and is not affected by extreme cold. It will stand for long periods the action of corrosive acids but should not be used where caustic is present. It is moisture- and steam-proof; (7) all manner of driving when not subjected to atmospheric conditions stated.

Formula for Hair Belting. Based on the fact that hair belting is a solid-woven product and not made up of plies, certain allowable effective pulls must be taken as a basis for power capacities, and these naturally must vary for the different weights.

On this basis, the same formula as used for the various types of leather belting can be used for hair belting.

The following allowable effective pulls on tight side of belt are used in the power ratings of the various weights of this class of belting:

Light or single.................... $\frac{3}{16}$ in. thick = 40 lb. per in. of width
Light or double.................. $\frac{1}{4}$ in. thick = 50 lb. per in. of width
Double......................$\frac{9}{32}$ to $\frac{3}{8}$ in. thick = 70 lb. per in. of width
Extra heavy...................... $\frac{9}{16}$ in. thick = 90 lb. per in. of width

Formula:

$$\text{Horsepower} = \frac{TWS}{33{,}000}$$

where T is the effective pull (in tight side per inch of width); W, the belt width, in inches; and S, the belt velocity in feet per minute.

Table 1

Weights of hair belting	Thickness, in.	Pulley diameter, in.	
		Under 2000 ft. per min.	2000 to 4000 ft. per min.
Light..................................	$\frac{3}{16}$	3	5
Light double...........................	$\frac{1}{4}$	5	7
Double.................................	$\frac{9}{32}$–$\frac{3}{8}$	10	14
Extra heavy............................	$\frac{9}{16}$	16	24

Belt Joining

Improper and careless joining is one of the principal factors of belting maintenance costs and one of the causes of short belting life. Based on this, the following should be adhered to when making a selection.

The fastener must not unduly strain the belt; must not cut or weaken the belt; must resist wear caused by pulley contact; must conform to the pulley circumference. It should allow the belt ends to fit closely and prevent breakage at joint. It must be quick of application.

The Endless Method. Joining a belt so that the result is an endless band is the most efficient method because the joint is integral with the belt and, therefore, eliminates the use of any foreign material. All types of belting have stretch, a factor which necessitates the use of take-up facilities where the endless belt is employed; otherwise considerable trouble will be experienced.

Leather belting offers the greatest possibilities for the endless or cemented joint because in reality the belt itself is composed of cemented joints. To make a leather belt endless the ends should be scarfed or scraped down to a thin edge, 3, 4, or 6 in. back, depending on the width, and the cement or glue applied to the scarfed surfaces. These surfaces are then placed together and subjected to continuous pressure for two or three hours. When joining with pyroxylin cement as on a waterproof leather belt, the scraped surfaces are first sized with the cement and allowed to dry. After drying, another application of the cement is made and the surfaces placed together. Pressure is then applied for at least three or four hours.

The endless joint should be employed on all drives where center adjustment is possible—on high-speed spindle driving of machine tools, on large engine installations, and on automatic idler and pivoted motor-base drives.

The endless joint should not be used on drives where center adjustment is not possible—on vertical drives without idlers and on quarter-turn drives.

Endless Rubber Belting. Rubber belting of the ply type can be made endless by cutting and stepping the plies back from the ends so that when placed together they coincide. When set in this position, vulcanization is necessary. This can be done at the point of application by the use of small portable vulcanizing machines. Rubber belting of the cord type is now manufactured endless, with no joint or splice.

The Laced Joint. Rawhide is the most common material employed for making the laced belting joint and if properly made results in a good joint. It should always be employed where hand shifting is necessary because of the safety feature. Rawhide lacing is being done away with on general driving because of the time required to apply it and the advent of the quicker and efficient metallic methods. By the courtesy of John Wiley & Sons, Inc., the following table is reproduced from Kent's, "Mechanical Engineers' Handbook," p. 1308:

Table 2

Width of belt, in.	Punching of lace holes, distance from ends of belt, in.		Distance of center line of first hole in first row from edge of belt, in.	Size of belt lace, in.
	First row	Second row		
2– 4	3/8	3/4	3/8	3/16
6– 8	1/2	1	1/2	1/4
10–12	5/8	1 1/4	5/8	5/16
14–16	3/4	1 1/2	3/4	3/8
18–20	7/8	1 3/4	7/8	7/16
22–24	1	2	1	1/2

Metallic Fasteners. There are a number of metallic fasteners on the market, and they are all used extensively because of their efficiency and time-saving factor of application. The wire hook and pressed-steel-pin type, the plate type, and the staple or prong type are the most popular.

The wire hook type can be used on single, light double, and regular double leather belting up to 8 in. wide and on all types of rubber and fabric belting from 2- to 6-ply up to 8 in. wide, on drives where unfastening must be done quickly, on high-speed work where the endless joint is not practicable, where an insert is necessary, and on shifting drives.

The wire hook should not be used on hand-shifted drives, on heavy double leather, or on rubber or fabric belting over 6 plies in thickness, on automatic idler drives.

Table 3. Minimum Pulley Diameters and Belt Thicknesses for Wire Hooks

Hook size	Thickness of belt	Minimum pulley diameter, in.
2	2 or 3-ply fabric	2½ and 3
4	4-ply fabric and single leather	6
6	6-ply fabric and regular double leather	12

The plate fastener can be used on single, light, and regular double leather from 1 in. to any width required, and on all types of fabric belting from 3 to 12 ply and in widths from 1 in. to any width required, on open and crossed drives, on hard-pull slow-speed driving, and on shifting drives.

The plate fastener should not be used on any type of idler installation, on high-speed small pulley work, on hand-shifted belting, or on special tannage leather belting.

Table 4. Plate Sizes for Various Belt Widths and Minimum Pulley Diameters

Belt width, in.	Number and size of plates				
	4-in. diam. pulley and larger	6-in. diam. pulley and larger	9-in. diam. pulley and larger	12-in. diam. pulley and larger	24-in. diam. pulley and larger
2	....	1–No. 607	1–No. 63		
4	....	2–No. 607	2–No. 63		
6	4–No. 66	4–No. 67	2–No. 103	2–No. 149	
8	....	2–No. 147	4–No. 63	2–No. 189	
10	....	....	4–No. 83	4–No. 109	
12	....	....	4–No. 103	4–No. 149	4–No. 1611
14	....	....	4–No. 123	4–No. 1409	4–No. 1611
18	....	....	....	6–No. 149	6–No. 1611
24	....	....	....	....	6–No. 2211

Care and Treatment of Belting

What to Consider. (1) Select correct type and size for the atmospheric and mechanical conditions involved and the power requirements when the initial installation is made; (2) over-power rather than under-power; (3) make pulleys slightly wider than the belt; (4) select proper joining method and see that ends are square before joining; (5) allow belt to pull on bottom if possible; (6) centralize the care of belting; (7) keep pulleys and shafting in line; (8) inspect the belting lap or splices frequently; (9) inspect the fastening method frequently; (10) keep belting clean and periodically dress it with the proper lubricant.

Nothing should be used on a belt with the mere object of making it sticky. Power is wasted in tearing a belt away from the pulleys, and this tearing action is destructive to the belt. Sticky dressings cause the formation of lumps on the surface of the pulleys which create loss of contact and breaking of the friction seal between belt and pulley.

Table 5. Minimum Center Distances

Based on power and ratios

Horsepower to be transmitted	Ratio of driver to driven	Center distance, ft.
1– 4	3/1–5/1	4–10
5– 9	2/1–5/1	8–12
10– 14	2/1–5/1	8–14
15– 24	3/1–5/1	9–15
25– 39	3/1–5/1	10–16
40– 49	2/1–5/1	12–17
50– 74	2/1–5/1	13–18
75– 99	3/1–5/1	18–24
100–124	2/1–5/1	20–26
125–149	2/1–5/1	22–28
150–200	2/1–5/1	26–32

BELT DRIVES

Vertical Drives. A belt drive in a vertical position without take-up facilities, although necessary at times, is always troublesome. The natural stretch of the belting substance causes sag of the belt away from the bottom pulley, resulting in power loss and excessive maintenance. When a vertical drive must be employed without take-up provisions, use a center distance as short as possible because this provides less accumulated stretch per foot.

Crossed-belt Drives. Belting is crossed for the purpose of reversing direction of rotation and arc of contact increase, but consideration must be given to pulley diameters, belting width, and type of fastener employed if good results are to be obtained. The pulleys should be of reasonable diameter and as near equal in this dimension as possible. Belting over 8 in. wide should not be crossed because of the large area of belt in contact. A poor joint or projecting metallic fastener will quickly cause the destruction of a crossed belt.

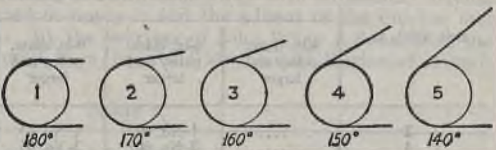


FIG. 4.—Loss of contact on driving pulley by high ratios.

Quarter-twist Drives. Quarter-twist drives are employed for transmitting power by belting at right angles, either vertically or horizontally. Leather belting gives the best results because it can be especially constructed to suit this type of drive. An empirical rule for this installation follows:

RULE: *The center of the face of the loose side of the driver must line with the center of the face of the tight side of the driven.*

Mule or Guide-pulley Drives. This type of driving makes it possible to transmit power by belting up over or around a corner. It is accomplished by the aid of a guide or mule pulley as it is generally termed, operating on independent spindles or shafts. Machine tools, such as single- and multiple-drill presses, are frequently driven in this manner. This type of drive is difficult to maintain and should not be used unless absolutely necessary. Best results are obtained from a narrow belt width, not in excess of 6 in.

Cone-pulley Driving. Tapered cone-pulley drives are employed when variation in speed is required, but, unless a properly constructed belt is used, constant trouble will result. Crossed belting operates to a better advantage on cone pulleys and especially where the taper is extreme. Cone-pulley belts are as a rule shifted frequently, and, therefore, lateral stiffness of one ply is necessary. This is accomplished by making the outer ply wider and of a stiffer material than that which is against the pulley.

Belt slippage causes power loss and destroys belting. It can be caused by (1) too small pulleys, (2) overloading, (3) underloading, (4) non-lubrication of belting, (5) sticky dressings, (6) atmospheric effects, (7) misalignment, and (8) bad joining.

Knowledge of these factors and frequent inspection of belting will prevent premature destruction from belt slippage. A polished, shiny face of a pulley denotes belt slippage.

The Crowning of Pulleys. Belting connecting the pulleys of parallel shafting tends to run toward that part of the pulley which is largest in diameter. It is for this reason that pulleys are higher at the center of the face or rim than they are on the edges. This raise is termed the *crown*. Crown-faced pulleys should be employed on all belted installations with the exception of the tight and loose pulley drive. The twin, or tight- and loose-driven, pulley should be crowned, but the driver, or drum pulley, should be flat or straight faced so as to permit easy sliding of the belt from the loose to tight pulley and vice versa.

PULLEYS

There are three general classes of pulleys for the transmission of power by belting, namely, **cast iron, steel,** and **wood.** There are several types and makes of each style manufactured, which in itself causes confusion when a selection is to be made. In the average production shop, it is difficult to differentiate in regard to the pulley to employ because conditions are generally normal, but in industrial plants where chemicals, dyes, paints, etc., are manufactured it is a problem.

Cast-iron Pulleys

The cast-iron pulley is made in the following types:

1. Single arm, solid or split.
2. Double arm, solid or split.
3. Keyless or interchangeable bushed.

Construction. Cast-iron pulleys are usually designed and built to withstand the strains under which the respective belt can be expected to operate. The cast-iron pulley is of rigid construction and therefore runs true if properly set and fastened to the shaft. Due to its rigidity, sudden shock, loads, or falls are liable to cause cracking or fracture of the iron.

Speeds. Standard pulleys are made for rim speeds of 3500 to 4000 ft. per min., and while rim speeds from 5000 to 6000 ft. per min. are permissible such pulleys should be specially made and balanced.

Applications. The cast-iron pulley will stand heat, moisture, steam, and acid fumes; therefore, it has a wide range of usefulness. A cast-iron pulley will last for years if operated according to its capacity.

Steel Pulleys

Types. The pressed-steel pulley is made in three distinct types: the one-piece rolled edge, crowned rim, split; the one-piece crowned rim, without edge roll, split; and the two-piece crowned rim, edge roll, split.

All have respective advantages and disadvantages; therefore, as to their usage, it is difficult to say which is the best because for the average industrial service the three types are efficient.

Construction. The steel pulley is a fabricated structure; therefore, there exists a wide difference in design. Within working limits of belting, however, strength of joints and the greatest possible rigidity have been secured. The difference in type particularly pertains to the construction of the rim, which has been noted in the above paragraph.

Speeds. The pressed-steel pulley can be operated with safety at high rim speeds, 4000 to 4500 ft. per min. being common practice. On tests, this type of pulley has been run at 14,000 ft. per min. for periods of 30 hr. without signs of distress. This would not be practicable in power transmission owing to the fact that 4500 ft. per min. is the most economical belt speed.

Applications. Steel pulleys can be operated in similar places to those where cast-iron ones are used, with the exception that they should not be used in the presence of corrosive acids or in severe moisture and steam.

This pulley is generally of the split type and is furnished with interchangeable metallic bushings which make it possible to use either clamp, key, or set screw.

Wooden Pulleys

Types. Under normal conditions the modern wooden pulley will perform in every way as efficiently as the cast-iron or steel pulley, and, in addition, it is light and low in price.

There are three distinct types made, namely, the heavy-duty type of split design with standard bore into which the required wooden bushing is placed, the wooden-rim iron-center type, and the solid-wood type.

Construction. The wooden pulley is usually of the split type and for average driving is clamped to the shaft. It is made from hard maple and finished to all standard sizes.

Speeds. The wooden pulley is safe to any speed up to 5000 ft. per min. and has long life if operated under correct mechanical and atmospheric conditions.

Applications. Wooden pulleys are satisfactory for all ordinary driving and in operations such as crushers, stamp mills, oil-well rigs, and in hazardous operations such as explosives manufacturing. Wooden pulleys should not be used where water, dampness, or steam is present.

Selection. In selecting pulleys the following points should be borne in mind: (1) The surface of the rim should have a high coefficient of friction, yet it must not be rough because belt slippage cannot be entirely eliminated and, when a belt slips on a rough pulley, wear is rapid. (2) A pulley should combine strength with light weight. Unnecessary weight loads up the shaft and, generally, increases the friction of the drive. (3) The pulley should be able to withstand severe service and resist breaking in case of sudden overload or shock. (4) A pulley should be, to a certain extent, a heat conductor as belt slip generates heat. (5) A split pulley has great advantages over a solid pulley because it can be mounted on the shaft with the minimum amount of shutdown and labor. (6) A pulley should be exactly round and of such material and construction that it is unaffected by atmospheric conditions. (7) A pulley should be designed so that it will rotate with the least amount of air resistance.

Ordering Pulleys. Pulley manufacturers are more familiar with their products than the consumer; therefore, when ordering pulleys careful observance of the following rules will avoid errors and save time:

1. **Service:** Give horsepower, r.p.m., and character of service.
2. **Description:** State whether solid, split, clamp hub, tight and loose, flanged or special.
3. **Diameter:** Specify diameter in inches. This should be the first dimension given. If exact diameter is required, mention this fact and state whether measurement shall be made at crown or edge of rim. Unless otherwise specified, the diameter of the pulley is the diameter at the top of the crown.
4. **Face:** Specify face in inches. This should be the second dimension given and should be specified in accordance with the width of the belt unless an exact width of face is desired in which case the fact should be noted by having the word *exact* follow dimension of face.
5. **Bore:** Specify exact diameter of shaft in inches. This should be the third dimension given.
6. **Crown or Straight Face:** After specifying dimensions of pulleys, state whether crown or straight face. If neither is specified, crown face is generally furnished. Drum pulleys for shifting belts have straight faces. Each pulley of a pair of tight and loose pulleys should have crown face.
7. **Key-seat or Set-screwed:** State whether key-seated or set-screwed, or both.

Belt Width and Pulley Face. The face of a pulley is generally determined by the width of belt to be employed. As an example, if a 6-in. double leather belt is to operate over a 24-in. diameter pulley, the pulley should be specified as a 24 by 6 in. However, the actual width of the face of the pulley would be approximately 6½ in. because it is a standard with pulley manufacturers to make the face of the pulley at least ½ in. wider than the belt to be used, so as to overcome the possibility of the belt running off the pulley due to weave. Pulleys to accommodate belts wider than 12 to 14 in. up are usually made 1 to 1½ in. wider in face than the width of the belt.

Relative Characteristics of the Various Types of Pulleys. *Cast-iron pulleys* are made for rim speeds of 3500 to 4000 ft. per min., and while rim speeds from 5000 to 6000 ft. per min. are permissible such pulleys should be specially made and balanced. The cast-iron pulley can be used for any practical power requirements.

Pressed-steel pulleys are made to operate with safety at rim speeds of 4000 to 4500 ft. per min. However, on tests, pressed-steel pulleys have run at 14,000 ft. per min. for periods of 30 hr. without signs of distress. The pressed-steel pulley can be used for any practical power requirements.

The *wooden pulley* is safe at any speed up to 5000 ft. per min. and can be used for any practical power requirement.

GEARING

The general subject of gearing is so well treated in Marks' "Mechanical Engineering Handbook" and similar handbooks that merely the various types will be mentioned here. The various types of power gears are as follows: **spur, miter, bevel, mortise, worm, herringbone,** and **helical.**

BEARINGS

For the industrial transmission of power there are various types of bearings manufactured, and if employed under the mechanical conditions for which they are designed will give efficient and economical service. The four general types of bearings are as follows: grease lubricated, self-oiling, antifriction, and oilless.

Grease-lubricated Bearings. There are three general types of grease-lubricated bearings, namely, the solid babbitted bearing which should be used on shafting or axles under $4\frac{7}{16}$ in. revolving at slow speed and where adjustment is not necessary; the split flat box which is extensively employed where moderate powers and speeds are involved but should not be employed on shafting over $3\frac{15}{16}$ in. diameter; and the plain rigid pillow block which is designed for almost any load on slow-speed shafting.

Self-oiling Bearings. These bearings are made in several types, such as the rigid and ball-and-socket pillow block, the ball-and-socket drop hanger, post hanger, and bracket hanger, and employ three distinct features of self-lubrication, namely, the ring oiling system, the capillary system, and the wick system. These bearings are applicable to all manner of driving.

The rigid or ball-and-socket pillow block is usually employed to support shafting from concrete or wooden piers, the drop hanger to support shafting from ceiling or roof truss, and the post hanger to support shafting from columns or posts.

Antifriction Bearings. There are two distinct types of antifriction bearings, namely, the ball and the roller, each possessing its own individual characteristics.

The steel-roller type can be subdivided into three types, namely, the flexible, the tapered, and the solid.

The ball-bearing type naturally can utilize only the ball.

Antifriction bearings are made for drop hangers, post hangers, and pillow blocks, and may be universally employed for transmission purposes.

Oilless Bearings. The oilless or self-lubricating bearings are those that contain within themselves sufficient lubrication to assure continuous service. The general types are as follows:

The bronze and graphite type which is constructed of high-grade phosphor bronze into which are cast symmetrical grooves for graphite, varying in design according to the service for which the bushing is intended; the hard-wood, lubricant-impregnated type which is made of hard seasoned wood, thoroughly impregnated with a specially prepared lubricating compound (no additional lubricant is ever required); and the metaline type which is constructed of either phosphor-bronze or gun metal-bronze and into which are inserted small-diameter metaline plugs. The metaline plugs are a mixture of graphite and antifriction metals. The oilless bearing is particularly adaptable for high-speed spindles and in the cotton and textile industries, generally, where cleanliness is a feature.

Spacing of Bearings. The number of bearings used on a shaft, and their spacing, has a direct influence on the dead-load capacity of the shaft. Deflection of shafting should not exceed 0.01 of 1 in. per ft. Beyond this, binding in the bearings occurs, causing increased friction load and the possibility of burned-out bearings. There is no definite rule for the exact spacing of bearings; therefore, in most cases it is a matter of judgment. On line shafting, 8-ft. centers is good practice where the pulleys can be set close to the bearings.

Bearings for Power Shafting. Based on the numerous designs of bearings manufactured, the selection of a type is at times difficult. The factors governing bearing selection are: (1) diameter and speed of shaft; (2) power and dead load; (3) support; (4) lubricant and lubrication method; (5) space limitations; (6) operating conditions; (7) initial and maintenance costs.

The actual bearing surface should be of a metal softer than the shaft or journal, enabling the bearing to wear and not the shaft. The most extensively employed bearing metal is known as *babbitt*, which is a mixture of antimony, lead, and tin. Brass is an excellent bearing metal if the bearing surfaces are well scraped together with the shaft; otherwise heating will occur. The rubbing surfaces of bearing and shaft are never perfectly true and smooth. For this reason babbitt or alloy metal is employed as it quickly in combination with the lubricant conforms to the shaft and the shaft to it. The shaft is always slightly less in diameter than the bearing. The difference is termed *bearing clearance*, which in average practice is about $\frac{1}{1000}$ in. per in. shaft diameter. Accurate alignment of shafting is necessary for efficient bearing performance.

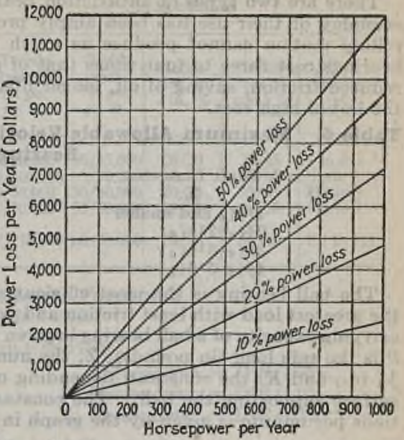


FIG. 5.—Monetary losses caused by excessive friction loads.

Bearing Pressures. These pressures are generally moderate or excessive: **moderate**, when bearings are designed adequately for the load; and **excessive**, when undersized for heavy dead loads and severe belt tensions. Bearings employed for industrial power transmission are safe for pressures up to 150 lb. per sq. in. projected area. This figure is rarely reached in this class of service.

Bearing Temperatures. Bearing temperatures are known as moderate and extreme: **moderate** are those not higher than 140°F., which is easily maintained in the average shafting installation if alignment, lubrication, and loading are given consideration. Temperatures above 140°F. are due to deficient radiation, internal friction, or the effect of high surrounding temperature.

Internal friction can be caused by improper mechanical conditions, wrong lubricant, or an insufficient amount of lubricant.

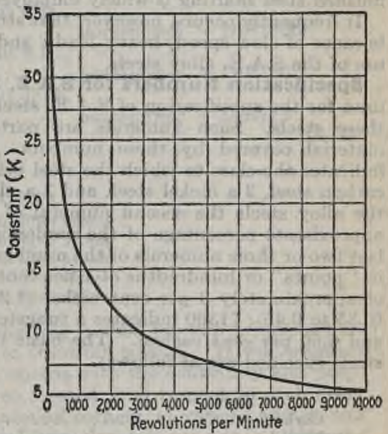


FIG. 6.—Relation of constant K to revolutions per minute.

Antifriction Bearings. Bearings of the ball and roller type reduce the internal friction of a bearing to a minimum. The graphic chart (Fig. 5) shows the monetary losses involved due to excessive bearing friction. It is based on 4 lb. coal per horsepower-hour, 300 working days per year, 10 running hours per day, and a coal cost of $4 per ton.

There are two types of antifriction bearings, the ball and the roller. The economy of their use has been amply proved in practice, mainly because a rolling motion cannot produce as much wear as a sliding motion. These bearings cost three to four times that of plain babbitted bearings, but their reduced friction, saving of oil, longer life, and decreased maintenance offset the initial high cost.

Table 6. Maximum Allowable Velocities for Line-shaft Solid Roller Bearings

Shaft Diameter, In.	Maximum Velocity R.p.m.
$1\frac{15}{16}$ and smaller	600
$2\frac{3}{16}$–$2\frac{15}{16}$	400
$3\frac{3}{16}$–$3\frac{15}{16}$	300
$4\frac{3}{16}$–$5\frac{15}{16}$	250

The ball bearing is the most efficient antifriction device known, carrying the greatest load with least friction and possessing great strength. The load-carrying capacity of a ball bearing is given by the formula, $P = KZD/5$, where P is the safe load (in pounds); Z, the number of balls; D, the ball diameter, $\frac{1}{8}$ in.; and K, the constant, depending upon the shape and material of the surface supporting the ball. The constant (K) as a function of the revolutions per minute is given by the graph in Fig. 6.

SHAFTING

Cold-finished Steel Shafting. Because of its low cost and adequate strength for the average power-transmission requirements of industry, cold-finished steel shafting is widely employed.

It frequently occurs, however, that stronger steels are required for shafting because of slow speed, heavy loads, and shocks. This makes necessary the use of the S.A.E. alloy steels.

Specification Numbers for S.A.E. Steels. A numeral index system is used for the specification of S.A.E. steels, which facilitates the specifying of these steels. Such numerals are partially descriptive of the quality of material covered by these numbers. The first numeral of the number indicates the class to which the steel belongs; thus the numeral 1 indicates a carbon steel, 2 a nickel steel, and 3 a nickel chromium steel. In the case of the alloy steels the second numeral of the number generally indicates the approximate percentage of the predominant alloying element. Usually the last two or three numerals of the number indicate the average carbon content in "points" or hundredths of 1 per cent. Thus 2340 indicates a nickel steel of approximately 3 per cent nickel (3.25 to 3.75) and 0.40 per cent carbon (0.35 to 0.45); 71360 indicates a tungsten steel of about 13 per cent tungsten and 0.60 per cent carbon. The basic numerals for the various qualities of steels specified are as follows:

Steel	Basic Numeral
Carbon	1
Nickel	2
Nickel-chromium	3
Molybdenum	4
Chromium	5
Chromium-vanadium	6
Tungsten	7
Silicomanganese	9

Physical Properties of Cold-finished Steel Shafting

Type	S.A.E. numbers				
	Tensile strength, lb. per sq. in.	Yield point, lb. per sq. in.	Elongation, in 2 in. %	Reduction of area, %	Size diameter, in.
Hot-rolled soft steel or screw stock	50/60,000	30/35,000	20/30	35/45	1 to 2
Cold-finished screw stock S.A.E. 1112 or 1120	70/100,000	55/90,000	10/20	35/60	Up to 1
Cold-finished screw stock S.A.E. 1112 or 1120	50/60,000	30/40,000	20/35	30/45	2¾ to 4
Cold-finished 35/45 carbon S.A.E. 1040	70/90,000	35/50,000	20/30	35/50	1½ to 2
Cold-finished annealed 35/45 carbon alloy—S.A.E. 3140, 2340, 6140, etc.	90/110,000	60/70,000	20/30	55/65	Up to 1½

Shafting Horsepower Formulas

Cold-finished steel

HEAD SHAFTING (First Shaft from Prime Mover):

Heavy strains and loads............ horsepower $= D^3R/125$
Average loading.................... horsepower $= D^3R/110$
Light loading...................... horsepower $= D^3R/100$

INTERMEDIATE SHAFTING (Such as Jacks and Counters):

Horsepower $= D^3R/90$

LINE SHAFTING:

Heavy load, with bearings about 8 ft., center-to-center.................. horsepower $= D^3R/100$
Medium load, with bearings about 8 ft., center-to-center.................. horsepower $= D^3R/90$
Light load, with bearings about 6 ft. center-to-center.................. horsepower $= D^3R/75$

D = diameter of shaft.
R = r.p.m. of shaft.

CONSTANTS

125 = 2800 lb. fiber stress
110 = 3000 lb. fiber stress
100 = 3200 lb. fiber stress
90 = 3400 lb. fiber stress
75 = 4000 lb. fiber stress

Flexible Shafting. This type of shafting is used for two principal purposes, namely to transmit power and to transmit motion. It will transmit small amounts of power at high or low speeds with the continuity and properties of a solid shaft between points so located with respect to each other that a solid shaft cannot be used, *i.e.*, around corners, at various angles, and in all other cases where the driving and driven elements are not in alignment or must be moved with respect to each other. It will transmit motion to elements which must be moved back and forth or turned through a full circle or any arc, whether by hand or automatically, in cases where the elements to be moved are so located that they cannot be readily reached by hand, or where a direct straight connection is impracticable.

In regard to the maximum size of flexible steel shafting, while it is entirely practicable to manufacture flexible shafts larger than ¾ in., factors enter into the design and construction of these larger shafts which materially increase the cost. The application of a shaft larger than ¾ in. becomes more or less a special problem.

Uses of the Flexible Steel Shaft.

1. Tachometers.
2. Speedometers.
3. Washing machines.
4. Moving-picture cameras.
5. Valve grinding machines.
6. Screw slotting machines.
7. Linotype machines.
8. Milling machines.
9. Glass-cutting machines.
10. Valve controls.
11. Printing presses.
12. Portable grinders.
13. Concrete surfacers.
14. All types of portable tools.

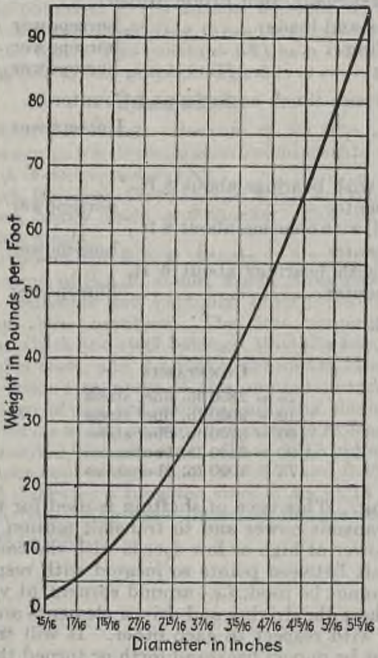


Fig. 7.—Weights of cold-finished steel shafting.

Characteristics. The selection of a flexible shaft best suited for a given application requires a definite knowledge of what the shaft should be used for, and the specific conditions under which it is to operate. The advisable procedure is to make a preliminary selection of a shaft which seems to have the proper characteristics of torsional strength and transverse flexibility and the necessary torque capacity and try it out under the actual operating conditions.

Drive Ends for Flexible Shafts. Shafting of this character is generally equipped with special ends for engaging the driving and driven members. The design of these ends is determined by the conditions of the particular application. In some cases the same design of end is used on both ends of the shaft, while in others different ends are used. The variety of shapes and sizes these ends may take is unlimited; for example, they may be forked, slotted, splined, square, flat, threaded, or, in fact, of any design and any material that the application may demand. It is a good practice, however, to have the ends made and attached by the shaft manufacturer.

Casings for Flexible Shafting. Practically every flexible shaft is provided with a casing which acts as a runway or guide for the shaft and protects it from foreign matter. It is sometimes referred to as the "sheath." Casings may be made of a variety of materials including braided fabric, metal, leather, rubber-filled materials, etc., but in practically every design of casing a metal winding forms the foundation over which the other material is laid. In the characteristics of transverse flexibility, the casing should conform as nearly as possible to the drive shaft which it encloses. In the ideal combination, casing and shaft will bend with exactly the same flexure and will parallel each other in the bends.

GEAR REDUCTION UNITS

The modern gear speed-reduction units utilize four general types of gearing, namely: (1) the worm and worm wheel, (2) the spur, (3) the herringbone, and (4) the bevel. These various types have been largely standardized so that they are now available in suitable sizes and ratios for almost any power and speed-reduction requirement. Each type has inherent limitations; therefore, care should be exercised based on the knowledge of the units in question before a selection is made for a particular installation. Trade catalogues of gear-reduction-unit manufacturers generally list ratios, horsepowers, and speed capacities of their respective units. Ratios in single reductions up to 7:1 can be efficiently handled by belting, chains, or open gearing; and up to 12:1 by the same mediums if efficiency is of minor importance. The modern speed-reduction-gear unit has made possible the elimination of a multiplicity of chains, gears, shafting, and belting for ratios such as 15:1 and higher.

1. Worm Reduction Gear Units.

Types. The modern worm-gear speed reducer has been perfected to such an extent that all conceptions as to the inefficiencies of this type of driving have vanished. Therefore, it is used extensively in all classes of industry and may be divided into the following types:

1. The bottom-driven unit.
2. The top-driven unit.
3. The vertical unit.
4. The compound unit.

Construction. The modern worm gear is constructed with as steep a helix angle on the worm as possible because the greater the helix angle (up to 42 deg.) the greater the efficiency, and the smaller the angle the greater the power loss.

The worm of a first-class modern reducer is usually integral with an alloy-steel shaft with threads hardened and ground. This combination is rigidly supported on ball or roller bearings, the rear bearing being of the double-row maximum type, capable of carrying the full thrust load of the worm in either direction as well as one-half the radial load. The other half of the radial load is taken by the forward bearing.

The worm gear usually consists of a chilled cast-bronze rim shrunk on, and pinned to, a high-grade cast-iron center. The entire mechanism is encased in a rugged oil-tight cast-iron housing.

Ratios and Speeds. It is possible to obtain extremely large reductions with worm-gear units, but in the higher reductions the power loss precludes their economic application. Ratios of 60 or 70 to 1 are common, but beyond that point the efficiency drops rapidly. To obtain higher reductions and still retain the right-angle feature of the worm drive, it is common practice to use an auxiliary shaft supported independently, on which can be mounted a pinion which in turn meshes with another gear, or to employ a reduction unit specially designed with suitable bearings to permit the mounting of the pinion directly on the slow-speed shaft.

It is also possible to get very high reductions by means of a compound worm-gear reduction unit.

Modern worm-gear reduction units can be operated at worm speeds up to 4000 r.p.m. and are, therefore, suitable for direct connection to steam turbines.

Applications. The worm-gear reduction unit is essentially a right-angle drive, and the housings and gearing arrangements are furnished by manufacturers in various forms to suit industrial requirements.

The worm-gear unit is applicable to almost any type of driving where the input speed is high and the output speed is low because it can be flexible coupled to a motor supported on a base plate, all making one integral unit.

Worm-reduction-gear Ordering. When ordering worm-gear reduction units the following information should be given: (1) horsepower of motor or driving shaft; (2) actual horsepower to be transmitted; (3) worm speed; (4) gear-shaft speed; (5) time during which unit will be in continuous operation; (6) description of driven machine or character of service.

Lubrication. The worm-reduction unit will not operate satisfactorily without oil. Its rotating parts must operate in a bath of good and suitable oil. Without this, short life and poor service can be expected. Never take it for granted that the reducer is supplied with sufficient oil when received from the manufacturer.

2. Spur-gear Reduction Units. Because the efficiency of the worm-gear reduction drops rapidly when ratios of 60 to 70:1 are exceeded, the spur-gear reduction unit has been evolved to cope with higher ratios. Ratios as low as 1:1 and as high as 500:1 are practical with the modern spur-gear reduction unit. This type reduces in a straight line, which in many instances is more advantageous than the right-angle feature of the worm-gear reducer.

The spur-gear reducer has peculiarities of its own and can be classified into two distinct groups: (*a*) the planetary and (*b*) the non-planetary types, both having certain advantages.

(a) Planetary Type. This reducer is capable of giving the largest speed reduction because it consists of spur gears and idlers radially disposed about a central pinion which in turn meshes with a stationary internal gear. The idlers must revolve on their own axes and, in addition, must be free to climb around the internal gear. To accomplish this, the idlers are mounted on studs which are rigidly fastened to a spider mounted upon the slow-speed shaft.

The planetary reducer with single reduction is suitable for ratios between 4 and 8:1 but should not be considered for ratios of less than 3:1. High ratios such as 100, 200, 400, and 500:1 are obtainable in the planetary type of reducer by the reductions being doubled or tripled and coupled in series in the same cast-iron housing.

(b) **Non-planetary Type.** This reducer has the advantage of giving comparatively low reductions and keeping the rotational speeds at minimum. Ratios as low as 1:1 and as high as 300:1 are practical.

This type consists of spur gears radially disposed about a central pinion. Each of the spur gears is keyed direct to pinions which in turn mesh with a central gear mounted upon the slow-speed shaft.

3. **Herringbone-gear Reduction Units.** The popularity of the worm- and spur-gear reduction units is now being shared by the herringbone type of reduction unit because it is highly efficient as well as silent in operation. It is especially adapted to the transmission of large powers although it is rapidly coming into use for intermediate and low horsepowers notwithstanding its comparatively high cost. The herringbone type of speed reducer does not possess the right-angle feature of the worm drive or the shaft-in-line feature of the spur-gear reduction unit. Its high- and low-speed shafts are offset, although they are parallel with each other.

The herringbone unit is made in the single-reduction, double-reduction, and triple-reduction types, and the gears themselves are either cut with a central groove or with teeth having sharp apices.

Ratios and Speeds. The single-reduction herringbone unit consists of one pinion and one gear, and is manufactured up to ratios of 7 or 8:1. The double-reduction gear is commonly manufactured in ratios up to 60:1. Greater ratios are more economically secured by the triple-reduction type.

The herringbone reducer may be direct coupled to high-speed motors.

VARIABLE-SPEED MECHANISMS

Many modern forms of apparatus require minute variations in applying the speed. Frequently when such variation is desired, the control must be rapidly and easily accessible to the operator.

For many operations of this character the ideal installation is the variable-speed motors, but it is difficult to obtain fine adjustment with this method. Therefore, there are two distinct types of variable-speed control—electric and mechanical, the mechanical device being most widely used for graduated speed reduction or increase.

Mechanical Infinitely Variable Speed Units. There are two types that appear to be the most popular; namely, the Reeves which utilizes smooth-surfaced cone disks with their apices facing inward, between which operates a specially constructed rubber and wood block belt; and the P.I.V. unit which utilizes radially toothed cone disks with their apices facing inward, between which operates a laminated toothed chain. The former is available in ratios to 16:1 and with horsepower capacities from fractional to 150, whereas the latter is available in ratios to 6:1 with horsepower capacities from 1 to 15. Selection depends upon installation space available and the refinement of infinite variation required. Both types can be had in either horizontal or vertical construction, and both can be supplied motorized.

Applications. These units can be used wherever variable speed is desired. Therefore, they are invaluable in canning, textile, machine, mining, paper, and cement industries.

Advantages. The advantages of utilizing the mechanical speed-variation unit are: (1) It provides an infinite number of speeds. (2) The transmission of power is positive at all speeds. There is practically no slippage. (3) It automatically locks in place at any desired speed. (4) It may be mounted in any position; therefore power may be applied easily by belting, chain, or gearing. (5) It may be driven in either direction. (6) The speeds may be changed while the machine is running. (7) Ratios may be obtained as high as 16:1.

Control. These units can be controlled either by hand or remote electric. When hand controlled, a handwheel is provided directly on the machine. If the machine is hung on the ceiling, a sprocket is provided on the shifting screw over which passes a chain to the hand-controlled sprocket below. When the unit is placed at a distance from the operator's position at the driven machine, the electric or remote control has been designed in order that the unit may be used to its full advantage.

SHORT-CENTER METHODS OF DRIVING

To overcome the defects of belted power transmission brought about by the short-center, open, flat-belting drive, many methods for transmitting power at close range have been devised. The ordinary, short-center, open, flat-belt drive is constantly a source of trouble due to the high belt tension necessitated and, as a rule, high ratios.

Methods.

1. V-belt.
2. Pivoted motor base.
3. Silent or roller chain.

The V-belt Drive. This is an economical and compact means of transmitting power from motor to various types of machinery. The drive consists of V-grooved cast-iron or pressed-steel sheaves in combination with one or more V-shaped belts made of fabric impregnated with rubber. The basic principle of this method is the locking effect developed as the belt seats itself in the groove. The top section is under tension and the lower one is forced into compression, thus inducing the sides of the belt to bulge, which causes the belt to grip the side walls of the groove through lateral stress.

Characteristics. V-belts are made endless and operate on finished V-grooved wheels. The drive is silent, operates without vibration, and is shock absorbing. There is a minimum of slip. The breakage of one belt does not shut down the drive. The drive can be operated at any angle and is reversible. The drive requires no oil or dressing. Belt speeds to 5000 ft. per min. are permissible.

Where the V-belt Drive Can Be Used. (1) High speeds, (2) for shock or pulsating loads, (3) where silence is necessary, (4) where cleanliness is necessary, (5) on short or medium centers, and (6) in wet and sloppy conditions.

Where the V-belt Drive Should Not Be Used. (1) In temperatures above 120°F., (2) where adjustment between centers is not possible, (3) where arc of contact on small wheel is less than 120 deg., (4) on quarter-turn drives, (5) on ratios over 7:1, and (6) where excess mineral oil or grease is present.

The Pivoted Motor-base Drive. This drive is available in either the gravity or reaction-torque types. Since the motor is free to adjust itself, the drive automatically synchronizes belt tension with the load requirements for any short-center, flat-belting installation. In addition, it corrects the effect of centrifugal tension, takes up belt stretch, and sustains maximum belt efficiency at peak loads.

The gravity type consists of a cast-iron or welded-steel base, a pivot shaft and two cast-iron or steel motor-supporting arms which oscillate on the pivot shaft. For the best results the flat belt used should be first-quality oak or mineral retanned leather made endless. Through the medium of this base and arms the motor may be mounted on floor, wall, or ceiling and can be set to give the belt tension required. Bases are available in power capacities from fractional to 500 hp.

The reaction-torque type employs a definite force inherent in all electric motors. This force, known as "reaction torque," is at any moment equal to the force exerted on the belt by the armature of the motor. In the base design of this type this force can be utilized because the motor is not attached to a fixed foundation but is suspended in a cradle and is free to rotate through a limited arc. Since this design is dependent upon reaction torque for its effectiveness and since torque acts in a circle, it will function satisfactorily when mounted on floor, wall, or ceiling. The drive must be so arranged, however, that the reaction torque will tighten, not loosen, the belt, which means that the tight or pulling strand of the belt is always nearest the pivot axis.

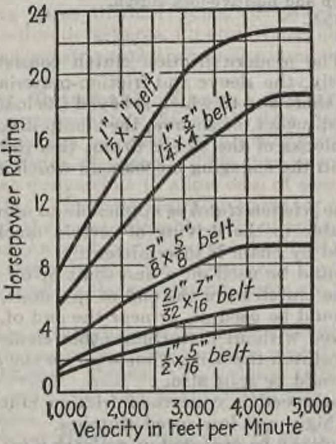


FIG. 8.—Horsepower ratings for V-belt. Ratings given are for one belt.

Silent- and Roller-chain Drives. Chains of this character provide positive short-center driving with high efficiency. Both types are employed for motor driving, but the silent chain is more extensively used for this purpose in view of its silence and smoothness of operation. Both types operate on cut-toothed sprockets.

CLUTCHES

The problem of connecting full-speed power to dead load is always present in industrial operations. In many instances it is solved by the tight-and-loose pulley, but where starting loads are severe and in cases of chain connection the clutch must be employed.

The clutch which controls the load by friction mediums is the one most universally used. There are numerous types and designs of such clutches on the market each possessing some individual characteristic as to the methods employed for engaging the friction-resistance portions. However, in the final analysis, while their mechanisms may differ, the result is the same.

Friction Clutches. Types. There are a number of friction clutches on the market which may be classified as follows:

1. Toggle and wood friction-block clutch.
2. The lined-disk and self-oiling sleeve clutch.
3. High-speed clutches.
4. The worm-and-gear-controlled clutch.
5. The friction pick-up and positive-lock clutch.
6. The coil-spring clutch.

Construction. The modern friction clutch consists generally of three principal parts, namely, the sleeve and friction-material drum or ring which operates loose on the shaft and to which is keyed the load-connecting medium such as pulley, gear, sprocket, or sheave; the clamp disk or part that engages the friction ring or blocks of the sleeve drum, this part being keyed to the power-input shaft; and the engaging mechanism which is controlled by a sliding yoke and collar.

Application. The *friction clutch* is applicable to almost any type of driving and is indispensable to the driving of certain machine tools and where the power transmitted by chain is intermittent.

The *split clutch* should be used on a line shaft or on any shaft where it is impossible to slide the clutch from the end of the shaft.

The *solid clutch* should be used at, or near the end of, a line shaft where it can be readily removed without disturbing other elements of transmission.

When using a split clutch the driving member on the sleeve such as pulley, gearing, or sheave should be split also.

Speeds. As far as speed is concerned friction clutches are designed in two classes, namely, high- and low-speed clutches.

The low-speed clutch can be operated up to 500 r.p.m., while the high-speed clutch can be safely operated at speeds as high as 2000 r.p.m.

Cut-off Friction Clutches. This mechanism, sometimes termed a *friction cut-off coupling*, is similar in construction to the regular friction clutch with the exception that it has no sleeve. It is used chiefly for connecting two lengths of shafting in such a manner that one section can be stopped or started at will while the other section is operating. The cut-off clutch makes possible a segregation or separate control of the line-shaft equipment as required for different departments, rooms, or floors of a manufacturing plant.

The cut-off clutch is the most effective means for the prevention of accidents and injuries to workmen by making possible the immediate stoppage of shafting or machinery.

Jaw Clutches. The jaw clutch is a convenient method for connecting or disconnecting a load where shock is of no consequence and the speed is low. Therefore it is usually employed for very rough driving. It should not be considered for the average industrial transmission system and not be used on speeds exceeding 60 r.p.m.

It is constructed with either square or spiral jaws, and either type may be equipped with a sleeve to hold the driving member. This type of clutch may also be used as a cut-off coupling.

CHAIN DRIVING

Selection. The transmission of power by chain has progressed rapidly, so that at the present time there is a type and design for almost any installation. The selection of type, therefore, is of the utmost importance because each style of chain manufactured is for a specific field and when misapplied is costly.

Advantages. The general advantages of chain driving are: (1) Certain types are approximately 98 per cent efficient. (2) Relatively high speeds can be obtained. (3) Not affected by heat, cold, or moisture. (4) Certain types will transmit within reason any amount of power. (5) Give a positive velocity ratio. (6) Can be used on short or long centers. (7) Chains cannot slip.

Types. There are three distinct types of power-transmission chains, namely: (1) malleable-iron detachable, (2) steel roller, and (3) the silent chain. Each type has its own construction and applications, and, therefore, will be discussed separately.

1. Malleable-iron Detachable Chain. Construction. This type is manufactured in approximately 25 stock sizes designated by number. Special sizes and designs can be obtained, but they are generally for purposes other than the transmission of power. This type is composed of individual malleable-iron links so designed as to allow ease of assembly.

Ratios and Speeds. The most desirable speed for this type is 400 ft. per min. and under, and the most desirable ratio of sprockets is not more than 5:1. Higher ratios have a tendency to shorten the life of the chain.

Application and Capacities. This type finds application in practically every form of industry where the speeds and ratios are held within the limits mentioned.

The power capacities of this chain as to tensile strength and horsepowers are usually given in catalogues of the various chain manufacturers.

2. Steel-roller Chains. Types. This chain is manufactured in three styles: (*a*) the light-roller, (*b*) the heavy-steel-roller, and (*c*) the finished steel-roller power chains.

(a) **Light Steel-roller Chain.** This type of chain possesses three times the tensile strength of the malleable-iron type and operates on cast tooth sprockets. It is constructed with medium-carbon hot-rolled-steel side bars, the pins and bushings are of hardened steel, and the rollers are of either malleable iron or hardened steel.

The most desirable speed for this type is 700 ft. per min. or under, and the most desirable ratio of sprockets is not more than 5:1.

This type is applicable in practically every industry where the speeds and ratios are held within the limits mentioned.

Power capacities are generally given in catalogues of the manufacturers.

(b) **Heavy Steel-roller Chain.** This type of chain is of more rugged construction and operates on cast tooth sprockets. It is constructed with medium-carbon hot-rolled-steel side bars, the pins and bushings are of hardened steel, and the rollers are of either malleable iron or hardened steel.

The most desirable speeds for this type of chain is 700 ft. per min. or under, and the most desirable ratio of sprockets is not more than 5:1.

This type is applicable in practically every industry where the speeds and ratios are held within the limits mentioned.

The power capacities can be obtained from catalogues of chain manufacturers.

(c) **Finished Steel-roller Chain.** This type is intended for general power-transmission purposes and is designed to operate on accurately cut sprocket wheels. It is much more expensive, but it has demonstrated its economy compared with the less costly chains which operate on cast tooth wheels.

It is constructed with special rolled-steel heat-treated side bars, the pins are made from alloy steel hardened, and the bushings are steel, case hardened. The rollers are high-carbon steel, heat treated.

It is possible to operate this type of chain at speeds as high as 1000 ft. per min. However, it is more desirable from an economical and long-life standpoint to limit its operating speed to approximately 700 ft. per min. This type allows a wide range of ratios and can be operated successfully as high as 10:1. However, the average commercial-ratio requirements vary up to 7:1.

Based on its strength and accuracy of design, this type of chain is applicable to all manner of industrial driving within the speeds and ratios mentioned. This includes driving by motor.

The ultimate or breaking tensile strengths of finished steel-roller chains are given in catalogues of the various chain manufacturers.

3. **Silent Chain.** This is a distinct type of power-transmission chain and is manufactured for almost any reasonable power capacity with a sustained efficiency of approximately 98.5 per cent.

It is manufactured in several types, but the difference of design is almost entirely in the construction of the pin or joint connection of the links.

There are two types of what may be termed rocker pin construction—one-pin and segmental-bushing joint, and a number of single-pin bushless joints.

All types of silent chains operate on cut-tooth-steel or cast-iron sprockets, and the permissible speeds and ratios are similar for all.

The most desirable speed for the silent chain is 1200 to 1500 ft. per min. At speeds higher than 1500 ft. per min., the silent chain should be encased to assure proper lubrication. The most desirable ratios for silent-chain driving ranges from 1:1 to 7:1. Ratios as high as 15:1, however, have been successfully employed.

The power capacities for silent chains are thoroughly given by tables, etc., in data books furnished by the various manufacturers of this type of chain.

Lubrication of Power Chains. The severe conditions under which most chains operate make exacting demands not only on the manufacturer of the chain but on the user. The former produces the best chain possible, and the consumer must also do his best to keep the chain clean and well lubricated.

Plastic lubrication such as grease and non-fluid oil should be used on the malleable-iron and unfinished-steel-roller chains. Fluid lubrication such as mineral oil should be used on finished-steel-roller and silent chains. Best results are obtained when these types are encased in what is known as an oil-retaining casing. These casings are supplied by the manufacturer of the chain.

LUBRICATION OF POWER-TRANSMISSION EQUIPMENT

Whether bearings, gears, or other relatively moving mechanical elements are being dealt with, complete lubrication implies the separation of the opposed working parts by an oil film which transmits forces from one part to another and prevents contact of metal on metal. Correct lubrication implies further that the fluid film consists of a correctly selected oil of a quality suited to the service, whether the service be one in which the oil performs its duty but once and goes to waste, or one in which it must perform its duty repeatedly over long periods of time.

The factors which are essential to the building of the oil film are: (1) The presence of a wedge-shaped clearance space, free from grooving such as might interfere with formation of an oil film; (2) speed or relative motion of the parts by which the carrying action is accomplished; (3) adhesiveness of the oil to assure that it will follow the moving surfaces; (4) a certain degree of oil body or viscosity—a characteristic which resists the too rapid escape of the oil from the film; (5) lubricating value of the oil which largely determines the strength of the oil film under pressure.

There are also conditions which tend to break down the oil film, chief of which are (1) heavy bearing loads, (2) high temperatures. Heavy bearing loads create intense pressures. If the carrying effect of speed or the adhesiveness or body of the oil is insufficient, a heavy load will prevent the formation or maintenance of a complete oil film.

The effect of increased temperature on all oils is to decrease their body or viscosity. With any given oil there is therefore a temperature beyond which the satisfactory lubrication of a given bearing becomes impossible. The maintenance of lubrication therefore depends on securing a satisfactory balance between the film-forming factors of oil wedge, speed, adhesiveness, oil body, and lubricating value, and the film-destroying influences of pressure and temperature.

Efficient lubrication is created and maintained by the action of an oil wedge, and this wedge must be maintained by a regular oil supply. In modern transmission equipment there are three ways in which the oil is supplied:

(1) By the ring, chain, collar, capillary, splash systems, circulating and pressure systems, when the same oil is to serve repeatedly; (2) by the automatic system such as drop-feed, wick-feed, and bottle oiler, when the oil loss must be compensated for; (3) by the hand-application system, when the need is intermittent, periodic, or irregular. When the parts cannot be lubricated by any available method of oil application, it may be necessary to use grease. Grease cups can at times be applied where an oiling device cannot.

HIGH-STARTING TORQUE CONTROL

Direct-connecting a motor to a load, particularly when high ratios are involved, generally necessitates rigid connecting methods. In view of the absence of mechanical slip with rigid connections, the motor is required to come to full speed under the applied load of the driven apparatus. The torque required to start a heavy applied load sets up destructive strains in both the electrical and the mechanical equipment, unless relieved by either electrical or mechanical devices. Economically it is desirable to use standard induction motors with "across the line" start, since such a start permits the motor to come to full speed in a few seconds provided the load is moderate or light. If the load is heavy, severe starting torque strains are produced. This condition can be relieved by special motors and electric starting equipment or by mechanical devices interposed between motor and load.

Electrical Methods. *A.C. Slip-ring Motor.* This type accelerates gradually under load in view of the use of collector rings and adjustable rheostats. The acceleration in speed is procured by wasting energy in resistance.

A.C. High-starting-torque Motor. This type differs from the standard induction motor in that it has two windings, one of which controls the resistance during starting and the other the reactance during running. The starting torque of the standard induction motor is approximately 175 per cent of the full-load torque, whereas for a light starting torque motor it is approximately 250 per cent.

D.C. Motor. This type possesses large starting torque capacity. The series motor can be used when the load is always connected to the motor and when the constancy of speed with variations in load is not especially desired. The series motor has a rapid rise in speed at high load; *i.e.*, if the load is not connected, the motor will race.

D.C. Compound Wound Motor. The advantage of this type is that at starting, when the current through the armature and series field winding is large, the total field excitation is large; hence there is an increase in torque capacity. Another advantage over the series motor is that it will not speed up indefinitely when released from load. A disadvantage is that its running speed decreases considerably with load increase.

Mechanical Methods. During the past few years, mechanical methods for overcoming high starting-torque resistance have been perfected. The number in actual use is relatively small when compared with the special motor equipment available. However, in view of many advantages, the mechanical is becoming a competitor of the electrical.

Advantages. (1) Elimination of expensive electrical starting equipment, (2) minimum size of mechanical transmission equipment, (3) economy in current where starting and stopping are frequent, (4) possibility of using standard induction across-the-line start motors, (5) the motor horsepower size can be based on the running instead of the starting load, (6) reduction in starting shock, and (7) overload release capacity.

Self-actuating mechanisms for high starting-torque service are of two types: those that serve as a coupling (combining flexibility) between motor and load and those that act integrally with belt pulleys, chain sprockets, gears, and V-belt sheaves. The former are known as high starting-torque couplings, and the latter as pulley-, sprocket-, or gear-type starters.

The Mechanical Slip-ring Starter. This is a unit which by slipping friction automatically starts machinery smoothly and with uniform acceleration from across-the-line start motors regardless of the starting torque involved. It consists of a spider, two friction bands, and a drum. The spider or driving element is keyed to the motor shaft, and the drum or driven element is keyed to the shaft of equipment to be driven. The weighted bands which lie inside the drum are secured at one end to the spider. The opposite or trailing ends of the bands are free. As the motor speeds up, centrifugal force causes the bands to expand and exert a frictional drag on the inside of the drum. The torque thus transmitted is dependent upon motor speed, diameter of drum, and the weight of the bands.

Performance. The mechanical slip-ring starter is designed to transmit any required torque at a specified speed in r.p.m. For any application the full-load speed of the motor and the maximum horsepower to be transmitted must be known. The device is usually proportioned to transmit a maximum of 125 per cent of the rated horsepower of the motor. The device can, under normal conditions, remain in a stalled condition, *i.e.*, with the motor revolving at full speed and the drum or driven element held stationary for a few minutes without injury to the device or motor. When the driving shaft is at rest, it is physically disconnected from the driven shaft. When power is applied, and as the motor increases in speed, centrifugal force acts on the bands and therefore develops very gradually a connection between the driving and driven shafts. When the driving shaft reaches full speed, the device is transmitting its maximum predetermined torque to the driven shaft. The driven shaft eventually reaches its full speed. During the period of acceleration, a gradually diminishing amount of slippage is occurring in the device, but, when the driven shaft reaches full speed, all slippage ceases, and the functioning is that of an ordinary flexible coupling, with the additional protection of slippage, in the event of overload. The device is inherently a limit-torque coupling, since it will not transmit to the driven machine, or impose on the motor more than the maximum load for which it is designed.

General Applications.

1. High inertia starting loads, such as are encountered in centrifugals, extractors, etc., where the power required to start is frequently five or six times the actual running load.

2. Where starting is difficult owing to excessive bearing friction, the device assists any motor by permitting almost full speed before the application of the load. This allows the use of motors with low starting-torque characteristics.

Horsepower and Speeds for Mechanical Slip-ring Starters
Coupling and Integral Types

Size of starter	R.p.m.						
	495	600	685	750	870	1160	1750
5½ × 2½	0.16	0.3	0.45	....	0.91	2.15	7.5
6 × 3	.42	0.75	1.13	1.47	2.3	5.5	19.1
8 × 2½	.95	1.63	2.5	3.27	5.1	12.1	42.2
9 × 3	1.92	3.4	5.1	6.68	10.4	24.7	86.5
10 × 3½	3.27	5.8	8.7	11.4	17.7	42.0	147.0
10 × 4	3.75	6.65	10.0	13.1	20.3	48.0	169.0
11 × 5	7.5	13.4	20.0	36.2	40.7	95.7	338.0
12 × 3½	6.8	12.2	18.2	24.0	37.5	89.0	310.0
12 × 4½	8.6	15.2	22.8	30.0	46.5	111.0	377.0
12 × 6	12.4	22.0	33.0	43.0	67.0	159.0	555.0
14 × 5	19.6	34.8	52.2	68.2	106.0	253.0	
15 × 5	24.2	43.0	64.5	84.0	131.0	312.0	
16 × 6	41.0	73.0	109.0	143.0	223.0	530.0	
18 × 6	59.5	105.0	158.0	207.0	322.0	765.0	
18 × 9	89.5	158.0	237.0	310.0	485.0	1150.0	
18 × 12	119.0	210.0	315.0	413.0	645.0	1530.0	
20 × 6	85.0	150.0	225.0	295.0	460.0	1090.0	
20 × 9	127.0	225.0	338.0	445.0	690.0	1640.0	
20 × 12	170.0	302.0	452.0	590.0	920.0	2180.0	
24 × 12	258.0	460.0	690.0	900.0	1410.0	3300.0	
26 × 12	438.0	780.0	1160.0	1520.0	2370.0	5620.0	
28 × 10	750.0	1330.0	2000.0	2520.0	4080.0	9700.0	
28 × 12	900.0	1610.0	2400.0	3150.0	4900.0	11600.0	

Note. Direction of rotation of driving shaft must always be given.

3. The torque-limiting feature of the device is useful in many cases where sudden running overloads or complete stalling of the motor is likely. It is possible to build the device to transmit a maximum of 150 to 200 per cent of the motor rating, so that, in the event of the driven machinery jamming, the device will slip and allow the motor to continue at full speed until the overload relay takes the motor off the line.

4. In the driving of ball, pebble, and tube mills, where the maximum running load can be definitely determined, improved starting and operation can be obtained by selecting a motor that is just equal to the running conditions and by applying a mechanical slip-ring starter of approximately 15 per cent more horsepower capacity. Such a combination produces jerkless starting with across-the-line control and permits the size of the mechanical power equipment to be proportioned to the running instead of the starting load.

5. The device makes possible the use of two-speed a.c. motors for very gradual starting of heavy loads instead of d.c. or complicated a.c. arrangements. It takes care of the acceleration up to the low-speed setting of the motor. The motor control then handles the load from low to high speed.

6. With heavy inertia loads it is at times impossible to use synchronous motors because of their inability to "pull in." This results in overmotoring. In such cases the mechanical slip-ring starter, by eliminating the inertia factor WR^2 of the driven machine, enables a motor of the correct size for the running load to pull in with ease.

Permissible Speeds. On many low-speed motors the mechanical slip-ring starter is practicable, but its proportions become large because of low centrifugal force. The most adaptable speeds are from 600 to 1800 r.p.m., although 3600 r.p.m. is possible.

Power Capacity. Horsepower capacities range from fractional to any practical requirement. Mechanically there is no limit.

SECTION 23

REFRIGERATION

BY

H. C. Duus, Ph. D., Member, American Institute of Chemical Engineers, American Chemical Society, Society of Automotive Engineers. Chemical Engineer, E. I. du Pont de Nemours & Co.
L. H. Morrison, B. S. in M. E., Business Journals.

REVISED BY

H. C. Duus

CONTENTS

REFRIGERATION

REFERENCES: Goodenough, "Principles of Thermodynamics," Holt, New York, 1920. Lorenz-Heinel "Neuere Kühlmaschinen," Oldenbourg, Munich and Berlin, 1913. Macintire, "Handbook of Mechanical Refrigeration," Wiley, New York, 1928. Morrison, "Power's Practical Refrigeration," McGraw-Hill, New York, 1928. Moyer and Fittz, "Refrigeration," McGraw-Hill, New York, 1932. Plank, "Amerikanische Kältetechnik," V.D.I. Verlag, Berlin, 1929. Siebel, "Compendium of Mechanical Refrigeration," Nickerson and Collins, Chicago, 1918. Instructions for Operation, Care and Repair of Refrigerating Plants, Navy Dept. Bur. Eng., Government Printing Office, Washington, 1918. Quinn and Jones, "Carbon Dioxide," Reinhold, New York, 1936. Periodicals: *Refrig. Eng.; Ice and Refrigeration*; "The Refrigerating Data Book," Vol. I (1939–1940), Refrigerating Principles and Machinery; Vol. II (1940), Refrigeration Applications, Am. Soc. Refrig. Engrs.

Refrigeration as considered in the following may be broadly defined as the art of producing cold, referring particularly to cooling below atmospheric temperature. The means most commonly employed for such cooling is to induce a change of phase in a heat-abstracting body such as is involved in the vaporization of liquid ammonia or the melting of ice. In the production of liquid air, however, cooling is brought about by expanding the gas through a nozzle (Joule-Thomson effect) or by causing it to do work against a brake (Claude system). Other physical changes, such as the contraction of stretched rubber or the extension of a steel spring, the passage of an electric current through a bimetallic junction, and, in fact, any reversible physical change involving the expenditure of work, are capable of producing cold. Radiation of heat from the surface of the earth to interstellar space may be used in certain localities for freezing water, and cool ground waters have from time immemorial been used for preserving foodstuffs.

Definitions.

A **refrigerant** in its broadest sense may be defined as any material which is used for abstracting heat. In a narrower, but more commonly used, sense the term refers only to those materials which are used in **mechanical refrigeration.**

Mechanical refrigeration includes those processes in which the refrigerant is recovered and recirculated, as distinguished from those in which the spent refrigerant is wasted (ice refrigeration, ice-cream freezing by salt-ice mixtures, cold ground waters in spring houses). Mechanical refrigeration falls into two general groupings, depending on the method used in recirculating the refrigerant.

In the **compression** system (dense air or vapor-compression machines) a compressor is used, which may have either a positive displacement mechanism (reciprocating or rotary compressor) or an impeller (centrifugal compressor). While the thermodynamic cycle is the same for both types of compression, the kinematic considerations, particularly with regard to the refrigerant used, are markedly different. The dense-air machine operates on the Carnot cycle and the vapor-compression machine on the reversed Rankine (steam-engine) cycle.

The **absorption system** differs essentially from the compression system in requiring no positive work input to produce cold though generally requiring some accessory power. Circulation is effected by absorption of the refrigerant in appropriate liquids or solids which are regenerated by heat in another part of the refrigerating system. Absorption machines are classified as **continuous** or **intermittent,** the latter corresponding quite closely to batch processes. All continuous machines use liquid absorbents because of the great practical difficulties in moving a solid absorbent about from the absorption side to the generator. When continuous refrigeration is required with intermittent units, two or more machines must be installed.

This classification is neither absolute nor all-inclusive. Combinations of absorption and compression systems have been developed (Westinghouse-Leblanc). The processes for manufacturing liquid air or solid carbon dioxide do not fall into the above scheme since they employ compression but do not recirculate the refrigerant. The distinctions are useful, however, in considering the operations involved.

Practically, mechanical refrigeration is the most important branch of the art, and developments have centered almost wholly about the vapor-compression machine.

Absorption machines still find considerable industrial application where large quantities of waste heat are available. Interest has also been revived in absorption systems tied in with heating arrangements for year-round air conditioning, involving summer cooling and winter heating. The main investigations have been in the direction of developing refrigerant-solvent combinations which can produce refrigeration with a minimum heat expenditure. [Zellhoefer, *Refrig. Eng.*, **33**, 317 (1937).]

Units of Refrigeration. The unit of refrigeration in the United States is the **standard ton** of 288,000 B.t.u. which is very nearly equal to the heat of fusion of 2000 lb. of ice at 32°F. The **standard commercial ton of refrigeration** is at the rate of 200 B.t.u. per min., 12,000 B.t.u. per hr., or 288,000 B.t.u. per 24 hr. Note that the standard ton has the dimensions of heat, while the standard commercial ton has the dimensions of heat divided by time. The **standard rating of a refrigerating machine,** which applies only to compression and absorption systems using a condensable vapor, is the number of commercial tons of refrigeration it performs under certain prescribed conditions. These conditions are:

1. Nothing but liquid shall enter the expansion valve, and nothing but vapor shall enter the compressor cylinder (impeller in a centrifugal compressor) of the compression refrigerating system or the absorber of the absorption system.
2. There shall be 9°F. (5°C.) subcooling of the liquid entering the expansion valve and 9°F. (5°C.) superheating of the vapor entering the compression cylinder or the absorber. The points at which subcooling and superheating are determined must be within 10 ft. of the cylinder or absorber.
3. The inlet pressure is that which corresponds to a saturation temperature of 5°F. (−15°C.).
4. The outlet pressure from the compressor cylinder or generator is that which corresponds to a saturation temperature of 86°F. (30°C.).

The **British unit of refrigeration** is based on a rate of cooling of 1 kg.-cal. per sec. or 237.6 B.t.u. per min., with inlet pressures corresponding to a saturation temperature of 23°F. (−5°C.) and outlet pressures corresponding to a saturation temperature of 59°F. (15°C.).

The Dense-air Machine. From a thermodynamic standpoint, the dense-air machine operating on the Carnot cycle is capable of the highest efficiency. In practice, the efficiency is low because of the large volumes of air which must be handled for a relatively small effect. The safety of air as regards toxicity, flammability, and odor was an important factor in its favor some years ago, but in recent years refrigerants have been developed which are practically as safe and more efficient. For operating details, the reader may consult U.S. Navy, "Manual of Engineering Instructions," Chap. 17. Government Printing Office, Washington.

DESCRIPTION AND THEORY OF OPERATION OF VAPOR-COMPRESSION AND ABSORPTION REFRIGERATION SYSTEMS

The basic principle involved in both systems is that of transferring heat from an environment at low temperature to one at a higher temperature, by causing a volatile liquid (the refrigerant) to absorb heat at the low temperature by vaporization and to dissipate this heat at the high temperature by condensation. Vaporization and condensation are respectively induced by maintaining a lower or higher pressure than the saturation pressures of the refrigerant at the lower and higher temperatures. Both systems employ an **evaporator** (expansion coil) and a **condenser.** The apparatus used for transferring the refrigerant from the low- to the high-pressure side is technically the **refrigerating machine,** but the term is not always used in strict conformity with this definition. In the vapor-compression system this machine is actuated by a prime mover and in the absorption machine by heat. Economy requires that a given amount of refrigeration be produced with a minimum expenditure of work or heat.

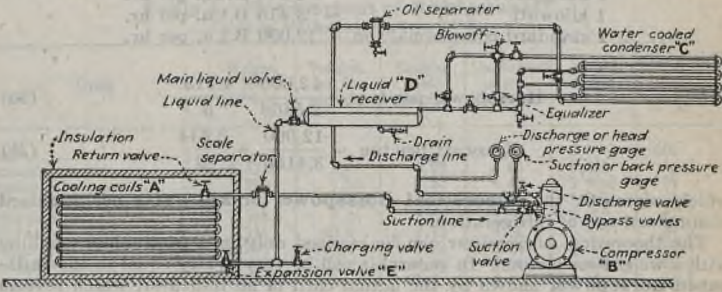


FIG. 1.—Simple vapor-compression machine.

The Simple Vapor-compression Refrigerating Machine. A diagrammatic representation of a simple compression system of refrigeration is shown in Fig. 1. Refrigerant vapor is drawn from the expansion coil A at the pressure p_1 by the compressor B; forced into the condenser C at a pressure p_2, dependent on the temperature of the cooling water, where the vapor liquefies; and collected in the receiver D from which it returns through the expansion valve E, under the pressure difference $p_2 - p_1$, into the expansion coil.

In this process the following heat quantities are involved, which may be taken from the tables of thermodynamic properties:

H_b = heat content of vapor leaving evaporator (expansion coil).
H_c = heat content of vapor leaving compressor.
h_e = heat content of liquid entering evaporator (expansion coil).

If the compression is adiabatic (isentropic) the work expended is

$$H_c - H_b$$

The net heat abstracted is

$$H_b - h_e$$

Note that this is the heat of evaporation less the heat required to cool the liquid refrigerant from the temperature of the receiver to the temperature of the expansion coil. The ratio of the net heat abstracted to the work expended is known as the performance coefficient β and may be written,

$$\beta = \frac{H_b - h_e}{H_c - H_b} \tag{1}$$

The inverse of this ratio is the efficiency coefficient E of the simple steam engine or turbine.

The horsepower and the kilowatts required per standard commercial ton of refrigeration follow directly from the definition of β and the conversion factors:

1 horsepower = 2,546 B.t.u. per hr.
1 kilowatt = 3,415 B.t.u. per hr.
1 standard commercial ton = 12,000 B.t.u. per hr.

This gives

$$\text{Horsepower per ton} = \frac{12{,}000}{2{,}546\beta} = \frac{4.713}{\beta} \tag{2a}$$

$$\text{Kilowatt per ton} = \frac{12{,}000}{3{,}415\beta} = \frac{3.514}{\beta} \tag{2b}$$

which represent the **theoretical horsepower** or **kilowatts** per standard commercial ton of refrigeration.

The theoretical horsepower can be realized only in a frictionless machine with a weightless piston. In general it will be very nearly equal to the **indicated horsepower** shown by the actual card of an indicator, which reveals the imperfections in the operation of the valves and the departure from strictly adiabatic compression. The **brake horsepower** is the indicated horsepower plus the power necessary to overcome the friction in the cylinder walls and bearings and the inertia of the piston. The brake horsepower will be 10 to 20 per cent more than the theoretical horsepower. The difference between brake horsepower and indicated horsepower is sometimes designated as **friction horsepower.** The friction horsepower is relatively larger in small than in large machines.

Since refrigeration involves a conversion of work into heat it might appear that the performance coefficient is dependent only on the temperature difference and independent of the refrigerant used. This is only approximately true as shown by Table 1. The temperature difference is, however, important, and consequently, a machine tested according to the standard American temperature interval will show much better performance when tested on the British standard.

Sample Calculation of Performance Coefficient. Assume that the refrigerant is ammonia; the absolute evaporator pressure (p_1) 35 lb. per sq. in. corresponding to 5.89°F., and the absolute condenser pressure (p_2) 160 lb. per sq. in. corresponding to 82.64°F. (see Table 14).

The heat content (H_B) of the saturated vapor in the evaporator is 613.6 B.t.u. per lb. and the entropy (S) is 1.3236 B.t.u. per lb. per °F.

To determine the heat content (H_C) of the vapor in the condenser, find the heat content of the superheated vapor corresponding to 160 lb. per sq. in. and entropy 1.3236. This is 707.5. The temperature of the superheated vapor is 199.5°F. (See Table 16, p. 2555.)

The work of compression per pound of vapor is therefore

$$H_C - H_B = 707.5 - 613.6 = 93.9 \text{ B.t.u.}$$

The heat content of the liquor (h_e) entering the evaporator at 82.64°F. is 135.0 B.t.u. (Table 14). Hence the net heat extracted is

$$H_B - h_e = 613.6 - 135.0 = 478.6 \text{ B.t.u.}$$

The performance coefficient is

$$\beta = \frac{478.6}{93.9} = 5.097$$

and

$$\text{Horsepower per ton refrigeration} = \frac{4.713}{5.097} = 0.925$$

Table 1. Comparison of Refrigerants*

One-ton refrigeration, 5 to 86°F.

Cycle	Weight, lb. per min.	Volume, cu. ft. per min.	Ratio of compression	Coefficient of performance	Horsepower per ton	Relative efficiencies, %
Ideal	......	....	.....	5.74	0.8214	100
Ammonia	0.4214	3.44	4.93	4.85	0.973	84.5
Propane	1.396	3.35	3.64	4.88	0.9668	85
Carbon dioxide	3.74	0.999	3.11	2.56	1.843	44.6
Sulfur dioxide	1.388	9.24	5.63	4.735	0.995	82.5
Ethyl ether	1.555	60.8	7.12	4.86	0.971	84.6
Dichloroethylene	1.768	108.4	8.23	5.14	0.918	89.4
Trichloroethylene	2.137	513	10.84	5.085	0.928	88.5
Water	0.1996	1972	21.9	4.1	1.15	71.5

* Carrier and Waterfill, *Refrig. Eng.*, **10**, 415 (1923).

Table 1 gives the theoretical performance coefficient for ammonia, which will be found to hold approximately for all other refrigerants except those boiling much below ammonia. Carbon dioxide, the notable exception, will

Table 2. Large Enclosed York V. S. A. Compressors with Ammonia and Freon

Evaporator temperature, 35°F., condenser temperature, 110°F.

Size	No. of cylinder	Speed, r.p.m.	Capacity, tons	Brake horsepower per ton
		Ammonia		
7½ × 7½	2	360	55.2	1.10
8 × 8	2	360	68.4	1.10
9 × 9	2	300	84.3	1.07
10 × 10	2	300	118.7	1.07
11 × 13	2	277	173.0	1.07
12½ × 14½	2	257	234.0	1.06
		Freon		
9 × 7½	2	360	43.7	1.27
11 × 9	2	300	66.5	1.23
12½ × 10	2	300	95.4	1.215
14 × 10	2	300	119.6	1.20
13½ × 13	2	277	133.7	1.20
15 × 13	2	277	165.1	1.19

require a greater power expenditure. The brake horsepower, arbitrarily taken as 10 per cent greater than the theoretical is shown in Figs. 2 and 3.

A very complete study of the efficiencies of Freon and ammonia was made by Williams [*Refrig. Eng.*, **31**, 36, (1936)]. The results are given in Table 2.

In case the thermodynamic tables are not available, the horsepower required to compress the vapor adiabatically (isentropically) may be calculated from the formula

$$\text{Horsepower} = \frac{144k}{33{,}000(k-1)} p_1 v_1 \left[\left(\frac{p_2}{p_1} \right)^{\frac{k-1}{k}} - 1 \right]$$

where p_1 = absolute intake pressure (35 lb. in above example).
p_2 = absolute discharge pressure (160 lb. in above example).
v_1 = volume compressed, cu. ft. per min.
k = the ratio of specific heat of the vapor, c_p/c_v.

The volume v_1 may usually be found with sufficient accuracy from the general gas law equation expressed in the appropriate units.

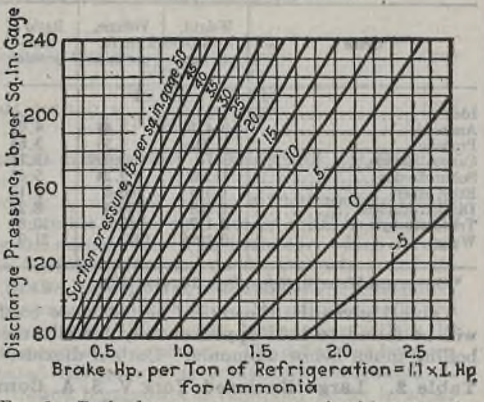


FIG. 2.—Brake horsepower per ton of refrigeration for ammonia.

The net heat extracted is similarly the difference between the heat of evaporation at the evaporating temperature and the sensible heat of the liquid between the condenser temperature and the evaporator temperature (specific heat times temperature difference).

While the performance coefficients of refrigerants having critical temperatures well above that of the available cooling water will be approximately equal for a given temperature interval, the degrees of superheating will vary greatly. In an adiabatic compression the following relation holds:

$$\frac{T_2}{T_1} = \left(\frac{p_2}{p_1} \right)^{\frac{k-1}{k}}$$

where T_1 and p_1 = absolute suction temperature and pressure.
T_2 and p_2 = absolute discharge temperature and pressure.
k = ratio of specific heats.

The compression ratio p_2/p_1, which is determined by the latent heats, will be nearly the same for all refrigerants (Trouton's law) while the specific heat ratio will be lowest for those refrigerants having high molal heat capacities. The refrigerants having complex molecules will therefore have a much smaller temperature rise in adiabatic compression than those having simple molecules. In the case of ethyl ether ($C_4H_{10}O$) there is actually no superheating but super-cooling and liquefaction, while ammonia (NH_3) and sulfur dioxide (SO_2) give the greatest superheating on compression.

Indicator Card and Volumetric Efficiency. The work of compression is shown by the indicator diagram (Fig. 4). In the ideal case the piston at the beginning of the suction stroke touches the cylinder head and the volume is zero. As the piston moves out, vapor is drawn in and the volume increases at the constant pressure p_1 along the line *AB*. The vapor is now compressed adiabatically along the line *BC* until the pressure equals the condenser pressure p_2, after which it is forced into the condenser and the volume decreases along the line *CH*. The area *ABCH* is then a measure of the compression work.

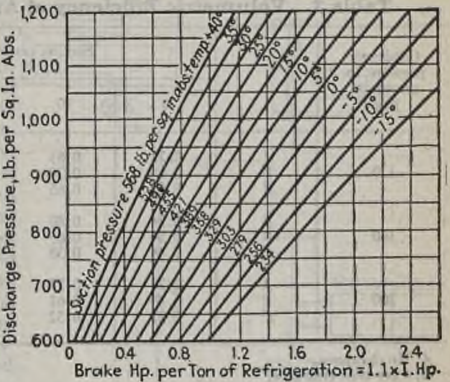


FIG. 3.—Brake horsepower per ton of refrigeration for carbon dioxide.

In an actual compressor it is necessary to provide a small clearance at the end of the compression stroke, represented by the segment *EH*. Before suction can actually start, the vapor enclosed in this space expands adiabatically along the line *EF*. The work in an actual compressor is therefore represented by the area *FBCE* which is smaller than the area *ABCH*. However, the volume of gas handled by an actual compressor is less than by an ideal compressor, the ratio being given by the ratio of the lines *FB*/*AB*. The work represented by area *EF'F* is the excess of work required by an actual compressor handling a volume of gas corresponding to the line *FB*. This excess will become a smaller and smaller fraction, as the clearance represented by the line *EH* is decreased.

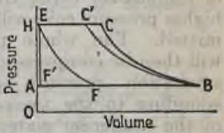


FIG. 4.—Ideal indicator diagram for vapor-compression machine.

If the compression were isothermal instead of adiabatic, it would follow a line *BC'* lying below the adiabatic *BC*. The work requirement would therefore be less. In practice this cannot be attained completely, but by compressing the vapor in two or more stages and cooling the superheated vapor the isothermal line may be approached (see section on Stage Compression, p. 2535.)

The **volumetric efficiency** is the ratio of vapor volume handled by an actual compressor to the volume handled by an ideal compressor of the same piston displacement and would therefore be equal to the ratio *FB*/*AB*. In practice the ratio will be lower due to superheating of the vapor entering the cylinder, leakage, etc., but reliable manufacturers will guarantee above 80 per cent. Volumetric efficiencies of ammonia compressors are shown in Table 3.

For a centrifugal compressor the compressor work and volumetric efficiency do not admit a graphical representation as does the reciprocating compressor. A centrifugal compressor has no clearance and therefore a 100 per cent volumetric efficiency. However, there are slippage losses around the impeller vanes which correspond roughly to clearance losses. The operation of a reciprocating compressor may be compared to a series of batch processes, while the operation of a centrifugal compressor is more like a continuous process.

Table 3. Volumetric Efficiency of Ammonia Compressors

Condenser pressure, lb. per sq. in. gage		Suction pressure, lb. per sq. in., gage				
		0	10	20	30	40
120	*A*	0.77	0.83	0.87	0.89	0.91
	B	0.60	0.70	0.77	0.81	0.84
	C	0.52	0.65	0.72	0.77	0.80
160	*A*	0.74	0.80	0.83	0.86	0.88
	B	0.54	0.65	0.72	0.76	0.80
	C	0.44	0.58	0.66	0.72	0.75
200	*A*	0.71	0.77	0.81	0.84	0.86
	B	0.49	0.61	0.68	0.72	0.75
	C	0.37	0.52	0.62	0.67	0.71

A, No clearance.
B, 4 per cent clearance.
C, 6 per cent clearance.

Dual Compression. When two refrigerating temperatures are required, the **dual-compression** system may be employed. The general principle of this design is to draw in the vapor from the low-pressure (low-temperature) expansion coils, and, at or near the end of the stroke, vapor from the higher pressure coils will be admitted. The whole mixture will then be compressed to the saturation pressure corresponding to the temperature of the condenser water. The advantages of this system are the use of one condenser for the two cooling systems and a markedly higher capacity, since the low-pressure cooling unit requires much larger piston displacement than the higher pressure unit. The chief disadvantage is the difficulty of securing continuous adjustment between the loads on the two systems. Two schemes of dual compression (Voorhees and Windhausen) with the corresponding indicator cards are shown in Fig. 5. In some household units, the dual-compression principle is employed to maintain two temperatures, one for freezing ice cubes and a higher one at the optimum value for food preservation.

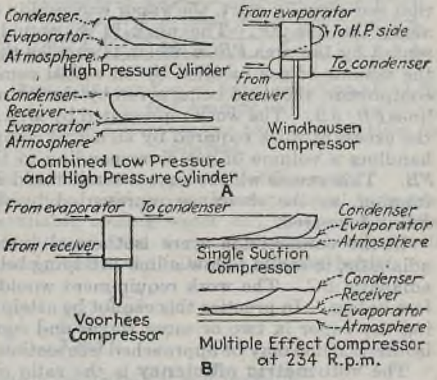


FIG. 5.—Indicator diagrams for Voorhees and Windhausen systems of dual compression. (*Reprinted by permission from "Handbook of Mechanical Refrigeration," by H. J. Macintire, John Wiley & Sons.*)

Wet Compression. In wet compression liquid refrigerant is admitted with the refrigerant vapor from the expansion coils. This liquid evaporates during adiabatic compression and thereby makes the compression approach more nearly to the isothermal line. The vapor leaves the compressor with a much smaller degree of superheat which also lessens the load on the condenser. While the scheme is sound in theory, it is difficult in practice to adjust the amount of liquid added, and the capacity is very much reduced owing to the increased cylinder volume required by the liquid after vaporization. The scheme can naturally not be used with a refrigerant like ethyl ether, which liquefies on adiabatic compression, and would not be particularly advantageous for those refrigerants which superheat slightly.

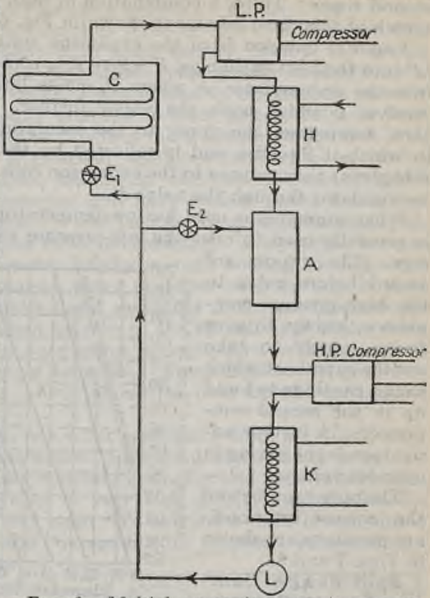


Fig. 6.—Multiple-stage (two-stage) compression. (*Macintire, "Handbook of Mechanical Refrigeration," Wiley.*)

Multistage Compression. The most promising developments toward diminishing the power requirements of refrigeration are along the lines of multistage compression. This principle has long been used in air compression, but in refrigeration the problem is more complicated. In general it will prove economical with ammonia only where suction pressures lower than 5 lb. per sq. in. (gage) are encountered. As in the case of wet compression, it can be applied with advantage only to those refrigerants which superheat markedly on adiabatic compression (ammonia, sulfur dioxide, and carbon dioxide but not ethyl ether, dichlorodifluoromethane, propane, and butane).

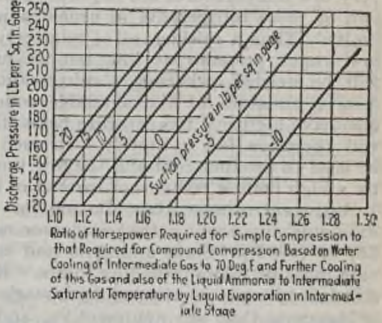


Fig. 7.—Power saving by multiple-stage compression, compared with single-stage compression and evaporation.

Several possibilities in multiple compression are presented. First, the vapor from the first stage may be cooled by water in a heat exchanger before going to the second stage. Second, the vapor may be cooled by the liquid refrigerant, injected into the vapors from the first stage, to the temperature corresponding to the vapor pressure of the refrigerant equal to the pressure at the beginning of the second stage. Third, a combination of both may be used. A diagrammatic sketch of this third scheme is shown in Fig. 6.

Vapor is pumped from the expansion coils *C* by low-pressure compressor *LP* into the heat exchanger *H* where it is cooled by water. From *H* it passes into the accumulator *A* where it meets liquid refrigerant from the liquid receiver *L* which cools the vapor further. The high-pressure compressor then compresses the vapor to the saturation pressure of the condenser *K* in which it liquefies and is collected by the liquid receiver *L*. The liquid refrigerant then returns to the expansion coils through the valve E_1 and to the accumulator through the valve E_2.

When ammonia is used for low-temperature cooling, a booster compressor is generally used to raise the low-pressure vapor to some intermediate pressure. The vapors are cooled before going to the high-pressure compressor, partly to save power, partly to take out the superheat which would continue to build up in the second compressor. A further advantage is the saving in cylinder volume.

The power saving and the correct intermediate pressures are shown in Figs. 7 and 8.

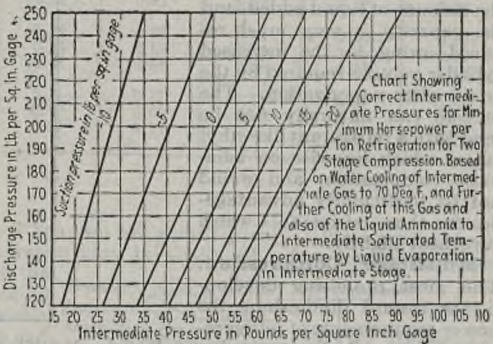


FIG. 8.—Correct intermediate pressures in multiple-stage compression.

Split-stage Compression or Binary Cycles (Two or More Refrigerants). When very low temperatures are required, it is sometimes advantageous to employ two or more refrigerants in order to maintain pressures within reasonable limits. Thus a refrigerant of relatively high boiling point will have such a low vapor pressure at low temperatures that excessive cylinder volume will be needed, whereas a refrigerant of low boiling point will have excessive pressures at the available condenser water temperatures. It may also have a critical temperature below that of the available cooling water and will therefore fail to liquefy. In such cases, a low boiling refrigerant may be used to produce cold and the low temperature, but the condenser is cooled by a high boiling refrigerant instead of by water.

A system using carbon dioxide for the low-temperature refrigerant and ammonia for cooling the condenser was described by Kitzmiller [*Power*, **75**, 92 (1932)]. Since ammonia and carbon dioxide react vigorously, great care must be taken in sealing joints. Recently developed refrigerants of the fluorinated and chlorinated hydrocarbons (Freon) permit combinations of refrigerants that are non-reactive, and may often find applications in low-temperature problems.

Compressors. Two general types of compressors are used for refrigeration: (1) compressors with positive displacement and (2) centrifugal compressors. Of those with positive displacement, the most common type is the reciprocating compressor which may be either **horizontal** or **vertical**.

The vertical type is largely displacing the horizontal type, the latter being found principally in very large refrigerating systems. In the United States two-cylinder compressors are common, but three or four cylinders are also used. Development at the present time tends largely toward the design of very high speed machines, since high speed gives greater capacity for a given cylinder volume. Electric drive, which is used in the greatest number of installations, is also more readily adapted to high speeds, 1200 to 1800 r.p.m. Oil- and steam-engine drives function more satisfactorily with low-speed machines (250 r.p.m.). High-speed machines require more careful design with respect to lubrication, balancing, and foundations; demand higher priced materials of construction; and have greater power losses due to friction and inertia in the piston. Water jackets are sometimes employed, chiefly to prevent the metal parts from overheating. They thus increase the volumetric efficiency but provide hardly enough cooling to make the compression isothermal. The poppet valve operated with springs is used on low-speed compressors, but high-speed machines often use light-weight plate or ribbon valves. Lubrication may be of the **splash** type if the compressor is enclosed, but for the open-frame construction or higher speed machines, **forced-feed** lubrication is desirable. An oil separator is generally provided near the condenser to return oil carried over by the refrigerant vapors.

Carbon dioxide compressors do not differ essentially from ammonia compressors except that heavier construction is required and stuffing boxes are more tightly packed. The cylinder diameter is smaller and the stroke longer.

Freon compressors handle a vapor of very high density, and the kinetic energy losses are therefore relatively high. To overcome these losses, ample gas passages must be provided and "streamlined." Suction valves should be 80 per cent larger than with ammonia and discharge valves about 100 per cent larger. However, water jackets are not needed with Freon because the gas superheats very little on compression. An extensive discussion of Freon compressor design is given by Williams [*Refrig. Eng.*, **31**, 36 (1936)].

Rotary compressors with positive displacement may be of the **blade** type operating on an eccentric, the blade being kept in contact with the casing by centrifugal force; or of the **pendulum** type. Rotary compressors are well adapted to high rotative speeds and direct connection to a high-speed electric motor; valves are not needed. They have been built only in small units, principally for household use, which show lower efficiencies than reciprocating compressors, but small units are in general less efficient than large ones.

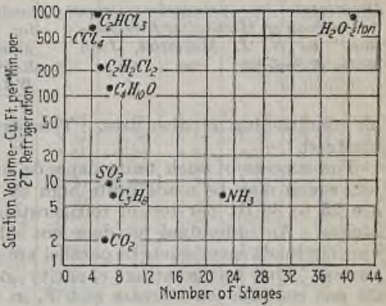
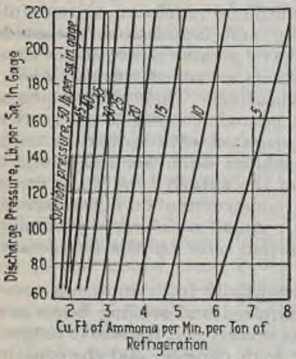


FIG. 9.—Relative size and number of stages required for various refrigerants (5° to 86°F.). [*Carrier and Waterfill, Refrigerating Eng.*, **10**, 423 (1924).]

In the **centrifugal compressor** the low-pressure vapor is drawn into impellers which impart a high velocity to the vapors. The high-speed vapors emerge through discharge vanes and their kinetic energy is then converted into pressure energy. The speed is usually 3000 to 6000 r.p.m. Several stages are necessary to raise the vapor to the discharge velocity (see Fig 9). Since the kinetic energy is a function of mass, refrigerants of high gas density are preferred. Slippage losses are less with low-pressure refrigerants than with high-pressure refrigerants, and since only high-molecular-weight refrigerants can give high densities at low pressures, a special class of such refrigerants has been developed for these machines.

Steam Jets. Low-pressure water vapor may be compressed by high-pressure steam in a steam jet, the operation involving conversion of kinetic energy into pressure-volume energy. In this way a vacuum can be created over water with resultant evaporation and cooling. This method is frequently very useful where moderate cooling is needed such as in chilling water for air conditioning or other uses. Temperatures from 35°F. up can generally be attained.



No liquid cooling, volumetric efficiency allowed for

FIG. 10.—Volume of ammonia compressed per ton of refrigeration. (*Reprinted by permission from "Handbook of Mechanical Refrigeration," by H. J. Macintire, John Wiley & Sons.*)

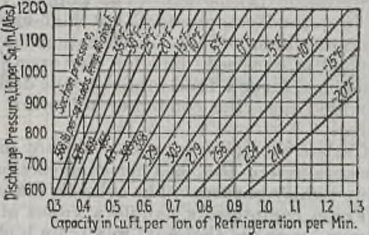


FIG. 11.—Volume of carbon dioxide pumped per ton of refrigeration. No liquid cooling; dry compression; allowance made for volumetric efficiency. (*Reprinted by permission from "Handbook of Mechanical Refrigeration," by H. J. Macintire, John Wiley & Sons.*)

The success of such units depends upon the mechanical efficiency of the jet, which may be made quite high by proper design. Steam requirements are 25 to 30 lb. per ton of refrigeration with steam pressures of 100 lb. or higher. An individual jet does not allow for much change in load, so for varying loads several jets in parallel are necessary to turn off or on as the load varies. The refrigerating capacity of the jet increases approximately 20 per cent for an increase of 5°F. in the temperature of the chilled water. However, if the temperature of the condensing water is increased 5°F. beyond the maximum anticipated in design, the jet may fail to function altogether. To avoid this contingency, a reserve of 25 to 50 lb. in steam pressure should be provided. Condensing water requirements are about three times greater than in conventional compression systems [Bancel, *Trans. Am. Inst. Chem. Engrs.*, **30**, 136 (1933–1934)].

If adequate height is available, it is desirable to provide a barometric condenser to lower the back pressure on the jet.

A full discussion of cycles, standards, thermodynamic diagrams, and test codes for steam jets may be found in "Steam Jet Ejector and Vacuum Cooling Systems," Heat Exchange Institute, New York, 1938.

Lubricants. The oil used for a refrigeration system should have a sufficiently low pour point so that it does not congeal on the coldest parts of the system. Most of the oil companies prepare such a grade of mineral oil, the usual specification being a pour point of −20°F. to −30°F. Such oils may be used for ammonia, carbon dioxide, and sulfur dioxide. Propane dissolves to a very appreciable extent, and the oil when removed from the oil separator foams extensively. Liquid sulfur dioxide has some lubricating effect and has a higher specific gravity than oil. The oil used for SO_2 machines must be highly anhydrous.

Light mineral oils can be used for methyl chloride and ethyl chloride machines, but since they are soluble in oil, glycerin is often used as a lubricant with these refrigerants. When propane or butane is used in household machines, the best results are obtained with glycerin or glycol mixed with deflocculated graphite. Household machines in general require extreme precautions against infiltration of moisture.

Compressor Capacity. The weight of dry-saturated gas that must be compressed by the compressor per minute to produce 1 ton of refrigeration per 24 hr. is given by the formula: $W = 200/(H_b - h_e)$, where W is the weight of gas compressed per minute, H_b is the total heat in 1 lb. of vapor at the evaporator pressure, h_e is the heat of the liquid at the receiver pressure, and 200 is the number of B.t.u. removed per minute to equal 1 ton of refrigeration per 24 hr. (Table 4 and Figs. 10 and 11).

Table 4. Cubic Feet of Ammonia Gas to Be Pumped per Minute to Produce 1 Ton of Refrigeration per Day of 24 Hr.

			Condenser								
	P	P	103	115	127	139	153	168	185	200	218
		T	65°	70°	75°	80°	85°	90°	95°	100°	105°
Evaporator	4	−20°	5.84	5.90	5.96	6.03	6.09	6.16	6.23	6.30	6.43
	6	−15°	5.35	5.40	5.46	5.52	5.58	5.64	5.70	5.77	5.83
	9	−10°	4.66	4.73	4.76	4.81	4.86	4.91	4.97	5.05	5.08
	13	− 5°	4.09	4.12	4.17	4.21	4.25	4.30	4.35	4.40	4.44
	16	0°	3.59	3.63	3.66	3.70	3.74	3.78	3.83	3.87	3.91
	20	5°	3.20	3.24	3.27	3.30	3.34	3.38	3.41	3.45	3.49
	24	10°	2.87	2.90	2.93	2.96	2.99	3.02	3.06	3.09	3.12
	28	15°	2.59	2.61	2.65	2.68	2.71	2.73	2.76	2.80	2.82
	33	20°	2.31	2.34	2.36	2.38	2.41	2.44	2.46	2.49	2.51
	39	25°	2.06	2.08	2.10	2.12	2.15	2.17	2.20	2.22	2.24
	45	30°	1.85	1.87	1.89	1.91	1.93	1.95	1.97	2.00	2.01
	51	35°	1.70	1.72	1.74	1.76	1.77	1.79	1.81	1.83	1.85

The values in this table are calculated theoretical values. In practice, allowance must be made for losses. Temperatures are in degrees Fahrenheit.

The formula for the theoretical capacity, in tons of refrigeration per 24 hr., of a double-acting compressor, taking into consideration the specified pressures in the condenser and the refrigerator, is

$$T = \frac{d^2 \times 0.7854 \times L \times 2 \times N \times h}{1728 \times 200}$$

where T is the tons refrigeration in 24 hr.; d is the diameter of compressor cylinder in inches; L is the stroke in inches; N is the number of revolutions per minute; h is the refrigeration effect of 1 cu. ft. of ammonia vapor at refrigerator pressure, completely evaporated from liquid at condenser pressure.

For single-acting machines divide the tons of refrigeration found from this formula by 2. The capacity of vertical single-acting enclosed machines is shown in Table 5.

Table 5. Capacities of V. S. A. Enclosed-type Machines

Cylinder			Piston speed per minute		Displacement per minute		Capacity in tons per 24 hr., 185-lb. per sq. in. condensing pressure								
										16 lb., 0°F.					
Number	Bore	Stroke	R.p.m.	Ft.	Cu. in.	Cu. ft.	6 lb. −15°F., refg.	9 lb. −10°F., refg.	12 lb. −5°F., refg.	Ice	Refg.	19 lb. 5°F., refg.	24 lb. 10°F., refg.	28 lb. 15°F., refg.	33 lb. 20°F., refg.
1	3	3	300	150	6,362	3.68	0.47	0.51	0.65	0.47	0.75	0.85	1.96	1.07	1.24
1	4	4	275	183	13,828	7.99	0.98	1.12	1.35	1.0	1.6	1.78	2.0	2.2	2.6
2	4	4	275	183	27,646	15.98	1.96	2.24	2.7	2.0	3.2	3.5	4.0	4.4	5.2
2	5	5	240	200	47,124	27.3	3.34	3.84	4.6	3.3	5.3	5.97	6.8	7.7	8.85
2	6	6	220	220	74,644	43.2	5.5	6.6	7.6	5.5	8.8	10.0	11.4	12.6	14.5
2	7	7	210	245	113,146	65.6	8.5	10.2	11.9	8.5	13.6	15.5	17.4	19.6	22.4
2	8	8	200	266	160,848	93.08	12.2	14.6	16.9	12.1	19.5	22.1	24.9	28.0	32.1
2	9	9	190	285	217,570	125.8	16.6	19.2	22.9	16.6	26.4	29.8	33.8	37.9	43.5
2	10	10	180	300	282,744	163.62	22.1	26.1	30.4	21.9	35.0	38.8	44.9	50.4	57.8
2	12	12	170	340	461,448	267.04	35.1	41.9	48.6	35.0	56.0	63.4	71.6	80.3	92.4

Multiply above capacities by factors in Table 6 to obtain capacities at different condenser pressures.

CONTINUOUS ABSORPTION SYSTEMS

In the conventional continuous absorption system (Fig. 12) the refrigerant, usually ammonia, is drawn from the expansion coils F at a given pressure p_1 and dissolved in the absorber G where the combined pressure of solvent and refrigerant must be less than p_1. The refrigerant is recovered by applying heat to the generator A; separated from entrained water by the analyzer or rectifier B; liquefied in the condenser C; collected in a liquid receiver D (not shown); and returned to the expansion coils F through the expansion valve E. Since the absorber is at lower pressure than the generator, it is necessary to insert between them a pump H to transfer the ammonia solution usually known as strong aqua. The generator does not remove all the ammonia from the water, and this weak aqua solution is returned to the absorber through the heat exchanger I. Considerable heat is evolved in the absorption of ammonia vapor, and cooling water must be supplied to keep down the temperature and

Table 6. Ammonia Suction Pressure

Condenser pressure, lb. per sq. in. gage	6 lb.	9 lb.	12 lb.	16 lb.	19 lb.	24 lb.	28 lb.	33 lb.
200	0.975	0.969	0.975	0.977	0.978	0.977	0.976	0.977
168	1.029	1.028	1.028	1.028	1.02	1.027	1.026	1.022
153	1.046	1.049	1.05	1.05	1.043	1.05	1.042	1.042

pressure. In the absorption system two complete cycles are in operation: (1) the refrigerant cycle, and (2) the absorbent cycle. Just as the absorbent contains some ammonia, so the refrigerant contains a small quantity of water.

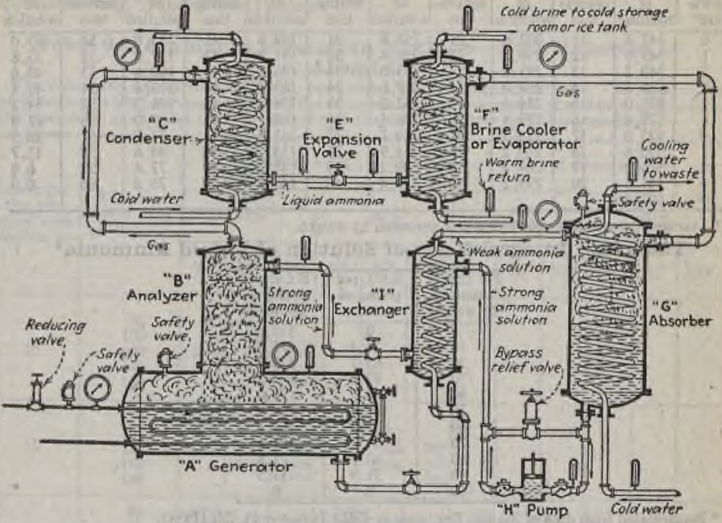


FIG. 12.—Continuous absorption system.

In the absorption cycle the heat quantities involved per pound of ammonia passing through the expansion valves are

Q_1 = heat input to the generator.
Q_2 = heat absorbed by the expansion coils.
Q_3 = heat equivalent of the work of the pump.
Q_4 = heat rejected in the condenser.
Q_5 = heat rejected in the absorber.
Q_6 = heat lost by radiation, etc.

The heat balance is then given by the equation

$$Q_1 + Q_2 + Q_3 = Q_4 + Q_5 + Q_6$$

Of these quantities only Q_2, Q_3, and Q_4 can be computed with any exactness. Q_2 is very nearly the heat of evaporation of 1 lb. of ammonia at the temperature of the expansion coils. Similarly Q_4 is very nearly the heat of condensation of 1 lb. of ammonia at the temperature of the condenser. Q_3 may be found by the equation

$$Q_3 = Gv(p_2 - p_1)$$

where G = strong aqua pumped per pound of refrigerant passed, lb.
v = volume of 1 lb. of strong aqua, cu. ft.
p_2 = generator pressure, lb. per sq. in.
p_1 = absorber pressure, lb. per sq. in.

The heat rejected in the absorber Q_5 is made up of three parts: (1) the heat of condensation; (2) the heat of solution in going from weak aqua to strong

Table 7. Differential Heats of Solution of Liquid Ammonia

B.t.u. given up per pound of ammonia dissolved

Concentration*	Heat of solution	Concentration	Heat of solution	Concentration	Heat of solution	Concentration	Heat of solution	Concentration	Heat of solution	Concentration	Heat of solution
0	347.4	11	302.8	21	253.8	31	197.6	41	135.0	51	63.0
1	343.8	12	298.2	22	248.4	32	191.9	42	127.8	52	55.8
2	340.2	13	293.6	23	243.0	33	186.1	43	120.6	53	48.6
3	336.6	14	289.0	24	237.6	34	180.4	44	113.4	54	41.4
4	333.0	15	284.4	25	232.2	35	174.6	45	106.2	55	34.2
5	329.4	16	279.4	26	226.4	36	168.1	46	99.0	56	27.4
6	325.0	17	274.3	27	220.7	37	161.6	47	91.8	57	20.5
7	320.6	18	269.2	28	214.9	38	155.2	48	84.6	58	13.7
8	316.2	19	264.2	29	209.2	39	148.7	49	77.4	59	6.8
9	311.8	20	259.2	30	203.4	40	142.2	50	70.2	60	0.0
10	307.4										

* Average concentration; percentages of ammonia by weight.

Table 7a. Integral Heats of Solution of Liquid Ammonia*

Per cent of ammonia by weight	B.t.u. per pound of mixture	B.t.u. per pound of ammonia
0	0	358.0
10	34.4	343.8
20	65.7	328.5
30	92.5	308.2
40	108.2	270.0
50	109.4	218.8
60	101.9	169.7
70	84.8	121.1
80	60.7	75.8
90	31.9	35.5
100	0	0

* Recalculated from data of Zinner, *Zeit. gesamte Kälte-Industrie* **41**, 21, (1934).

Note: Tables 7 and 7a are based on data from different sources and are not perfectly consistent; the differences are negligible for most engineering calculations. The data of Table 7a are probably more accurate.

Table 8. Test Results at Quincy Market Cold Storage and Warehouse Co.

Machine	Capacity of machine, tons	Capacity during test, tons	Suction temperature, °F.	Discharge pressure, lb. gage	Steam conditions			Compressor i.h.p. per ton of refrigeration	Steam per ton, lb.
					Pressure, lb. per sq. in.	Superheat, °F.	Vacuum, in. Hg		
1	1000	750	+10	115	150	100	28	0.9	228
2	400	400	−10	115	150	100	28	1.25	390
2	400	400	+10	115	150	100	28	0.9	282
3	500	288	−27	139	140	125	28	1.34	426
4	500	370	−20	130	...	...	..	1.64	
5	150	150	−10	115	125	125	28	1.28	430
5	150	150	+10	115	125	125	28	0.92	376
6	100	100	−10	115	125	125	28	1.32	459
6	100	100	+10	115	125	125	28	0.94	389
7	225	225	−10	115	...	...	..	....	342.2
7	225	225	+10	115	...	...	..	....	316.8

1 and 2, cross-compound Corliss engines; 3, two-stage feather-valve compressor driven by Uniflow engine; 4, two-stage electrically driven compressor; 5 and 6, tandem compound engines; 7, absorption machine.

The discharge pressures in these tests are exceptionally favorable and results in low power consumption. The absorption machine was more economical than the 400-ton compression machine with the low-temperature suction gas, and less economical than the same machine with high-temperature suction gas.

Station operation results show that absorption machines require 25 to 35 lb. of steam per ton of refrigeration with 0° brine and 150 lb. per sq. in. condenser pressure. Makers' guarantees are from 55 to 60 lb. of live steam per hour per ton of ice.

aqua; and (3) the heat which must be removed from the weak aqua solution to reduce it to the temperature of the absorber. The theoretical heat input to the generator Q_1 has three corresponding parts. The values may be approximately found from Table 7.

Table 9. Ammonia Absorption Machine—Heat Removed in Absorber

The tabulated quantities are B.t.u. per pound of ammonia absorbed. The pressure in the absorber is that corresponding to the temperature and pressure of the strong solution. The ammonia gas is assumed to be dry and saturated as it enters the absorber.

Concentration per cent of weak solution, x_2	Temperature of weak solution, °F.	Concentration (%), x_1, and temperature (°F.) of strong solution											
		25%			30%			35%			40%		
		60°	80°	100°	60°	80°	100°	60°	80°	100°	60°	80°	100°
10	80	853											
	100	954	838										
	120	1056	939	821									
	140	....	1041	924									
	160	....	....	1026									
15	80	892	....	....	828								
	100	1046	877	....	923	812							
	120	1200	1032	862	1019	908	795						
	140	....	1186	1016	....	1004	891						
	160	....	....	1173	....	....	989						
20	80	1035	....	....	863	....	....	799					
	100	1344	1021	....	1007	848	....	889	783				
	120	1656	1332	1006	1153	993	831	979	873	766			
	140	....	1644	1318	....	1139	976	....	963	856			
	160	....	....	1636	....	....	1125	....	....	948			
25	80	....	....	....	995	....	....	830	....	....	769		
	100	....	....	....	1286	980	....	966	814	....	852	752	
	120	....	....	....	1579	1274	965	1102	950	798	936	836	735
	140	....	....	....	....	1568	1259	....	1087	934	....	920	819
	160	....	....	....	....	....	1559	....	....	1074	....	....	905
30	80	....	....	....	....	....	....	952	....	....	796		
	100	....	....	....	....	....	....	1224	937	....	922	779	
	120	....	....	....	....	....	....	1499	1211	921	1047	906	762
	140	....	....	....	....	....	....	....	1487	1197	....	1033	890
	160	....	....	....	....	....	....	....	....	1480	....	....	1020
35	80	....	....	....	....	....	....	....	....	....	907		
	100	....	....	....	....	....	....	....	....	....	1161	891	
	120	....	....	....	....	....	....	....	....	....	1416	1147	876
	140	....	....	....	....	....	....	....	....	....	....	1404	1132
	160	....	....	....	....	....	....	....	....	....	....	....	1397

The value of Q_6 is obviously indeterminate.

In view of the lack of thermal data on the processes in the absorption machine it is practically impossible to calculate efficiencies. Tests conducted at the Quincy Market Cold Storage and Warehouse Company (Table 8) show how the steam consumption on absorption machines compares with the steam consumption if used to operate a steam engine driving a compressor unit. Table 9 gives a similar comparison of theoretical performances.

Table 10. Total Vapor Pressures of Aqua Ammonia

Lb. per sq. in. abs.

Temp., °F.	Molal concentration of ammonia in the solutions in percentages										
	0	5	10	15	20	25	30	35	40	45	50
32	0.09	0.34	0.60	0.97	1.58	2.60	4.20	6.54	9.93	14.18	19.40
40	0.12	0.45	0.77	1.24	2.01	3.25	5.21	8.06	12.05	17.20	23.39
50	0.18	0.64	1.05	1.65	2.67	4.29	6.75	10.35	15.34	21.65	29.26
60	0.26	0.86	1.42	2.21	3.51	5.55	8.65	13.22	19.30	27.05	36.26
70	0.36	1.17	1.84	2.90	4.56	7.13	11.01	16.56	24.05	33.39	44.42
80	0.51	1.52	2.43	3.76	5.85	9.06	13.86	20.61	29.69	40.96	54.08
90	0.70	2.02	3.15	4.83	7.43	11.40	17.23	25.48	36.34	49.82	65.32
100	0.95	2.62	4.05	6.13	9.34	14.22	21.32	31.16	44.12	59.99	78.30
110	1.27	3.34	5.14	7.72	11.64	17.58	26.07	37.81	53.16	71.87	93.19
120	1.69	4.27	6.46	9.63	14.42	21.54	31.69	45.62	63.59	85.33	110.20
130	2.22	5.38	8.07	11.91	17.67	26.20	38.25	54.55	75.55	100.86	129.50
140	2.89	6.70	9.98	14.63	21.49	31.54	45.73	64.78	89.19	118.24	151.30
150	3.72	8.29	12.23	17.81	26.00	37.81	54.43	76.61	104.65	138.10	175.40
160	4.74	10.16	14.92	21.54	31.16	45.02	64.25	89.88	122.10	160.20	202.70
170	5.99	12.41	18.01	25.87	37.11	53.27	75.55	104.84	141.75	185.10	233.20
180	7.51	15.00	21.65	30.86	44.02	62.68	88.17	121.68	163.70	212.60	267.00
190	9.34	18.06	25.87	36.60	51.81	73.32	102.56	140.75	188.10	243.30	304.30
200	11.53	21.60	30.72	43.14	60.62	85.33	118.68	161.81	215.20	277.00	345.50
210	14.12	25.61	36.26	50.58	70.72	98.80	136.42	185.10	245.10	314.50	390.70
220	17.19	30.27	42.47	59.00	81.91	113.81	156.41	211.24	278.20	355.10	439.60
230	20.78	35.59	49.60	68.46	94.43	130.64	178.28	239.70	314.50	400.20	493.40
240	24.97	41.52	57.65	78.91	108.60	149.20	202.74	270.92	354.10	448.90	552.30
250	29.83	48.32	66.67	90.74	124.08	169.48	229.62	305.60	397.60	502.40	

Table 11. Partial Pressures of Water Vapor above Aqua Ammonia

Lb. per sq. in. abs.

Temp., °F.	Molal concentration of ammonia in the solutions in percentages										
	0	5	10	15	20	25	30	35	40	45	50
32	0.09	0.084	0.079	0.074	0.070	0.065	0.060	0.056	0.051	0.047	0.042
40	0.12	0.115	0.108	0.101	0.095	0.089	0.083	0.076	0.070	0.064	0.058
50	0.18	0.17	0.16	0.15	0.14	0.13	0.12	0.11	0.10	0.094	0.085
60	0.26	0.24	0.23	0.21	0.20	0.19	0.17	0.16	0.15	0.13	0.12
70	0.36	0.34	0.32	0.30	0.28	0.26	0.25	0.23	0.21	0.19	0.17
80	0.51	0.48	0.45	0.42	0.40	0.37	0.34	0.32	0.29	0.27	0.24
90	0.70	0.66	0.63	0.58	0.55	0.51	0.47	0.44	0.40	0.37	0.33
100	0.95	0.90	0.85	0.79	0.74	0.69	0.64	0.59	0.55	0.50	0.45
110	1.27	1.20	1.14	1.07	1.00	0.93	0.86	0.80	0.73	0.67	0.60
120	1.69	1.60	1.51	1.42	1.33	1.24	1.15	1.06	0.97	0.89	0.80
130	2.22	2.10	1.98	1.86	1.74	1.62	1.51	1.39	1.28	1.17	1.05
140	2.89	2.73	2.57	2.42	2.26	2.11	1.96	1.81	1.66	1.52	1.37
150	3.72	3.51	3.31	3.11	2.91	2.72	2.52	2.33	2.14	1.95	1.76
160	4.74	4.48	4.22	3.97	3.71	3.46	3.22	2.97	2.73	2.49	2.25
170	5.99	5.66	5.34	5.02	4.70	4.38	4.07	3.75	3.45	3.15	2.84
180	7.51	7.10	6.69	6.30	5.89	5.49	5.10	4.71	4.33	3.94	3.57
190	9.34	8.83	8.32	7.82	7.32	6.83	6.34	5.86	5.38	4.91	4.44
200	11.53	10.90	10.27	9.65	9.04	8.43	7.83	7.23	6.64	6.06	5.48
210	14.12	13.35	12.58	11.82	11.07	10.32	9.59	8.86	8.13	7.42	6.71
220	17.19	16.25	15.32	14.39	13.48	12.57	11.67	10.78	9.90	9.03	8.17
230	20.78	19.64	18.51	17.40	16.29	15.19	14.11	13.03	11.97	10.91	9.87
240	24.97	23.60	22.25	20.91	19.58	18.26	16.95	15.66	14.38	13.12	11.86
250	29.83	28.20	26.58	25.00	23.39	21.82	20.25	18.71	17.18	15.67	

Table 10 gives values of the total vapor pressures of aqua ammonia as a function of the temperature and of the concentration. Table 11 gives values of the partial pressure of water vapor for various concentrations of aqua ammonia as a function of the temperature.

Table 12. Performance of Absorption Machine with Different Condenser and Suction Pressures*

	Condenser pressure, lb. per sq. in. abs.								
	155			185			215		
	Suction pressure, lb. per sq. in. abs.								
	15	30	45	15	30	45	15	30	45
S.L., per cent	24.0	35.0	42.0	22.0	32.0	38.0	18.0	28.0	36.0
W.L., per cent	13.1	25.8	33.7	10.9	22.3	29.2	6.3	17.7	26.9
S.G., lb.	30.1	27.9	22.9	41.3	30.9	26.2	48.7	34.1	27.9
S.P., lb.	31.8	29.5	24.3	43.4	32.8	28.0	51.1	36.4	30.1
Relative capacities { Absorption	0.97	1.03	1.09	0.94	1.00	1.05	0.91	0.97	1.03
Relative capacities { Compression	0.46	1.05	1.62	0.43	1.00	1.56	0.41	0.95	1.49

S.L. is strong liquor.
W.L. is weak liquor.
S.G. is the steam consumption of the generator.
S.P. is the steam consumption of the pump.
* Voorhees, "Refrigerating Machines."

Absorption machines are gradually becoming obsolete in industrial work. They were formerly much favored for low-temperature work. However, when large quantities of waste heat are available, it will always be well to examine their use. This obsolescence may be traced to several causes: (1) the bulkiness of the apparatus, (2) the increasing availability of cheap electric power, (3) the development of multistage compression for very low temperature refrigeration, (4) the lack of flexibility in its operation, (5) the necessity of constant and skillful attention for maximum economy, and (6) the corrosion of the system by the aqua ammonia solutions, which, however, may be largely nullified by sodium bichromate to the extent of 0.2 per cent of the total aqua charge (McKelvy and Isaacs, *J. Am. Soc. Refrig. Eng.*, March, 1918).

Absorption Systems with Other Refrigerants. Interest in the use of absorption systems for air conditioning has been revived recently (Zellhoefer, *Refrig. Eng.*, May, 1937). Because of its odor, ammonia cannot be used in this application, and considerable research has been devoted toward finding other combinations of refrigerants and solvents which will be as efficient as the ammonia water system. The prime requirement is that the refrigerant shall be highly soluble to minimize the heat waste in warming the solvent.

Platen-Munters Continuous Absorption System. In the Platen-Munters system (**Electrolux Household Refrigerator**), circulation of refrigerant and absorbent is effected by the operation of hydrostatic forces developed within the system itself, and consequently there are no mechanical moving parts. In order to produce these hydrostatic forces, it is necessary to add a third component, which must differ in density (molecular weight) from the refrigerant and be capable of separation from the refrigerant. The usual combination is ammonia, water, and hydrogen.

The essential features are shown in Fig. 13. Strong ammonia solution is heated in the lower portion of the generator *G*, the weak liquor flowing back to the absorber *A*, while the ammonia-water vapors are separated in the rectifier *R* and the ammonia vapors liquefied in the condenser *C*. The liquid ammonia passes through a heat exchanger *D* into the evaporator *E*. In the evaporator it flows into an atmosphere of hydrogen and is vaporized, though the process is more analogous to humidification. The ammonia-hydrogen mixture then moves back to the absorber *A* through the heat exchanger, and the ammonia is dissolved leaving substantially pure hydrogen. Circulation is produced because the column of hydrogen in the absorber is opposed by a heavier column of hydrogen-ammonia vapors in the evaporator, although the total gas pressure throughout the unit is equal. A difference in hydrostatic pressure, obtained partly by heat, partly by change in liquid density, also prevails between the generator and absorber.

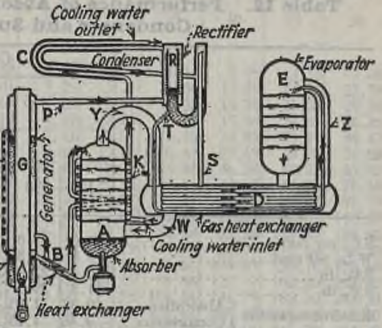


FIG. 13.—Electrolux gas-heated refrigerating unit.

The Platen-Munters principle has as yet found application only in the small household refrigerator. To function properly it is necessary that leakage be avoided, and this entails excessive construction costs on large units. The efficiencies and capacities of a small unit are shown in Fig. 14, efficiency being based on the ratio of refrigeration produced to heat supplied.

Intermittent Absorption Machine. In the intermittent absorption machine the absorber and generator, and the condenser and the evaporator, are combined. During the cooling period the refrigerant is taken up by the absorbent which may be water, silica gel, activated charcoal, or chlorides of the alkaline earth metals. When the absorbent has become saturated it is regenerated by heating and the vapors condensed in the evaporator which is cooled by running water. These machines have not proved very successful for two reasons: (1) the general desirability of maintaining continuous refrigeration and (2) the great explosion hazard in overheating during regeneration.

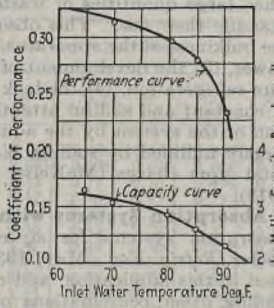


FIG. 14.—Effect of varying cooling-water temperature. Gas rate 2.5 cu. ft. per hr. Room temperature 70°F. Quantity of water 7 gal. per hr.

For air-conditioning work they have recently acquired some promise, silica gel being used for absorbing moisture from the air and later regenerated by gas firing. A good deal of research has been expended on their adaptation to truck and railway refrigeration [Keyes, *Ind. Eng. Chem.*, **21**, 477 (1929)].

REFRIGERANTS USED IN VAPOR-COMPRESSION AND ABSORPTION SYSTEMS

The choice of a refrigerant is always determined by the specific conditions of refrigeration to be met. A refrigerant cannot be used in a compression machine if the temperature of the available cooling water lies above the critical temperature of the refrigerant. The temperature of cooling waters in America is usually too high to use carbon dioxide with advantage (see Fig. 15). For absorption machines, it is necessary that the absorbent be capable of taking up large quantities of the refrigerant. In centrifugal compression machines it is necessary that the vapor pressures be as low as possible so as to reduce the number of stages and that the molecular weight be as high as possible so as to reduce the peripheral speed of the rotor.

FIG. 15.—Surface-water temperatures (during summer months) in the United States. (*Reprinted by permission from "Handbook of Mechanical Refrigeration," by H. J. Macintire, John Wiley & Sons.*)

In vapor-compression machines of the reciprocating type, refrigerants of high vapor pressure are desired since these give low piston displacement and smaller friction losses. A comparison of the vapor pressures of various materials is shown in Fig. 16, where the logarithm of the vapor pressure in atmospheres is plotted against the reciprocal of the absolute centigrade temperature. The slopes of all the lines are very nearly parallel (Trouton's law), water being the notable exception, and the compression ratios for any given temperature interval are very nearly equal.

Other properties of importance are: chemical stability, inertness to metals (corrosion), ease of detection in case of leaks, and behavior with lubricants. Cheapness of the refrigerant is a very important item in larger units since it is not practicable to prevent leakage altogether.

Ammonia is the most important refrigerant used in industrial work, but the irritating and toxic character of the vapor excludes it from certain applications, such as marine refrigeration or air conditioning, where carbon dioxide and dichlorodifluoromethane are preferred. Although the thermodynamic efficiency of ammonia is not very different from other refrigerants, its low gas density is a distinct practical advantage in reducing power losses. In certain chemical processes, notably those in which free chlorine may be present, ammonia is also undesirable. Propane having nearly the same pressures as ammonia may be substituted in machines designed for ammonia. For household refrigeration, air condensers are preferred because of the convenience in moving the units about, and, since ammonia does not work satisfactorily with an air condenser, it is not used in any of the small machines of the vapor-compression type, its use in this field being confined to the gas-fired absorption machines. Automatic regulation is also more difficult with ammonia because of its high latent heat per pound, which means that a small departure from the average flow upsets the temperature a great deal more than with a refrigerant of low latent heat.

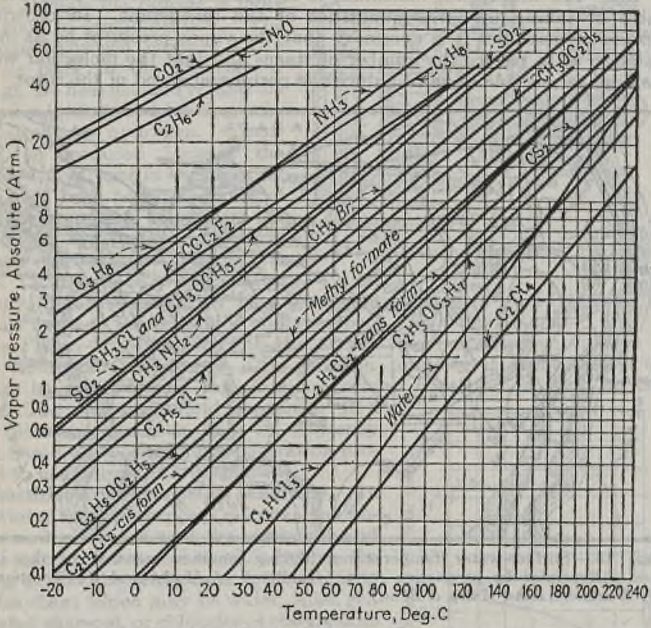


FIG. 16.—Vapor pressures of refrigerants.

The most important refrigerants used in household machines are sulfur dioxide, methyl chloride, and difluorodichloromethane. The last-named refrigerant has been developed in response to the demand for a non-toxic, non-flammable material for household machines. [Midgeley and Henne, *Ind. Eng. Chem.*, **22**, 542 (1930).] Dimethyl ether, first proposed by Linde at the beginning of the modern period of refrigeration developments, is used in some

German units. The low-pressure, high-molecular-weight refrigerants—methylene chloride and the more complex fluoro- chloro- methanes—are used in centrifugal compressors for air-conditioning work.

Binary mixtures have been proposed as refrigerating fluids, but they have been unsuccessful because one of the components is always likely to leak out more rapidly than the other and when this happens it becomes impossible to interpret indicated pressures satisfactorily. They also require a greater work expenditure if any rectification occurs in the evaporator. If two refrigerants have nearly identical vapor pressures, such as methyl chloride or dimethyl ether, or carbon dioxide and nitrous oxide, they cannot be separated and would act as a one-component fluid. Two refrigerants may also form constant boiling mixtures which behave similarly, but the vapor pressures of such binaries have not been determined. There are no essential advantages in binary mixtures except that the freezing points might be lowered.

Small admixtures of warning or detecting agents are sometimes added. Thus peppermint may be added to carbon dioxide, acrolein to methyl chloride (both of which have characteristic odors), or ethyl nitrite for the detection of pinholes by discoloration of starch-potassium iodide paper. With ammonia, a sulfur candle, a piece of litmus paper, or a glass rod dipped in acid serves as means of detection. Refrigerants containing chlorine can usually be detected by lighting a taper which gives a bluish-green flame in the escaping vapors, but this procedure is to be used with care in the case of methyl chloride which may form an explosive mixture with air.

The heat-transfer characteristics of refrigerants, in both the liquid and gaseous phases, are also important, but very few comparative data have been obtained so far. According to R. C. Doremus (Eng. Sec., *Elec. Refrigeration News*, Sept. 23, 1931) the rate of heat transfer for sulfur dioxide is about 85 per cent as great as for methyl chloride under the same conditions of evaporation. Thermal conductivities have been measured by Schreiber [*Z. Ver. deut. Ing.*, **75**, 969 (1931)] with the following results:

SO_2	0.194 B.t.u./sq. ft. (hr.) (°F. per ft.)
CO_2	0.121 B.t.u./sq. ft. (hr.) (°F. per ft.)
NH_3	0.36 B.t.u./sq. ft. (hr.) (°F. per ft.) (approx.)
H_2O	0.36 B.t.u./sq. ft. (hr.) (°F. per ft.)

Thermodynamic Properties of Refrigerants. The thermodynamic properties of various refrigerants are given in the following tables:

Freezing points of refrigerants	12a
Ammonia	13, 14, 15, 16
Carbon dioxide	17
Sulfur dioxide	18, 19
Methyl chloride	20, 21
Ethyl chloride	22
Dichlorodifluoromethane	23, 24
Dichloromonofluoromethane	25, 26
Trichloromonofluoromethane	27, 28
Propane	29
Butane	30
Isobutane	31
Ethane	32
Methyl bromide	33
Nitrous oxide	34
Ethyl ether	35

Table 12a. Freezing Points of Refrigerants

Refrigerant	°F.	°C.
Ammonia,* NH_3	−107.9	− 77.7
Carbon dioxide,* CO_2	−109.3	− 78.5
Methyl chloride,* CH_3Cl	−132.7	− 91.5
Ethyl chloride,* C_2H_5Cl	−217.7	−138.7
Sulfur dioxide,* SO_2	−103.2	− 75.2
"Freon-12,"† CCl_2F_2	−247.0	−155.0
"Freon-11,"† CCl_3F	−126.4	− 88.0
"Freon-21,"† $CHCl_2F$	−196.6	−127.0
"Freon-114,"† $C_2Cl_2F_4$	−158.0	−105.5

* Circ. 2, *Am. Soc. Refrig. Eng.* (1926).
† Courtesy Kinetic Chemicals, Inc.

Table 13. Properties of Liquid Ammonia*

Temp., °F.	Saturation						Latent heat of pressure variation, B.t.u. per lb./lb. per sq. in.	Variation of h with p (t constant), B.t.u. per lb./lb. per sq. in.	Compressibility, lb. per sq. in. × 10^6
	Pressure (abs.), lb. per sq. in.	Volume, cu. ft. per lb.	Density, lb. per cu. ft.	Specific heat, B.t.u. per lb. per °F.	Heat content, B.t.u. per lb.	Latent heat, B.t.u. per lb.			
t	p	v	$\frac{1}{v}$	c	h	L	l	$\left(\frac{\partial h}{\partial p}\right)_t$	$-\frac{1}{v}\left(\frac{\partial}{\partial p}\right)_t$
Triple point	......	0.01961†	51.00†						
	0.88	.02182	45.83						
−100	1.24	.02197	45.52	(1.040)	(−63.0)	(633)			
− 95	1.52	.02207	45.32	(1.042)	(−57.8)	(631)			
− 90	1.86	.02216	45.12	(1.043)	(−52.6)	(628)			
− 85	2.27	.02226	44.92	(1.045)	(−47.4)	(625)			
− 80	2.74	.02236	44.72	(1.046)	(−42.2)	(622)			
− 75	3.29	.02246	44.52	(1.048)	(−36.9)	(619)			
− 70	3.94	.02256	44.32	(1.050)	(−31.7)	(616)			
− 65	4.69	.02267	44.11	(1.052)	(−26.4)	(613)			
− 60	5.55	.02278	43.91	1.054	−21.18	610.8	−0.0016	0.0026	4.4
− 55	6.54	.02288	43.70	1.056	−15.90	607.5	− .0016	.0026	4.5
− 50	7.67	.02299	43.49	1.058	−10.61	604.3	− .0017	.0026	4.6
− 45	8.95	.02310	43.28	1.060	− 5.31	600.9	− .0017	.0026	4.7
− 40	10.41	.02322	43.08	1.062	0.00	597.6	− .0018	.0025	4.8
− 35	12.05	.02333	42.86	1.064	+ 5.32	594.2	− .0018	.0025	5.0
− 30	13.90	.02345	42.65	1.066	10.66	590.7	− .0019	.0025	5.1
− 25	15.98	.02357	42.44	1.068	16.00	587.2	− .0019	.0024	5.2
− 20	18.30	.02369	42.22	1.070	21.36	583.6	− .0020	.0024	5.4
− 15	20.88	.02381	42.00	1.073	26.73	580.0	− .0020	.0024	5.5
− 10	23.74	.02393	41.78	1.075	32.11	576.4	− .0021	.0023	5.7
− 5	26.92	.02406	41.56	1.078	37.51	572.6	− .0022	.0023	5.8
0	30.42	.02419	41.34	1.080	42.92	568.9	− .0022	.0022	6.0
5	34.27	.02432	41.11	1.083	48.35	565.0	− .0023	.0022	6.2
10	38.51	.02446	40.89	1.085	53.79	561.1	− .0024	.0021	6.4
15	43.14	.02460	40.66	1.088	59.24	557.1	− .0025	.0021	6.6
20	48.21	.02474	40.43	1.091	64.71	553.1	− .0025	.0020	6.8
25	53.73	.02488	40.20	1.094	70.20	548.9	− .0026	.0020	7.0
30	59.74	.02503	39.96	1.097	75.71	544.8	− .0027	.0019	7.3
35	66.26	.02518	39.72	1.100	81.23	540.5	− .0028	.0019	7.5
40	73.32	.02533	39.49	1.104	86.77	536.2	− .0029	.0018	7.8
45	80.96	.02548	39.24	1.108	92.34	531.8	− .0030	.0017	8.1

* Dept. of Commerce, *U. S. Bur. Standards, Circ.* 142, 1923.
† Properties of solid ammonia at the triple point (−107.86°F.).

Table 13. Properties of Liquid Ammonia—(*Concluded*)

Temp., °F.	Saturation						Latent heat of pressure variation, B.t.u. per lb. / lb. per sq. in.	Variation of *h* with *p* (*t* constant), B.t.u. per lb. / lb. per sq. in.	Compressibility, lb. per sq. in. $\times 10^6$
t	Pressure (abs.), lb. per sq. in. p	Volume, cu. ft. per lb. v	Density, lb. per cu. ft. $\frac{1}{v}$	Specific heat, B.t.u. per lb. per °F. c	Heat content, B.t.u. per lb. h	Latent heat, B.t.u. per lb. L	l	$\left(\frac{\partial h}{\partial p}\right)_t$	$-\frac{1}{v}\left(\frac{\partial v}{\partial p}\right)_t$
50	89.19	0.02564	39.00	1.112	97.93	527.3	−0.0031	0.0017	8.4
55	98.06	.02581	38.75	1.116	103.54	522.8	− .0032	.0016	8.8
60	107.6	.02597	38.50	1.120	109.18	518.1	− .0033	.0015	9.1
65	117.8	.02614	38.25	1.125	114.85	513.4	− .0034	.0014	9.5
70	128.8	.02632	38.00	1.129	120.54	508.6	− .0035	.0013	10.0
75	140.5	.02650	37.74	1.133	126.25	503.7	− .0037	.0012	10.4
80	153.0	.02668	37.48	1.138	131.99	498.7	− .0038	.0011	10.9
85	166.4	.02687	37.21	1.142	137.75	493.6	− .0040	.0010	11.4
90	180.6	.02707	36.95	1.147	143.54	488.5	− .0041	.0009	12.0
95	195.8	.02727	36.67	1.151	149.36	483.2	− .0043	.0008	12.6
100	211.9	.02747	36.40	1.156	155.21	477.8	− .0045	.0006	13.3
105	228.9	.02769	36.12	1.162	161.09	472.3	− .0047	.0005	14.1
110	247.0	.02790	35.84	1.168	167.01	466.7	− .0049	.0003	14.9
115	266.2	.02813	35.55	1.176	172.97	460.9	− .0051	.0001	15.8
120	286.4	.02836	35.26	1.183	178.98	455.0	− .0053	.0000	16.7
125	307.8	.02860	34.96	(1.189)	(185)	(499)			
130	330.3	.02885	34.66	(1.197)	(191)	(443)			
135	354.1	.02911	34.35	(1.205)	(197)	(436)			
140	379.1	.02938	34.04	(1.213)	(203)	(430)			
145	405.5	.02966	33.72	(1.222)	(210)	(423)			
150	433.2	.02995	33.39	(1.23)	(216)	(416)			
155	462.3	.03025	33.06	(1.24)	(222)	(409)			
160	492.8	.03056	32.72	(1.25)	(229)	(401)			
165	524.8	.03089	32.37	(1.26)	(235)	(394)			
170	558.4	.03124	32.01	(1.27)	(241)	(386)			
175	593.5	.03160	31.65	(1.29)	(248)	(377)			
180	630.3	.03198	31.27	(1.30)	(255)	(369)			
185	668.7	.03238	30.88	(1.32)	(262)	(360)			
190	708.9	.03281	30.48	(1.34)	(269)	(351)			
195	750.9	.03326	30.06	(1.36)	(276)	(342)			
200	794.7	.03375	29.63	(1.38)	(283)	(332)			
210	888.1	.03482	28.72	(1.43)	(297)	(310)			
220	989.5	.0361	27.7	(1.49)	(313)	(287)			
230	1,099.5	.0376	26.6	(1.57)	(329)	(260)			
240	1,218.5	.0395	25.3	(1.70)	(346)	(229)			
250	1,347	.0422	23.7	(1.90)	(365)	(192)			
260	1,486	.0463	21.6	(2.33)	(387)	(142)			
270	1,635	.0577	17.3	(5.30)	(419)	(52)			
Critical	1,657	.0686	14.6	∞	(433)	0	−∞	−∞	∞

NOTE.—The figures in parentheses were calculated from empirical equations given in *Bur. Standards Sci. Papers* 313 and 315 and represent values obtained by extrapolation beyond the range covered in the experimental work.

Table 14. Saturated Ammonia, Temperature Table†

Temp., °F. t	Pressure		Volume of vapor, cu. ft. per lb. V	Density of vapor, lb. per cu. ft. 1/V	Heat content		Latent heat, B.t.u. per lb. L	Entropy	
	Abs., lb. per sq. in. p	Gage, lb. per sq. in. gp			Liquid, B.t.u. per lb. h	Vapor, B.t.u. per lb. H		Liquid, B.t.u. per lb. per °F. s	Vapor, B.t.u. per lb. per °F. S
-60	5.55	*18.6	44.73	0.02235	-21.2	589.6	610.8	-0.0517	1.4769
-58	5.93	*17.8	42.05	.02378	-19.1	590.4	609.5	- .0464	1.4713
-56	6.33	*17.0	39.56	.02528	-17.0	591.2	608.2	- .0412	1.4658
-54	6.75	*16.2	37.24	.02685	-14.8	592.1	606.9	- .0360	1.4604
-52	7.20	*15.3	35.09	.02850	-12.7	592.9	605.6	- .0307	1.4551
-50	7.67	*14.3	33.08	.03023	-10.6	593.7	604.3	- .0256	1.4497
-48	8.16	*13.3	31.20	.03205	- 8.5	594.4	602.9	- .0204	1.4445
-46	8.68	*12.2	29.45	.03395	- 6.4	595.2	601.6	- .0153	1.4393
-44	9.23	*11.1	27.82	.03595	- 4.3	596.0	600.3	- .0102	1.4342
-42	9.81	*10.0	26.29	.03804	- 2.1	596.8	598.9	- .0051	1.4292
-40	10.41	*8.7	24.86	.04022	0.0	597.6	597.6	.0000	1.4242
-38	11.04	*7.4	23.53	.04251	2.1	598.3	596.2	.0051	1.4193
-36	11.71	*6.1	22.27	.04489	4.3	599.1	594.8	.0101	1.4144
-34	12.41	*4.7	21.10	.04739	6.4	599.9	593.5	.0151	1.4096
-32	13.14	*3.2	20.00	.04999	8.5	600.6	592.1	.0201	1.4048
-30	13.90	*1.6	18.97	.05271	10.7	601.4	590.7	.0250	1.4001
-28	14.71	0.0	18.00	.05555	12.8	602.1	589.3	.0300	1.3955
-26	15.55	0.8	17.09	.05850	14.9	602.8	587.9	.0350	1.3909
-24	16.42	1.7	16.24	.06158	17.1	603.6	586.5	.0399	1.3863
-22	17.34	2.6	15.43	.06479	19.2	604.3	585.1	.0448	1.3818
-20	18.30	3.6	14.68	.06813	21.4	605.0	583.6	.0497	1.3774
-18	19.30	4.6	13.97	.07161	23.5	605.7	582.2	.0545	1.3729
-16	20.34	5.6	13.29	.07522	25.6	606.4	580.8	.0594	1.3686
-14	21.43	6.7	12.66	.07898	27.8	607.1	579.3	.0642	1.3643
-12	22.56	7.9	12.06	.08289	30.0	607.8	577.8	.0690	1.3600
-10	23.74	9.0	11.50	.08695	32.1	608.5	576.4	.0738	1.3558
- 8	24.97	10.3	10.97	.09117	34.3	609.2	574.9	.0786	1.3516
- 6	26.26	11.6	10.47	.09555	36.4	609.8	573.4	.0833	1.3474
- 4	27.59	12.9	9.991	.1001	38.6	610.5	571.9	.0880	1.3433
- 2	28.98	14.3	9.541	.1048	40.7	611.1	570.4	.0928	1.3393
0	30.42	15.7	9.116	.1097	42.9	611.8	568.9	.0975	1.3352
2	31.92	17.2	8.714	.1148	45.1	612.4	567.3	.1022	1.3312
4	33.47	18.8	8.333	.1200	47.2	613.0	565.8	.1069	1.3273
6	35.09	20.4	7.971	.1254	49.4	613.6	564.2	.1115	1.3234
8	36.77	22.1	7.629	.1311	51.6	614.3	562.7	.1162	1.3195
10	38.51	23.8	7.304	.1369	53.8	614.9	561.1	.1208	1.3157
12	40.31	25.6	6.996	.1429	56.0	615.5	559.5	.1254	1.3118
14	42.18	27.5	6.703	.1492	58.2	616.1	557.9	.1300	1.3081
16	44.12	29.4	6.425	.1556	60.3	616.6	556.3	.1346	1.3043
18	46.13	31.4	6.161	.1623	62.5	617.2	554.7	.1392	1.3006
20	48.21	33.5	5.910	.1692	64.7	617.8	553.1	.1437	1.2969
22	50.36	35.7	5.671	.1763	66.9	618.3	551.4	.1483	1.2933
24	52.59	37.9	5.443	.1837	69.1	618.9	549.8	.1528	1.2897
26	54.90	40.2	5.227	.1913	71.3	619.4	548.1	.1573	1.2861
28	57.28	42.6	5.021	.1992	73.5	619.9	546.4	.1618	1.2825
30	59.74	45.0	4.825	.2073	75.7	620.5	544.8	.1663	1.2790
32	62.29	47.6	4.637	.2156	77.9	621.0	543.1	.1708	1.2755
34	64.91	50.2	4.459	.2243	80.1	621.5	541.4	.1753	1.2721
36	67.63	52.9	4.289	.2332	82.3	622.0	539.7	.1797	1.2686
38	70.43	55.7	4.126	.2423	84.6	622.5	537.9	.1841	1.2652

† Dept. of Commerce, *U. S. Bur. Standards, Circ.* 142, 1923.

* In. Hg below 1 standard atm. (29.92 in.)

Table 14. Saturated Ammonia, Temperature Table—(*Concluded*)

Temp., °F. t	Pressure		Volume of vapor, cu. ft. per lb. V	Density of vapor, lb. per cu. ft. $1/V$	Heat content		Latent heat, B.t.u. per lb. L	Entropy	
	Abs., lb. per sq. in. p	Gage, lb. per sq. in. gp			Liquid, B.t.u. per lb. h	Vapor, B.t u. per lb. H		Liquid, B.t.u. per lb. per °F. s	Vapor, B.t.u. per lb. per °F. S
40	73.32	58.6	3.971	0.2518	86.8	623.0	536.2	0.1885	1.2618
42	76.31	61.6	3.823	.2616	89.0	623.4	534.4	.1930	1.2585
44	79.38	64.7	3.682	.2716	91.2	623.9	532.7	.1974	1.2552
46	82.55	67.9	3.547	.2819	93.5	624.4	530.9	.2018	1.2519
48	85.82	71.1	3.418	.2926	95.7	624.8	529.1	.2062	1.2486
50	89.19	74.5	3.294	.3036	97.9	625.2	527.3	.2105	1.2453
52	92.66	78.0	3.176	.3149	100.2	625.7	525.5	.2149	1.2421
54	96.23	81.5	3.063	.3265	102.4	626.1	523.7	.2192	1.2389
56	99.91	85.2	2.954	.3385	104.7	626.5	521.8	.2236	1.2357
58	103.7	89.0	2.851	.3508	106.9	626.9	520.0	.2279	1.2325
60	107.6	92.9	2.751	.3635	109.2	627.3	518.1	.2322	1.2294
62	111.6	96.9	2.656	.3765	111.5	627.7	516.2	.2365	1.2262
64	115.7	101.0	2.565	.3899	113.7	628.0	514.3	.2408	1.2231
66	120.0	105.3	2.477	.4037	116.0	628.4	512.4	.2451	1.2201
68	124.3	109.6	2.393	.4179	118.3	628.8	510.5	.2494	1.2170
70	128.8	114.1	2.312	.4325	120.5	629.1	508.6	.2537	1.2140
72	133.4	118.7	2.235	.4474	122.8	629.4	506.6	.2579	1.2110
74	138.1	123.4	2.161	.4628	125.1	629.8	504.7	.2622	1.2080
76	143.0	128.3	2.089	.4786	127.4	630.1	502.7	.2664	1.2050
78	147.9	133.2	2.021	.4949	129.7	630.4	500.7	.2706	1.2020
80	153.0	138.3	1.955	.5115	132.0	630.7	498.7	.2749	1.1991
82	158.3	143.6	1.892	.5287	134.3	631.0	496.7	.2791	1.1962
84	163.7	149.0	1.831	.5462	136.6	631.3	494.7	.2833	1.1933
86	169.2	154.5	1.772	.5643	138.9	631.5	492.6	.2875	1.1904
88	174.8	160.1	1.716	.5828	141.2	631.8	490.6	.2917	1.1875
90	180.6	165.9	1.661	.6019	143.5	632.0	488.5	.2958	1.1846
92	186.6	171.9	1.609	.6214	145.8	632.2	486.4	.3000	1.1818
94	192.7	178.0	1.559	.6415	148.2	632.5	484.3	.3041	1.1789
96	198.9	184.2	1.510	.6620	150.5	632.6	482.1	.3083	1.1761
98	205.3	190.6	1.464	.6832	152.9	632.9	480.0	.3125	1.1733
100	211.9	197.2	1.419	.7048	155.2	633.0	477.8	.3166	1.1705
102	218.6	203.9	1.375	.7270	157.6	633.2	475.6	.3207	1.1677
104	225.4	210.7	1.334	.7498	159.9	633.4	473.5	.3248	1.1649
106	232.5	217.8	1.293	.7732	162.3	633.5	471.2	.3289	1.1621
108	239.7	225.0	1.254	.7972	164.6	633.6	469.0	.3330	1.1593
110	247.0	232.3	1.217	.8219	167.0	633.7	466.7	.3372	1.1566
112	254.5	239.8	1.180	.8471	169.4	633.8	464.4	.3413	1.1538
114	262.2	247.5	1.145	.8730	171.8	633.9	462.1	.3453	1.1510
116	270.1	255.4	1.112	.8996	174.2	634.0	459.8	.3495	1.1483
118	278.2	263.5	1.079	.9269	176.6	634.0	457.4	.3535	1.1455
120	286.4	271.7	1.047	.9549	179.0	634.0	455.0	.3576	1.1427
122	294.8	280.1	1.017	.9837	181.4	634.0	452.6	.3618	1.1400
124	303.4	288.7	0.987	1.0132	183.9	634.0	450.1	.3659	1.1372
125	307.8	293.1	0.973	1.028	185.1	634.0	448.9	.3679	1.1358

Table 15. Saturated Ammonia, Absolute Pressure Table*

Pressure (abs.), lb. per sq. in.	Temp., °F.	Volume of vapor, cu. ft. per lb.	Density of vapor, lb. per cu. ft.	Heat content		Latent heat, B.t.u. per lb.	Entropy		
				Liquid, B.t.u. per lb.	Vapor, B.t.u. per lb.		Liquid, B.t.u. per lb. per °F.	Evaporation, B.t.u. per lb. per °F.	Vapor, B.t.u. per lb. per °F.
p	t	V	$1/V$	h	H	L	s	L/T	S
5.0	−63.11	49.31	0.02029	−24.5	588.3	612.8	−0.0599	1.5456	1.4857
7.5	−50.70	33.77	.02962	−11.3	593.4	604.7	− .0274	1.4790	1.4516
10.0	−41.34	25.81	.03874	− 1.4	597.1	598.5	− .0034	1.4310	1.4276
12.5	−33.74	20.96	.04772	6.7	600.0	593.3	.0157	1.3933	1.4090
15.0	−27.29	17.67	.05658	13.6	602.4	588.8	.0318	1.3620	1.3938
17.5	−21.66	15.30	.06535	19.6	604.4	584.8	.0456	1.3354	1.3810
20.0	−16.64	13.50	.07405	25.0	606.2	581.2	.0578	1.3122	1.3700
22.5	−12.11	12.09	.08268	29.8	607.7	577.9	.0687	1.2915	1.3602
25.0	− 7.96	10.96	.09126	34.3	609.1	574.8	.0787	1.2728	1.3515
27.5	− 4.13	10.02	.09979	38.4	610.4	572.0	.0878	1.2558	1.3436
30	− 0.57	9.236	.1083	42.3	611.6	569.3	.0962	1.2402	1.3364
35	5.89	7.991	.1251	49.3	613.6	564.3	.1113	1.2123	1.3236
40	11.66	7.047	.1419	55.6	615.4	559.8	.1246	1.1879	1.3125
45	16.88	6.307	.1586	61.3	616.9	555.6	.1366	1.1661	1.3027
50	21.67	5.710	.1751	66.5	618.2	551.7	.1475	1.1464	1.2939
55	26.09	5.218	.1917	71.4	619.4	548.0	.1575	1.1284	1.2859
60	30.21	4.805	.2081	75.9	620.5	544.6	.1668	1.1119	1.2787
65	34.06	4.453	.2245	80.2	621.5	541.3	.1754	1.0966	1.2720
70	37.70	4.151	.2409	84.2	622.4	538.2	.1835	1.0823	1.2658
75	41.13	3.887	.2573	88.0	623.2	535.2	.1910	1.0689	1.2599
80	44.40	3.655	.2736	91.7	624.0	532.3	.1982	1.0563	1.2545
85	47.50	3.449	.2899	95.1	624.7	529.6	.2051	1.0443	1.2494
90	50.47	3.266	.3062	98.4	625.3	526.9	.2115	1.0330	1.2445
95	53.32	3.101	.3225	101.6	625.9	524.3	.2177	1.0222	1.2399
100	56.05	2.952	.3388	104.7	626.5	521.8	.2237	1.0119	1.2356
110	61.21	2.693	.3713	110.5	627.5	517.0	.2348	0.9927	1.2275
120	66.02	2.476	.4039	116.0	628.4	512.4	.2452	0.9749	1.2201
130	70.53	2.291	.4364	121.1	629.2	508.1	.2548	0.9584	1.2132
140	74.79	2.132	.4690	126.0	629.9	503.9	.2638	0.9430	1.2068
150	78.81	1.994	.5016	130.6	630.5	499.9	.2724	0.9285	1.2009
160	82.64	1.872	.5343	135.0	631.1	496.1	.2804	0.9148	1.1952
170	86.29	1.764	.5670	139.3	631.6	492.3	.2881	0.9019	1.1900
180	89.78	1.667	.5998	143.3	632.0	488.7	.2954	0.8896	1.1850
190	93.13	1.581	.6326	147.2	632.4	485.2	.3024	0.8778	1.1802
200	96.34	1.502	.6656	150.9	632.7	481.8	.3090	0.8666	1.1756
225	103.87	1.336	.7484	159.7	633.3	473.6	.3246	0.8405	1.1651
250	110.80	1.202	.8319	168.0	633.8	465.8	.3388	0.8167	1.1555
275	117.22	1.091	.9162	175.6	634.0	458.4	.3519	0.7947	1.1466
300	123.21	0.999	1.0015	182.9	634.0	451.1	.3642	0.7741	1.1383

* Dept. of Commerce, *U. S. Bur. Standards, Circ.* 142, 1923.

Table 16. Properties of Superheated Ammonia Vapor*

(V = volume in cu. ft. per lb.; H = heat content in B.t.u. per lb.; S = entropy in B.t.u. per lb. per °F.)

Temp., °F.	Absolute pressure in lb. per sq. in. (Saturation temperature in italics.)								
	5 *−63.11°*			6 *−57.64°*			7 *−52.88°*		
	V	*H*	*S*	*V*	*H*	*S*	*V*	*H*	*S*
Sat.	*49.31*	*588.3*	*1.4857*	*41.59*	*590.6*	*1.4703*	*36.01*	*592.5*	*1.4574*
−50	51.05	595.2	1.5025	42.44	594.6	1.4803	36.29	594.0	1.4611
−40	52.36	600.3	1.5149	43.55	599.8	1.4928	37.25	599.3	1.4739
−30	53.67	605.4	1.5269	44.64	604.9	1.5049	38.19	604.5	1.4861
−20	54.97	610.4	1.5385	45.73	610.0	1.5166	39.13	609.6	1.4979
−10	56.26	615.4	1.5498	46.82	615.1	1.5280	40.07	614.7	1.5094
0	57.55	620.4	1.5608	47.90	620.1	1.5391	41.00	619.8	1.5206
10	58.84	625.4	1.5716	48.98	625.2	1.5499	41.93	624.9	1.5314
20	60.12	630.4	1.5821	50.05	630.2	1.5605	42.85	629.9	1.5421
30	61.41	635.4	1.5925	51.12	635.2	1.5708	43.77	635.0	1.5525
40	62.69	640.4	1.6026	52.19	640.2	1.5810	44.69	640.0	1.5627
50	63.96	645.5	1.6125	53.26	645.2	1.5910	45.61	645.0	1.5727
60	65.24	650.5	1.6223	54.32	650.3	1.6008	46.53	650.1	1.5825
70	66.51	655.5	1.6319	55.39	655.3	1.6104	47.44	655.2	1.5921
80	67.79	660.6	1.6413	56.45	660.4	1.6199	48.36	660.2	1.6016
90	69.06	665.6	1.6506	57.51	665.5	1.6292	49.27	665.3	1.6110
100	70.33	670.7	1.6598	58.58	670.6	1.6384	50.18	670.4	1.6202
110	71.60	675.8	1.6689	59.64	675.7	1.6474	51.09	675.5	1.6292
120	72.87	680.9	1.6778	60.70	680.8	1.6563	52.00	680.7	1.6382
130	74.14	686.1	1.6865	61.76	685.9	1.6651	52.91	685.8	1.6470
140	75.41	691.2	1.6952	62.82	691.1	1.6738	53.82	691.0	1.6557
150	76.68	696.4	1.7038	63.87	696.3	1.6824	54.73	696.2	1.6643
160	77.95	701.6	1.7122	64.93	701.5	1.6909	55.63	701.4	1.6727
170	79.21	706.8	1.7206	65.99	706.7	1.6992	56.54	706.6	1.6811
180	80.48	712.1	1.7289	67.05	712.0	1.7075	57.45	711.9	1.6894
	8 *−48.64°*			9 *−44.38°*			10 *−41.34°*		
Sat.	*31.79*	*594.2*	*1.4462*	*28.48*	*595.7*	*1.4363*	*25.81*	*597.1*	*1.4276*
−50									
−40	32.52	598.8	1.4573	28.85	598.3	1.4426			
−30	33.36	604.1	1.4697	29.59	603.6	1.4551	26.58	603.2	1.4420
−20	34.19	609.3	1.4816	30.34	608.9	1.4672	27.26	608.5	1.4542
−10	35.01	614.4	1.4932	31.07	614.0	1.4788	27.92	613.7	1.4659
0	35.83	619.5	1.5044	31.80	619.2	1.4902	28.58	618.9	1.4773
10	36.64	624.6	1.5154	32.53	624.3	1.5012	29.24	624.0	1.4884
20	37.45	629.7	1.5261	33.26	629.4	1.5119	29.90	629.1	1.4992
30	38.26	634.7	1.5365	33.98	634.5	1.5224	30.55	634.2	1.5097
40	39.07	639.8	1.5467	34.70	639.5	1.5327	31.20	639.3	1.5200
50	39.88	644.8	1.5568	35.42	644.6	1.5427	31.85	644.4	1.5301
60	40.68	649.9	1.5666	36.13	649.7	1.5526	32.49	649.5	1.5400
70	41.48	655.0	1.5763	36.85	654.8	1.5623	33.14	654.6	1.5497
80	42.28	660.1	1.5858	37.56	659.9	1.5718	33.78	659.7	1.5593
90	43.08	665.2	1.5952	38.27	665.0	1.5812	34.42	664.8	1.5687
100	43.88	670.3	1.6044	38.98	670.1	1.5904	35.07	670.0	1.5779
110	44.68	675.4	1.6135	39.70	675.3	1.5995	35.71	675.1	1.5870
120	45.48	680.5	1.6224	40.40	680.4	1.6085	36.35	680.3	1.5960
130	46.27	685.7	1.6312	41.11	685.6	1.6173	36.99	685.4	1.6049
140	47.07	690.9	1.6399	41.82	690.7	1.6260	37.62	690.6	1.6136
150	47.87	696.1	1.6485	42.53	695.9	1.6346	38.26	695.8	1.6222
160	48.66	701.3	1.6570	43.24	701.2	1.6431	38.90	701.1	1.6307
170	49.46	706.5	1.6654	43.95	706.4	1.6515	39.54	706.3	1.6391
180	50.25	711.8	1.6737	44.65	711.7	1.6598	40.17	711.6	1.6474
190	.....	.....	......	.....	.....	......	40.81	716.9	1.6556
200	.....	.....	......	.....	.....	......	41.45	722.2	1.6637

* Dept. of Commerce, *U. S. Bur. Standards, Circ.* 142, 1923.

Table 16. Properties of Superheated Ammonia Vapor—(*Continued*)

V = volume in cu. ft. per lb.; H = heat content in B.t.u. per lb.; S = entropy in B.t.u. per lb. per °F.

Temp., °F.	Absolute pressure in lb. per sq. in. (Saturation temperature in italics.)								
	12 *−35.16°*			14 *−29.76°*			16 *−24.95°*		
	V	H	S	V	H	S	V	H	S
Sat.	*21.77*	*599.4*	*1.4124*	*18.85*	*601.4*	*1.3996*	*16.64*	*603.2*	*1.3885*
−30	22.07	602.3	1.4190						
−20	22.64	607.7	1.4314	19.33	606.8	1.4119	16.86	606.0	1.3948
−10	23.20	613.0	1.4434	19.82	612.2	1.4241	17.29	611.5	1.4072
0	23.75	618.2	1.4549	20.30	617.6	1.4358	17.72	616.9	1.4191
10	24.31	623.4	1.4661	20.78	622.8	1.4472	18.14	622.2	1.4306
20	24.86	628.6	1.4770	21.26	628.0	1.4582	18.56	627.5	1.4417
30	25.41	633.7	1.4877	21.73	633.2	1.4688	18.97	632.7	1.4525
40	25.95	638.9	1.4980	22.20	638.4	1.4793	19.39	638.0	1.4630
50	26.49	644.0	1.5082	22.67	643.6	1.4896	19.80	643.2	1.4733
60	27.03	649.1	1.5182	23.14	648.7	1.4996	20.21	648.3	1.4834
70	27.57	654.3	1.5279	23.60	653.9	1.5094	20.62	653.5	1.4933
80	28.11	659.4	1.5375	24.06	659.0	1.5191	21.03	658.7	1.5030
90	28.65	664.5	1.5470	24.53	664.2	1.5285	21.43	663.9	1.5125
100	29.19	669.7	1.5562	24.99	669.4	1.5378	21.84	669.1	1.5218
110	29.72	674.8	1.5654	25.45	674.5	1.5470	22.24	674.3	1.5310
120	30.26	680.0	1.5744	25.91	679.7	1.5560	22.65	679.5	1.5401
130	30.79	685.2	1.5833	26.37	684.9	1.5649	23.05	684.7	1.5490
140	31.33	690.4	1.5920	26.83	690.1	1.5737	23.45	689.9	1.5578
150	31.86	695.6	1.6006	27.29	695.4	1.5824	23.86	695.1	1.5665
160	32.39	700.8	1.6092	27.74	700.6	1.5909	24.26	700.4	1.5750
170	32.92	706.1	1.6176	28.20	705.9	1.5993	24.66	705.7	1.5835
180	33.46	711.4	1.6259	28.66	711.2	1.6076	25.06	711.0	1.5918
190	33.99	716.7	1.6341	29.11	716.5	1.6159	25.46	716.3	1.6001
200	34.52	722.0	1.6422	29.57	721.8	1.6240	25.86	721.6	1.6082
220	.....	.....	.....	.....	.....	.....	26.66	732.3	1.6242
	18 *−20.61°*			20 *−16.64°*			22 *−12.98°*		
Sat.	*14.90*	*604.8*	*1.3787*	*13.50*	*606.2*	*1.3700*	*12.35*	*607.4*	*1.3621*
−20	14.93	605.1	1.3795						
−10	15.32	610.7	1.3921	13.74	610.0	1.3784	12.45	609.2	1.3659
0	15.70	616.2	1.4042	14.09	615.5	1.3907	12.77	614.8	1.3784
10	16.08	621.6	1.4158	14.44	621.0	1.4025	13.09	620.4	1.3903
20	16.46	626.9	1.4270	14.78	626.4	1.4138	13.40	625.8	1.4018
30	16.83	632.2	1.4380	15.11	631.7	1.4248	13.71	631.2	1.4129
40	17.20	637.5	1.4486	15.45	637.0	1.4356	14.02	636.6	1.4237
50	17.57	642.7	1.4590	15.78	642.3	1.4460	14.32	641.9	1.4342
60	17.94	647.9	1.4691	16.12	647.5	1.4562	14.63	647.1	1.4445
70	18.30	653.1	1.4790	16.45	652.8	1.4662	14.93	652.4	1.4545
80	18.67	658.4	1.4887	16.78	658.0	1.4760	15.23	657.7	1.4643
90	19.03	663.6	1.4983	17.10	663.2	1.4856	15.53	662.9	1.4740
100	19.39	668.8	1.5077	17.43	668.5	1.4950	15.83	668.1	1.4834
110	19.75	674.0	1.5169	17.76	673.7	1.5042	16.12	673.4	1.4927
120	20.11	679.2	1.5260	18.08	678.9	1.5133	16.42	678.6	1.5019
130	20.47	684.4	1.5349	18.41	684.2	1.5223	16.72	683.9	1.5109
140	20.83	689.7	1.5438	18.73	689.4	1.5312	17.01	689.2	1.5197
150	21.19	694.9	1.5525	19.05	694.7	1.5399	17.31	694.4	1.5285
160	21.54	700.2	1.5610	19.37	700.0	1.5485	17.60	699.7	1.5371
170	21.90	705.5	1.5695	19.70	705.3	1.5569	17.89	705.0	1.5456
180	22.26	710.8	1.5778	20.02	710.6	1.5653	18.19	710.4	1.5539
190	22.61	716.1	1.5861	20.34	715.9	1.5736	18.48	715.7	1.5622
200	22.97	721.4	1.5943	20.66	721.2	1.5817	18.77	721.1	1.5704
220	23.68	732.2	1.6103	21.30	732.0	1.5978	19.35	731.8	1.5865
240	.....	.....	.....	21.94	742.8	1.6135	19.94	742.7	1.6022

Table 16. Properties of Superheated Ammonia Vapor—(*Continued*)

V = volume in cu. ft. per lb.; H = heat content in B.t.u. per lb.; S = entropy in B.t.u. per lb. per °F.

Temp., °F.	Absolute pressure in lb. per sq. in. (Saturation temperature in italics.)								
	24 *−9.58°*			26 *−6.39°*			28 *−3.40°*		
	V	H	S	V	H	S	V	H	S
Sat.	*11.89*	*608.6*	*1.3549*	*10.56*	*609.7*	*1.3482*	*9.853*	*610.7*	*1.3421*
0	11.67	614.1	1.3670	10.74	613.4	1.3564	9.942	612.7	1.3465
10	11.96	619.7	1.3791	11.01	619.1	1.3686	10.20	618.4	1.3589
20	12.25	625.2	1.3907	11.28	624.7	1.3804	10.45	624.1	1.3708
30	12.54	630.7	1.4019	11.55	630.2	1.3917	10.70	629.6	1.3822
40	12.82	636.1	1.4128	11.81	635.6	1.4027	10.95	635.1	1.3933
50	13.11	641.4	1.4234	12.08	641.0	1.4134	11.19	640.5	1.4041
60	13.39	646.7	1.4337	12.34	646.3	1.4238	11.44	645.9	1.4145
70	13.66	652.0	1.4438	12.59	651.6	1.4339	11.68	651.2	1.4247
80	13.94	657.3	1.4537	12.85	656.9	1.4439	11.92	656.6	1.4347
90	14.22	662.6	1.4634	13.11	662.2	1.4536	12.15	661.9	1.4445
100	14.49	667.8	1.4729	13.36	667.5	1.4631	12.39	667.2	1.4540
110	14.76	673.1	1.4822	13.61	672.8	1.4725	12.63	672.5	1.4634
120	15.04	678.4	1.4914	13.87	678.1	1.4817	12.86	677.8	1.4726
130	15.31	683.6	1.5004	14.12	683.4	1.4907	13.10	683.1	1.4817
140	15.58	688.9	1.5093	14.37	688.7	1.4996	13.33	688.4	1.4906
150	15.85	694.2	1.5180	14.62	694.0	1.5084	13.56	693.7	1.4994
160	16.12	699.5	1.5266	14.87	699.3	1.5170	13.80	699.1	1.5081
170	16.39	704.8	1.5352	15.12	704.6	1.5256	14.03	704.4	1.5167
180	16.66	710.2	1.5436	15.37	710.0	1.5340	14.26	709.8	1.5251
190	16.93	715.5	1.5518	15.62	715.3	1.5423	14.49	715.1	1.5334
200	17.20	720.9	1.5600	15.86	720.7	1.5505	14.72	720.5	1.5416
220	17.73	731.7	1.5761	16.36	731.5	1.5666	15.18	731.3	1.5578
240	18.27	742.6	1.5919	16.85	742.4	1.5824	15.64	742.2	1.5736
260	.....	.....	......	17.35	753.3	1.5978	16.10	753.2	1.5890
	30 *−0.67°*			32 *+2.11°*			34 *4.66°*		
Sat.	*9.236*	*611.6*	*1.3364*	*8.693*	*612.4*	*1.3310*	*8.211*	*613.2*	*1.3260*
0	9.250	611.9	1.3371						
10	9.492	617.8	1.3497	8.874	617.1	1.3411	8.328	616.4	1.3328
20	9.731	623.5	1.3618	9.099	622.9	1.3532	8.542	622.3	1.3452
30	9.966	629.1	1.3733	9.321	628.5	1.3649	8.753	628.0	1.3570
40	10.20	634.6	1.3845	9.540	634.1	1.3762	8.960	633.6	1.3684
50	10.43	640.1	1.3953	9.757	639.6	1.3871	9.166	639.2	1.3793
60	10.65	645.5	1.4059	9.972	645.1	1.3977	9.369	644.7	1.3900
70	10.88	650.9	1.4161	10.18	650.5	1.4080	9.570	650.1	1.4004
80	11.10	656.2	1.4261	10.40	655.9	1.4181	9.770	655.5	1.4105
90	11.33	661.6	1.4359	10.61	661.2	1.4280	9.969	660.9	1.4204
100	11.55	666.9	1.4456	10.81	666.6	1.4376	10.17	666.3	1.4301
110	11.77	672.2	1.4550	11.02	671.9	1.4470	10.36	671.6	1.4396
120	11.99	677.5	1.4642	11.23	677.3	1.4563	10.56	677.0	1.4489
130	12.21	682.9	1.4733	11.44	682.6	1.4655	10.75	682.3	1.4581
140	12.43	688.2	1.4823	11.64	687.9	1.4744	10.95	687.7	1.4671
150	12.65	693.5	1.4911	11.85	693.3	1.4833	11.14	693.0	1.4759
160	12.87	698.8	1.4998	12.05	698.6	1.4920	11.33	698.4	1.4846
170	13.08	704.2	1.5083	12.26	704.0	1.5006	11.53	703.8	1.4932
180	13.30	709.6	1.5168	12.46	709.4	1.5090	11.72	709.2	1.5017
190	13.52	714.9	1.5251	12.66	714.7	1.5174	11.91	714.5	1.5101
200	13.73	720.3	1.5334	12.86	720.1	1.5256	12.10	720.0	1.5183
220	14.16	731.1	1.5495	13.27	731.0	1.5418	12.48	730.8	1.5346
240	14.59	742.0	1.5653	13.67	741.9	1.5576	12.86	741.7	1.5504
260	15.02	753.0	1.5808	14.08	752.9	1.5731	13.24	752.7	1.5659
280	.....	.....	.....	14.48	763.9	1.5883	13.62	763.8	1.5811

Table 16. Properties of Superheated Ammonia Vapor—(*Continued*)

V = volume in cu. ft. per lb.; H = heat content in B.t.u. per lb.; S = entropy in B.t.u. per lb. per °F.

Temp., °F.	Absolute pressure in lb. per sq. in. (Saturation temperature in italics.)								
	36 *7.09°*			38 *9.42°*			40 *11.66°*		
	V	H	S	V	H	S	V	H	S
Sat.	*7.782*	*614.0*	*1.3213*	*7.396*	*614.7*	*1.3168*	*7.047*	*615.4*	*1.3125*
10	7.842	615.7	1.3250	7.407	615.0	1.3175			
20	8.046	621.7	1.3375	7.603	621.0	1.3301	7.203	620.4	1.3231
30	8.247	627.4	1.3494	7.795	626.9	1.3422	7.387	626.3	1.3353
40	8.445	633.1	1.3609	7.983	632.6	1.3538	7.568	632.1	1.3470
50	8.640	638.7	1.3720	8.170	638.3	1.3650	7.746	637.8	1.3583
60	8.833	644.2	1.3827	8.353	643.8	1.3758	7.922	643.4	1.3692
70	9.024	649.7	1.3932	8.535	649.3	1.3863	8.096	648.9	1.3797
80	9.214	655.2	1.4033	8.716	654.8	1.3965	8.268	654.4	1.3900
90	9.402	660.6	1.4133	8.895	660.2	1.4065	8.439	659.9	1.4000
100	9.589	666.0	1.4230	9.073	665.6	1.4163	8.609	665.3	1.4098
110	9.775	671.3	1.4325	9.250	671.0	1.4258	8.777	670.7	1.4194
120	9.961	676.7	1.4419	9.426	676.4	1.4352	8.945	676.1	1.4288
130	10.15	682.1	1.4510	9.602	681.8	1.4444	9.112	681.5	1.4381
140	10.33	687.4	1.4601	9.776	687.2	1.4534	9.278	686.9	1.4471
150	10.51	692.8	1.4689	9.950	692.6	1.4623	9.444	692.3	1.4561
160	10.69	698.2	1.4777	10.12	698.0	1.4711	9.609	697.7	1.4648
170	10.88	703.6	1.4863	10.30	703.3	1.4797	9.774	703.1	1.4735
180	11.06	709.0	1.4948	10.47	708.7	1.4883	9.938	708.5	1.4820
190	11.24	714.4	1.5032	10.64	714.2	1.4966	10.10	714.0	1.4904
200	11.42	719.8	1.5115	10.81	719.6	1.5049	10.27	719.4	1.4987
220	11.78	730.6	1.5277	11.16	730.5	1.5212	10.59	730.3	1.5150
240	12.14	741.6	1.5436	11.50	741.4	1.5371	10.92	741.3	1.5309
260	12.50	752.6	1.5591	11.84	752.4	1.5526	11.24	752.3	1.5465
280	12.86	763.7	1.5743	12.18	763.5	1.5678	11.56	763.4	1.5617
300	......	......	......	......	......	......	11.88	774.6	1.5766

	44 *15.88°*			48 *19.80°*			52 *23.48°*		
Sat.	*6.442*	*616.6*	*1.3046*	*5.934*	*617.7*	*1.2973*	*5.502*	*618.7*	*1.2906*
20	6.513	619.1	1.3099	5.937	617.8	1.2976			
30	6.683	625.2	1.3224	6.096	624.0	1.3103	5.599	622.8	1.2991
40	6.850	631.1	1.3343	6.251	630.0	1.3225	5.744	629.0	1.3114
50	7.014	636.8	1.3457	6.404	635.9	1.3341	5.887	634.9	1.3233
60	7.176	642.5	1.3567	6.554	641.6	1.3453	6.027	640.8	1.3346
70	7.336	648.1	1.3674	6.702	647.3	1.3561	6.165	646.5	1.3456
80	7.494	653.7	1.3778	6.848	652.9	1.3666	6.302	652.2	1.3562
90	7.650	659.2	1.3880	6.993	658.5	1.3768	6.437	657.8	1.3665
100	7.806	664.7	1.3978	7.137	664.0	1.3868	6.571	663.4	1.3766
110	7.960	670.1	1.4075	7.280	669.5	1.3965	6.704	668.9	1.3864
120	8.114	675.6	1.4170	7.421	675.0	1.4061	6.835	674.4	1.3960
130	8.267	681.0	1.4263	7.562	680.5	1.4154	6.966	679.9	1.4054
140	8.419	686.4	1.4354	7.702	685.9	1.4246	7.096	685.4	1.4146
150	8.570	691.9	1.4444	7.842	691.4	1.4336	7.225	690.9	1.4237
160	8.721	697.3	1.4532	7.981	696.8	1.4425	7.354	696.4	1.4326
170	8.871	702.7	1.4619	8.119	702.3	1.4512	7.483	701.8	1.4413
180	9.021	708.1	1.4704	8.257	707.7	1.4598	7.611	707.3	1.4500
190	9.171	713.6	1.4789	8.395	713.2	1.4683	7.738	712.8	1.4585
200	9.320	719.0	1.4872	8.532	718.7	1.4766	7.865	718.3	1.4668
210	......	......	......	......	......	......	7.992	723.8	1.4751
220	9.617	730.0	1.5035	8.805	729.6	1.4930	8.118	729.3	1.4833
240	9.913	741.0	1.5195	9.077	740.6	1.5090	8.370	740.3	1.4993
260	10.21	752.0	1.5350	9.348	751.7	1.5246	8.621	751.4	1.5149
280	10.50	763.1	1.5503	9.619	762.9	1.5399	8.871	762.6	1.5303
300	10.80	774.3	1.5652	9.888	774.1	1.5548	9.120	773.8	1.5453

Table 16. Properties of Superheated Ammonia Vapor—(*Continued*)

V = volume in cu. ft. per lb.; H = heat content in B.t.u. per lb.; S = entropy in B.t.u. per lb. per °F.

Temp., °F.	Absolute pressure in lb. per sq. in. (Saturation temperature in italics.)								
	56 *26.94°*			60 *30.21°*			70 *37.70°*		
	V	H	S	V	H	S	V	H	S
Sat.	*5.129*	*619.7*	*1.2844*	*4.805*	*620.5*	*1.2787*	*4.151*	*622.4*	*1.2658*
30	5.172	621.6	1.2884						
40	5.310	627.9	1.3011	4.933	626.8	1.2913	4.177	623.9	1.2688
50	5.444	633.9	1.3131	5.060	632.9	1.3035	4.290	630.4	1.2816
60	5.576	639.9	1.3246	5.184	639.0	1.3152	4.401	636.6	1.2937
70	5.706	645.7	1.3357	5.307	644.9	1.3265	4.509	642.7	1.3054
80	5.834	651.4	1.3465	5.428	650.7	1.3373	4.615	648.7	1.3166
90	5.960	657.1	1.3569	5.547	656.4	1.3479	4.719	654.6	1.3274
100	6.085	662.7	1.3670	5.665	662.1	1.3581	4.822	660.4	1.3378
110	6.209	668.3	1.3769	5.781	667.7	1.3681	4.924	666.1	1.3480
120	6.333	673.9	1.3866	5.897	673.3	1.3778	5.025	671.8	1.3579
130	6.455	679.4	1.3961	6.012	678.9	1.3873	5.125	677.5	1.3676
140	6.576	684.9	1.4053	6.126	684.4	1.3966	5.224	683.1	1.3770
150	6.697	690.4	1.4144	6.239	689.9	1.4058	5.323	688.7	1.3863
160	6.817	695.9	1.4234	6.352	695.5	1.4148	5.420	694.3	1.3954
170	6.937	701.4	1.4322	6.464	701.0	1.4236	5.518	699.9	1.4043
180	7.056	706.9	1.4408	6.576	706.5	1.4323	5.615	705.5	1.4131
190	7.175	712.4	1.4494	6.687	712.0	1.4409	5.711	711.0	1.4217
200	7.294	717.9	1.4578	6.798	717.5	1.4493	5.807	716.6	1.4302
210	7.412	723.4	1.4661	6.909	723.1	1.4576	5.902	722.2	1.4386
220	7.529	728.9	1.4743	7.019	728.6	1.4658	5.998	727.7	1.4469
230	......	......	......	......	......	......	6.093	733.3	1.4550
240	7.764	740.0	1.4903	7.238	739.7	1.4819	6.187	738.9	1.4631
260	7.998	751.1	1.5060	7.457	750.9	1.4976	6.376	750.1	1.4789
280	8.230	762.3	1.5213	7.675	762.1	1.5130	6.563	761.4	1.4943
300	8.462	773.6	1.5364	7.892	773.3	1.5281	6.750	772.7	1.5095
	80 *44.40°*			90 *50.47°*			100 *56.05°*		
Sat.	*3.655*	*624.0*	*1.2545*	*3.266*	*625.3*	*1.2445*	*2.952*	*626.5*	*1.2356*
50	3.712	627.7	1.2619						
60	3.812	634.3	1.2745	3.353	631.8	1.2571	2.985	629.3	1.2409
70	3.909	640.6	1.2866	3.442	638.3	1.2695	3.068	636.0	1.2539
80	4.005	646.7	1.2981	3.529	644.7	1.2814	3.149	642.6	1.2661
90	4.098	652.8	1.3092	3.614	650.9	1.2928	3.227	649.0	1.2778
100	4.190	658.7	1.3199	3.698	657.0	1.3038	3.304	655.2	1.2891
110	4.281	664.6	1.3303	3.780	663.0	1.3144	3.380	661.3	1.2999
120	4.371	670.4	1.3404	3.862	668.9	1.3247	3.454	667.3	1.3104
130	4.460	676.1	1.3502	3.942	674.7	1.3347	3.527	673.3	1.3206
140	4.548	681.8	1.3598	4.021	680.5	1.3444	3.600	679.2	1.3305
150	4.635	687.5	1.3692	4.100	686.3	1.3539	3.672	685.0	1.3401
160	4.722	693.2	1.3784	4.178	692.0	1.3633	3.743	690.8	1.3495
170	4.808	698.8	1.3874	4.255	697.7	1.3724	3.813	696.6	1.3588
180	4.893	704.4	1.3963	4.332	703.4	1.3813	3.883	702.3	1.3678
190	4.978	710.0	1.4050	4.408	709.0	1.3901	3.952	708.0	1.3767
200	5.063	715.6	1.4136	4.484	714.7	1.3988	4.021	713.7	1.3854
210	5.147	721.3	1.4220	4.560	720.4	1.4073	4.090	719.4	1.3940
220	5.231	726.9	1.4304	4.635	726.0	1.4157	4.158	725.1	1.4024
230	5.315	732.5	1.4386	4.710	731.7	1.4239	4.226	730.8	1.4108
240	5.398	738.1	1.4467	4.785	737.3	1.4321	4.294	736.5	1.4190
250	5.482	743.8	1.4547	4.859	743.0	1.4401	4.361	742.2	1.4271
260	5.565	749.4	1.4626	4.933	748.7	1.4481	4.428	747.9	1.4350
280	5.730	760.7	1.4781	5.081	760.0	1.4637	4.562	759.4	1.4507
300	5.894	772.1	1.4933	5.228	771.5	1.4789	4.695	770.8	1.4660

Table 16. Properties of Superheated Ammonia Vapor—(*Continued*)

V = volume in cu. ft. per lb.; *H* = heat content in B.t.u. per lb.; *S* = entropy in B.t.u. per lb. per °F.

Temp., °F.	Absolute pressure in lb. per sq. in. (Saturation temperature in italics.)								
	110 *61.21°*			120 *66.02°*			130 *70.53°*		
	V	*H*	*S*	*V*	*H*	*S*	*V*	*H*	*S*
Sat.	*2.698*	*627.5*	*1.2275*	*2.476*	*628.4*	*1.2201*	*2.291*	*629.2*	*1.2132*
70	2.761	633.7	1.2392	2.505	631.3	1.2255			
80	2.837	640.5	1.2519	2.576	638.3	1.2386	2.355	636.0	1.2260
90	2.910	647.0	1.2640	2.645	645.0	1.2510	2.421	643.0	1.2388
100	2.981	653.4	1.2755	2.712	651.6	1.2628	2.484	649.7	1.2509
110	3.051	659.7	1.2866	2.778	658.0	1.2741	2.546	656.3	1.2625
120	3.120	665.8	1.2972	2.842	664.2	1.2850	2.606	662.7	1.2736
130	3.188	671.9	1.3076	2.905	670.4	1.2956	2.665	688.9	1.2843
140	3.255	677.8	1.3176	2.967	676.5	1.3058	2.724	675.1	1.2947
150	3.321	683.7	1.3274	3.029	682.5	1.3157	2.781	681.2	1.3048
160	3.386	689.6	1.3370	3.089	688.4	1.3254	2.838	687.2	1.3146
170	3.451	695.4	1.3463	3.149	694.3	1.3348	2.894	693.2	1.3241
180	3.515	701.2	1.3555	3.209	700.2	1.3441	2.949	699.1	1.3335
190	3.579	707.0	1.3644	3.268	706.0	1.3531	3.004	705.0	1.3426
200	3.642	712.8	1.3732	3.326	711.8	1.3620	3.059	710.9	1.3516
210	3.705	718.5	1.3819	3.385	717.6	1.3707	3.113	716.7	1.3604
220	3.768	724.3	1.3904	3.442	723.4	1.3793	3.167	722.5	1.3690
230	3.830	730.0	1.3988	3.500	729.2	1.3877	3.220	728.3	1.3775
240	3.892	735.7	1.4070	3.557	734.9	1.3960	3.273	734.1	1.3858
250	3.954	741.5	1.4151	3.614	740.7	1.4042	3.326	739.9	1.3941
260	4.015	747.2	1.4232	3.671	746.5	1.4123	3.379	745.7	1.4022
270	4.076	752.9	1.4311	3.727	752.2	1.4202	3.431	751.5	1.4102
280	4.137	758.7	1.4389	3.783	758.0	1.4281	3.483	757.3	1.4181
290	4.198	764.5	1.4466	3.839	763.8	1.4359	3.535	763.1	1.4259
300	4.259	770.2	1.4543	3.895	769.6	1.4435	3.587	769.0	1.4336
320	.....	.....	.....	.....	.....	.....	3.690	780.6	1.4487
	140 *74.79°*			150 *78.81°*			160 *82.64°*		
Sat.	*2.132*	*629.9*	*1.2068*	*1.994*	*630.5*	*1.2009*	*1.872*	*631.1*	*1.1952*
80	2.166	633.8	1.2140	2.001	631.4	1.2025			
90	2.228	640.9	1.2272	2.061	638.8	1.2161	1.914	636.6	1.2055
100	2.288	647.8	1.2396	2.118	645.9	1.2289	1.969	643.9	1.2186
110	2.347	654.5	1.2515	2.174	652.8	1.2410	2.023	651.0	1.2311
120	2.404	661.1	1.2628	2.228	659.4	1.2526	2.075	657.8	1.2429
130	2.460	667.4	1.2738	2.281	665.9	1.2638	2.125	664.4	1.2542
140	2.515	673.7	1.2843	2.334	672.3	1.2745	2.175	670.9	1.2652
150	2.569	679.9	1.2945	2.385	678.6	1.2849	2.224	677.2	1.2757
160	2.622	686.0	1.3045	2.435	684.8	1.2949	2.272	683.5	1.2859
170	2.675	692.0	1.3141	2.485	690.9	1.3047	2.319	689.7	1.2958
180	2.727	698.0	1.3236	2.534	696.9	1.3142	2.365	695.8	1.3054
190	2.779	704.0	1.3328	2.583	702.9	1.3236	2.411	701.9	1.3148
200	2.830	709.9	1.3418	2.631	708.9	1.3327	2.457	707.9	1.3240
210	2.880	715.8	1.3507	2.679	714.8	1.3416	2.502	713.9	1.3331
220	2.931	721.6	1.3594	2.276	720.7	1.3504	2.547	719.9	1.3419
230	2.981	727.5	1.3679	2.773	726.6	1.3590	2.591	725.8	1.3506
240	3.030	733.3	1.3763	2.820	732.5	1.3675	2.635	731.7	1.3591
250	3.080	739.2	1.3846	2.866	738.4	1.3758	2.679	737.6	1.3675
260	3.129	745.0	1.3928	2.912	744.3	1.3840	2.723	743.5	1.3757
270	3.179	750.8	1.4008	2.958	750.1	1.3921	2.766	749.4	1.3838
280	3.227	756.7	1.4088	3.004	756.0	1.4001	2.809	755.3	1.3919
290	3.275	762.5	1.4166	3.049	761.8	1.4079	2.852	761.2	1.3998
300	3.323	768.3	1.4243	3.095	767.7	1.4157	2.895	767.1	1.4076
320	3.420	780.0	1.4395	3.185	779.4	1.4310	2.980	778.9	1.4229
340	.....	.....	.....	.....	.....	.....	3.064	790.7	1.4379

Table 16. Properties of Superheated Ammonia Vapor—(*Continued*)

V = volume in cu. ft. per lb.; H = heat content in B.t.u. per lb.; S = entropy in B.t.u. per lb. per °F.

Temp., °F.	Absolute pressure in lb. per sq. in. (Saturation temperature in italics.)								
	170 *86.29°*			180 *89.78°*			190 *93.13°*		
	V	H	S	V	H	S	V	H	S
Sat.	*1.764*	*631.6*	*1.1900*	*1.667*	*632.0*	*1.1850*	*1.581*	*632.4*	*1.1802*
90	1.784	634.4	1.1952	1.668	632.2	1.1853			
100	1.837	641.9	1.2087	1.720	639.9	1.1992	1.615	637.8	1.1899
110	1.889	649.1	1.2215	1.770	647.3	1.2123	1.663	645.4	1.2034
120	1.939	656.1	1.2336	1.818	654.4	1.2247	1.710	652.6	1.2160
130	1.988	662.8	1.2452	1.865	661.3	1.2364	1.755	659.7	1.2281
140	2.035	669.4	1.2563	1.910	668.0	1.2477	1.799	666.5	1.2396
150	2.081	675.9	1.2669	1.955	674.6	1.2586	1.842	673.2	1.2506
160	2.127	682.3	1.2773	1.999	681.0	1.2691	1.884	679.7	1.2612
170	2.172	688.5	1.2873	2.042	687.3	1.2792	1.925	686.1	1.2715
180	2.216	694.7	1.2971	2.084	693.6	1.2891	1.966	692.5	1.2815
190	2.260	700.8	1.3066	2.126	699.8	1.2987	2.005	698.7	1.2912
200	2.303	706.9	1.3159	2.167	705.9	1.3081	2.045	704.9	1.3007
210	2.346	713.0	1.3249	2.208	712.0	1.3172	2.084	711.1	1.3099
220	2.389	719.0	1.3338	2.248	718.1	1.3262	2.123	717.2	1.3189
230	2.431	724.9	1.3426	2.288	724.1	1.3350	2.161	723.2	1.3278
240	2.473	730.9	1.3512	2.328	730.1	1.3436	2.199	729.3	1.3365
250	2.514	736.8	1.3596	2.367	736.1	1.3521	2.236	735.3	1.3450
260	2.555	742.8	1.3679	2.407	742.0	1.3605	2.274	741.3	1.3534
270	2.596	748.7	1.3761	2.446	748.0	1.3687	2.311	747.3	1.3617
280	2.637	754.6	1.3841	2.484	753.9	1.3768	2.348	753.2	1.3698
290	2.678	760.5	1.3921	2.523	759.9	1.3847	2.384	759.2	1.3778
300	2.718	766.4	1.3999	2.561	765.8	1.3926	2.421	765.2	1.3857
320	2.798	778.3	1.4153	2.637	777.7	1.4081	2.493	777.1	1.4012
340	2.878	790.1	1.4303	2.713	789.6	1.4231	2.565	789.0	1.4163
	200 *96.34°*			220 *102.42°*			240 *108.09°*		
Sat.	*1.502*	*632.7*	*1.1756*	*1.367*	*633.2*	*1.1671*	*1.253*	*633.6*	*1.1592*
110	1.567	643.4	1.1947	1.400	639.4	1.1781	1.261	635.3	1.1621
120	1.612	650.9	1.2077	1.443	647.3	1.1917	1.302	643.5	1.1764
130	1.656	658.1	1.2200	1.485	654.8	1.2045	1.342	651.3	1.1898
140	1.698	665.0	1.2317	1.525	662.0	1.2167	1.380	658.8	1.2025
150	1.740	671.8	1.2429	1.564	669.0	1.2281	1.416	666.1	1.2145
160	1.780	678.4	1.2537	1.601	675.8	1.2394	1.452	673.1	1.2259
170	1.820	684.9	1.2641	1.638	682.5	1.2501	1.487	680.0	1.2369
180	1.859	691.3	1.2742	1.675	689.1	1.2604	1.521	686.7	1.2475
190	1.897	697.7	1.2840	1.710	695.5	1.2704	1.554	693.3	1.2577
200	1.935	703.9	1.2935	1.745	701.9	1.2801	1.587	699.8	1.2677
210	1.972	710.1	1.3029	1.780	708.2	1.2896	1.619	706.2	1.2773
220	2.009	716.3	1.3120	1.814	714.4	1.2989	1.651	712.6	1.2867
230	2.046	722.4	1.3209	1.848	720.6	1.3079	1.683	718.9	1.2959
240	2.082	728.4	1.3296	1.881	726.8	1.3168	1.714	725.1	1.3049
250	2.118	734.5	1.3382	1.914	732.9	1.3255	1.745	731.3	1.3137
260	2.154	740.5	1.3467	1.947	739.0	1.3340	1.775	737.5	1.3224
270	2.189	746.5	1.3550	1.980	745.1	1.3424	1.805	743.6	1.3308
280	2.225	752.5	1.3631	2.012	751.1	1.3507	1.835	749.8	1.3392
290	2.260	758.5	1.3712	2.044	757.2	1.3588	1.865	755.9	1.3474
300	2.295	764.5	1.3791	2.076	763.2	1.3668	1.895	762.0	1.3554
320	2.364	776.5	1.3947	2.140	775.3	1.3825	1.954	774.1	1.3712
340	2.432	788.5	1.4099	2.203	787.4	1.3978	2.012	786.3	1.3866
360	2.500	800.5	1.4247	2.265	799.5	1.4127	2.069	798.4	1.4016
380	2.568	812.5	1.4392	2.327	811.6	1.4273	2.126	810.6	1.4163

Table 16. Properties of Superheated Ammonia Vapor—(*Concluded*)

V = volume in cu. ft. per lb.; H = heat content in B.t.u. per lb.; S = entropy in B.t.u. per lb. per °F.

Temp., °F.	Absolute pressure in lb. per sq. in. (Saturation temperature in italics.)								
	260 *113.42°*			280 *118.45°*			300 *123.21°*		
	V	H	S	V	H	S	V	H	S
Sat.	*1.155*	*633.9*	*1.1518*	*1.072*	*634.0*	*1.1449*	*0.999*	*634.0*	*1.1383*
120	1.182	639.5	1.1617	1.078	635.4	1.1473			
130	1.220	647.8	1.1757	1.115	644.0	1.1621	1.023	640.1	1.1487
140	1.257	655.6	1.1889	1.151	652.2	1.1759	1.058	648.7	1.1632
150	1.292	663.1	1.2014	1.184	660.1	1.1888	1.091	656.9	1.1767
160	1.326	670.4	1.2132	1.217	667.6	1.2011	1.123	664.7	1.1894
170	1.359	677.5	1.2245	1.249	674.9	1.2127	1.153	672.2	1.2014
180	1.391	684.4	1.2354	1.279	681.9	1.2239	1.183	679.5	1.2129
190	1.422	691.1	1.2458	1.309	688.9	1.2346	1.211	686.5	1.2239
200	1.453	697.7	1.2560	1.339	695.6	1.2449	1.239	693.5	1.2344
210	1.484	704.3	1.2658	1.367	702.3	1.2550	1.267	700.3	1.2447
220	1.514	710.7	1.2754	1.396	708.8	1.2647	1.294	706.9	1.2546
230	1.543	717.1	1.2847	1.424	715.3	1.2742	1.320	713.5	1.2642
240	1.572	723.4	1.2938	1.451	721.8	1.2834	1.346	720.0	1.2736
250	1.601	729.7	1.3027	1.478	728.1	1.2924	1.372	726.5	1.2827
260	1.630	736.0	1.3115	1.505	734.4	1.3013	1.397	732.9	1.2917
270	1.658	742.2	1.3200	1.532	740.7	1.3099	1.422	739.2	1.3004
280	1.686	748.4	1.3285	1.558	747.0	1.3184	1.447	745.5	1.3090
290	1.714	754.5	1.3367	1.584	753.2	1.3268	1.472	751.8	1.3175
300	1.741	760.7	1.3449	1.610	759.4	1.3350	1.496	758.1	1.3257
320	1.796	772.9	1.3608	1.661	771.7	1.3511	1.544	770.5	1.3419
340	1.850	785.2	1.3763	1.712	784.0	1.3667	1.592	782.9	1.3576
360	1.904	797.4	1.3914	1.762	796.3	1.3819	1.639	795.3	1.3729
380	1.957	809.6	1.4062	1.811	808.7	1.3967	1.686	807.7	1.3878
400	2.009	821.9	1.4206	1.861	821.0	1.4112	1.732	820.1	1.4024

Table 17. Carbon Dioxide (CO_2), Saturated Vapor*

Temp., °F.	Pressure			Volume		Density		Heat content above 32°F.			Entropy from 32°F.		
	Abs., lb. per sq. in.	Gage, lb. per sq. in.	Gage, atm. per sq. in.	Liquid, cu. ft. per lb.	Vapor, cu. ft. per lb.	Liquid, lb. per cu. ft.	Vapor, lb. per cu. ft.	Liquid, B.t.u. per lb.	Latent, B.t.u. per lb.	Vapor, B.t.u. per lb.	Liquid, B.t.u. per lb. per °F.	Evapor., B.t.u. per lb. per °F.	Vapor, B.t.u. per lb. per °F.
−20	220.6	205.9	14.00	0.0155	0.4166	64.34	2.401	−23.96	126.0	102.0	−0.0514	0.2867	0.2353
−18	228.4	213.7	14.53	.0156	.4018	64.15	2.489	−23.13	125.2	102.1	− .0495	.2837	.2342
−16	236.4	221.7	15.08	.0156	.3876	63.94	2.580	−22.30	124.5	102.2	− .0476	.2807	.2331
−14	244.6	229.9	15.64	.0157	.3739	63.73	2.674	−21.46	123.7	102.2	− .0458	.2777	.2319
−12	253.0	238.3	16.21	.0157	.3608	63.49	2.772	−20.61	122.9	102.3	− .0439	.2746	.2307
−10	261.7	247.0	16.80	.0158	.3482	63.25	2.872	−19.76	122.0	102.3	− .0420	.2716	.2296
− 8	270.6	255.9	17.41	.0159	.3360	63.01	2.976	−18.90	121.2	102.3	− .0401	.2685	.2284
− 6	279.7	265.0	18.03	.0159	.3243	62.76	3.083	−18.04	120.3	102.3	− .0382	.2654	.2273
− 4	289.1	274.4	18.67	.0160	.3131	62.50	3.194	−17.17	119.5	102.3	− .0362	.2623	.2261
− 2	298.7	284.0	19.32	.0161	.3022	62.23	3.309	−16.29	118.6	102.3	− .0343	.2592	.2249
0	308.6	293.9	20.00	.0161	.2918	61.95	3.427	−15.41	117.7	102.2	− .0324	.2561	.2237
2	318.7	304.0	20.68	.0162	.2817	61.65	3.550	−14.51	116.7	102.2	− .0304	.2530	.2225
4	329.1	314.4	21.39	.0163	.2720	61.36	3.676	−13.61	115.8	102.1	− .0285	.2498	.2213
5	334.4	319.7	21.75	.0163	.2673	61.22	3.741	−13.16	115.3	102.1	− .0275	.2482	.2207
6	339.8	325.1	22.11	.0164	.2627	61.07	3.807	−12.71	114.8	102.1	− .0266	.2466	.2201
8	350.7	336.0	22.85	.0165	.2537	60.77	3.942	−11.79	113.8	102.0	− .0246	.2435	.2189
10	361.8	347.1	23.61	.0165	.2450	60.48	4.082	−10.87	112.8	101.9	− .0226	.2402	.2176
12	373.3	358.6	24.39	.0166	.2366	60.18	4.227	− 9.934	111.7	101.8	− .0206	.2370	.2164
16	397.1	382.4	26.01	.0168	.2207	59.58	4.532	− 8.038	109.6	101.5	− .0166	.2305	.2139
25	454.8	440.1	29.94	.0172	.1886	58.14	5.303	− 3.618	104.3	100.7	− .0074	.2154	.2080
35	526.4	511.7	34.81	.0177	.1581	56.41	6.323	1.604	97.77	99.38	.0033	.1978	.2010
45	606.5	591.8	40.26	.0184	.1321	54.41	7.571	7.251	90.21	97.46	.0146	.1788	.1934
55	695.9	681.2	46.34	.0192	.1095	52.05	9.132	13.49	81.29	94.78	.0268	.1580	.1849
65	795.1	780.4	53.09	.0203	.0894	49.14	11.18	20.66	70.35	91.01	.0406	.1342	.1748
75	905.1	890.4	60.57	.0222	.0708	45.05	14.13	29.50	55.83	85.33	.0573	.1045	.1618
85	1027.0	1012.3	68.83	.0258	.0500	38.76	20.00	43.18	31.29	74.47	.0826	.0575	.1401
86	1039.0	1024.3	69.70	.0267	.0474	37.41	21.09	45.45	27.00	72.46	.0868	.0495	.1363
88.43	1071.0	1056.3	71.86	.0346	.0346	28.90	28.90	59.23	0.00	59.23	.1120	.0000	.1120

* Mollier (Amagat), Hodson, "Ice and Cold Storage," London, 1914. Rearranged to conform to A.S.R.E. Standard. [*Am. Soc. Refrig. Eng., Circ.* 9 (1926).]

Table 18. Sulfur Dioxide (SO_2), Saturated Vapor[1]

Temp., °F.	Pressure		Volume		Density		Heat content, above 40°			Entropy, from 40°		
	Abs., lb. per sq. in.	Gage, lb. per sq. in.	Liquid, cu. ft. per lb.	Vapor, cu. ft. per lb.	Liquid, lb. per cu. ft.	Vapor, lb. per cu. ft.	Liquid, B.t.u. per lb.	Latent,† B.t.u. per lb.	Vapor, B.t.u. per lb.	Liquid, B.t.u. per lb. per °F.	Evapor., B.t.u. per lb. per °F.	Vapor, B.t.u. per lb. per °F.
−40	3.136	23.54*	0.010440	22.42	95.79	0.04460	0.00	178.61	178.61	0.00000	0.42562	0.42562
−35	3.693	22.41*	.010486	19.23	95.36	0.05200	1.45	177.82	179.27	.00334	.41875	.42209
−30	4.331	21.10*	.010532	16.56	94.94	0.06039	2.93	176.97	179.90	.00674	.41190	.41864
−25	5.058	19.63*	.010580	14.31	94.52	0.06988	4.44	176.06	180.50	.01016	.40509	.41525
−20	5.883	17.93*	.010627	12.42	94.10	0.08119	5.98	175.09	181.07	.01366	.39826	.41192
−15	6.814	16.05*	.010674	10.81	93.68	0.09250	7.56	174.06	181.62	.01719	.39146	.40865
−10	7.863	13.91*	.010721	9.44	93.27	0.1025	9.16	172.97	182.13	.02075	.38469	.40544
− 5	9.038	11.52*	.010770	8.28	92.85	0.1208	10.79	171.83	182.62	.02443	.37795	.40228
0	10.35	8.85*	.010820	7.280	92.42	0.1374	12.44	170.63	183.07	.02795	.37122	.39917
2	10.91	7.34*	.010840	6.923	92.25	0.1444	13.12	170.13	183.25	.02941	.36853	.39794
4	11.50	6.50*	.010860	6.584	92.08	0.1501	13.78	169.63	183.41	.03084	.36586	.39670
‡5	11.81	5.87*	.010870	6.421	92.00	0.1558	14.11	169.38	183.49	.03155	.36454	.39609
6	12.12	5.24*	.010880	6.266	91.91	0.1596	14.45	169.12	183.57	.03228	.36319	.39547
8	12.75	3.96*	.010900	5.967	91.74	0.1676	15.13	168.60	183.73	.03373	.36053	.39426
10	13.42	2.59*	.010920	5.682	91.58	0.1760	15.80	168.07	183.87	.03519	.35787	.39306
12	14.12	1.17*	.010940	5.417	91.41	0.1846	16.48	167.53	184.01	.03664	.35521	.39185
14	14.84	0.14	.010960	5.164	91.24	0.1936	17.15	166.97	184.14	.03808	.35257	.39065
16	15.59	0.89	.010981	4.926	91.07	0.2030	17.84	166.44	184.28	.03953	.34993	.38946
18	16.37	1.67	.011003	4.701	90.89	0.2127	18.52	165.88	184.40	.04098	.34729	.38827
20	17.18	2.48	.011025	4.487	90.71	0.2228	19.20	165.32	184.52	.04241	.34466	.38707
22	18.03	3.33	.011047	4.287	90.53	0.2332	19.90	164.74	184.64	.04385	.34204	.38589
24	18.89	4.19	.011070	4.096	90.33	0.2441	20.58	164.16	184.74	.04528	.33943	.38471
26	19.80	5.10	.011093	3.915	90.15	0.2559	21.26	163.58	184.84	.04671	.33683	.38354
28	20.73	6.03	.011116	3.744	89.96	0.2671	21.96	162.98	184.94	.04814	.33422	.38237
30	21.70	7.00	.011140	3.581	89.76	0.2800	22.64	162.38	185.02	.04956	.33163	.38119
32	22.71	8.01	.011164	3.437	89.58	0.2909	23.33	161.77	185.10	.05099	.32904	.38003
34	23.75	9.05	.011188	3.283	89.39	0.3046	24.03	161.15	185.18	.05242	.32645	.37887
36	24.82	10.12	.011212	3.144	89.18	0.3181	24.72	160.53	185.25	.05384	.32388	.37772
38	25.95	11.25	.011236	3.013	89.00	0.3319	25.42	159.89	185.31	.05527	.32130	.37657
40	27.10	12.40	.011260	2.887	88.81	0.3464	26.12	159.25	185.37	.05668	.31873	.37541
42	28.29	13.59	.011284	2.769	88.62	0.3611	26.81	158.61	185.42	.05809	.31616	.37425
44	29.52	14.82	.011308	2.656	88.43	0.3765	27.51	157.95	185.46	.05949	.31362	.37311
46	30.79	16.09	.011332	2.548	88.24	0.3925	28.21	157.29	185.50	.06090	.31107	.37197

48	32.10	17.40	.011356	2.446	88.05	0.4088	28.92	156.62	185.54	.06231	.30852	.37083
50	33.45	18.75	.011380	2.348	87.87	0.4259	29.61	155.95	185.56	.06370	.30599	.36969
52	34.86	20.16	.011404	2.256	87.67	0.4433	30.31	155.27	185.58	.06509	.30348	.36857
54	36.31	21.61	.011428	2.167	87.51	0.4615	31.00	154.59	185.59	.06646	.30097	.36743
56	37.80	23.10	.011452	2.083	87.31	0.4801	31.72	153.89	185.61	.06785	.29844	.36629
58	39.33	24.63	.011476	2.003	87.13	0.4992	32.42	153.19	185.61	.06923	.29594	.36517
60	40.93	26.23	.011500	1.926	86.95	0.5194	33.10	152.49	185.59	.07060	.29345	.36405
62	42.58	27.88	.011524	1.853	86.77	0.5396	33.79	151.78	185.57	.07196	.29097	.36293
64	44.27	29.57	.011548	1.783	86.59	0.5609	34.49	151.06	185.55	.07333	.28848	.36181
66	46.00	31.30	.011572	1.716	86.41	0.5827	35.19	150.34	185.53	.07469	.28601	.36070
68	47.78	33.08	.011598	1.652	86.22	0.6054	35.88	149.62	185.50	.07602	.28356	.35958
70	49.62	34.92	.011626	1.590	86.02	0.6290	36.58	148.88	185.46	.07736	.28110	.35846
72	51.54	36.84	.011652	1.532	85.82	0.6527	37.28	148.14	185.42	.07871	.27865	.35736
74	53.48	38.78	.011680	1.476	85.62	0.6777	37.97	147.40	185.37	.08003	.27621	.35624
76	55.48	40.78	.011706	1.422	85.42	0.7030	38.67	146.64	185.31	.08135	.27377	.35512
78	57.56	42.86	.011732	1.371	85.23	0.7295	39.36	145.88	185.24	.08268	.27133	.35401
80	59.68	44.98	.011760	1.321	85.03	0.7570	40.05	145.12	185.17	.08399	.26897	.35291
82	61.88	47.18	.011786	1.274	84.84	0.7850	40.73	144.36	185.09,	.08525	.26652	.35177
84	64.14	49.44	.011814	1.229	84.64	0.8140	41.43	143.58	185.01	.08653	.26412	.35065
‡86	66.45	51.75	.011841	1.185	84.44	0.8440	42.12	142.80	184.92	.08783	.26171	.34954
88	68.84	54.14	.011868	1.144	84.25	0.8740	42.80	142.02	184.82	.08910	.25933	.34843
90	71.25	56.55	.011896	1.104	84.05	0.9058	43.50	141.22	184.72	.09038	.25693	.34731
92	73.70	59.00	.011923	1.065	83.86	0.9390	44.19	140.42	184.61	.09165	.25455	.34620
94	76.30	61.60	.011951	1.028	83.67	0.9730	44.86	139.62	184.49	.09389	.25219	.34508
96	79.03	64.33	.011979	0.9931	83.47	1.007	45.54	138.83	184.37	.09411	.24986	.34397
98	81.77	67.07	.012008	0.9591	83.27	1.043	46.22	138.03	184.25	.09532	.24753	.34285
100	84.52	69.82	.012037	0.9262	83.07	1.080	46.90	137.20	184.10	.09657	.24516	.34173
105	91.85	77.15	.012110	0.8498	82.57	1.176	48.88	135.14	183.72	.09958	.23934	.33892
110	99.76	85.06	.012190	0.7804	82.03	1.281	50.26	133.05	183.31	.10254	.23357	.33611
115	108.02	93.32	.012275	0.7174	81.46	1.394	51.93	130.92	182.85	.10546	.22783	.33329
120	120.93	106.23	.012360	0.6598	80.90	1.515	53.58	128.78	182.36	.10829	.22217	.33046
125	126.43	111.73	.012445	0.6079	80.35	1.645	55.31	126.51	181.82	.11120	.21639	.32759
130	136.48	121.78	.012530	0.5595	79.81	1.787	56.85	124.39	181.24	.11376	.21096	.32472
135	147.21	132.51	.012620	0.5158	79.23	1.947	58.47	122.15	180.62	.11640	.20542	.32182
140	158.61	143.91	.012720	0.4758	78.61	2.102	60.04	119.90	179.94	.11893	.19990	.31888

---

[1] David L. Fiske, Urbana, Ill., 1925.

* In. Hg below 1 atm. (29.92 in.) 14.696 lb. abs.

† For internal latent heat see *Refrigerating Eng.*, vol. 2, No. 6, p. 235 (Dec., 1924).

‡ Standard ton temperatures.

Table 19. Sulfur Dioxide (SO_2), Superheated Vapor*

Temp., °F.	Abs. pressure 4 lb. per sq. in. Gage pressure, 21.7 in. vac. (Sat'n. temp., 32.60°F.)			Abs. pressure 6 lb. per sq. in. Gage pressure, 17.7 in. vac. (Sat'n. temp., 19.37°F.)			Abs. pressure 8 lb. per sq. in. Gage pressure, 13.6 in. vac. (Sat'n. temp., 8.99°F.)			Abs. pressure 10 lb. per sq. in. Gage pressure, 9.6 in. vac. (Sat'n. temp., 1 34°F.)		
	Volume, cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F	Volume, cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Volume, cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Volume, cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.
(At sat'n)	*(17.30)*	*(179.57)*	*(0.42043)*	*(12.22)*	*(181.14)*	*(0.41151)*	*(9.220)*	*(182.23)*	*(0.40482)*	*(7.520)*	*(182.95)*	*(0.40000)*
−20	18.40	181.5	0.42487									
−10	18.83	183.0	.42836									
0	19.27	184.6	.43179	12.75	184.3	0.41850	9.516	183.7	0.40871	7.545	183.2	0.40046
10	19.70	186.1	.43516	13.04	185.9	.42198	9.751	185.4	.41230	7.744	185.0	.40432
20	20.14	187.7	.43847	13.34	187.5	.42538	9.983	187.1	.41579	7.939	186.7	.40802
30	20.57	189.3	.44161	13.63	189.1	.42869	10.21	188.8	.41922	8.030	188.4	.41159
40	21.00	190.9	.44491	13.93	190.7	.43196	10.44	190.5	.42256	8.316	190.1	.41505
50	21.42	192.5	.44806	14.23	192.3	.43517	10.66	192.2	.42582	8.500	191.8	.41837
60	21.85	194.1	.45116	14.52	193.9	.43833	10.88	193.8	.42903	8.681	193.5	.42161
70	22.27	196.7	.45421	14.71	195.6	.44140	11.10	195.5	.43216	8.860	195.2	.42480
80	22.70	197.3	.45722	15.11	197.2	.44443	11.32	197.1	.43524	9.038	196.9	.42795
90	23.12	198.9	.46018	15.40	199.9	.44741	11.54	198.8	.43825	9.214	198.6	.43104
100	23.54	200.5	.46311	15.69	200.5	.45035	11.75	200.4	.44123	9.389	200.3	.43407
110	23.96	202.1	.46600	15.97	202.2	.45326	11.97	202.1	.44416	9.563	202.0	.43705
120	24.39	203.8	.46885	16.26	203.8	.45613	12.18	203.7	.44705	9.736	203.7	.43997
130	24.81	205.2	.47167	16.54	205.3	.45890	12.39	205.4	.44990	9.908	205.4	.44283
140	25.23	207.1	.47445	16.82	207.1	.46176	12.61	207.0	.45271	10.08	207.1	.44565
150	25.65	208.8	.47720	17.09	208.8	.46451	12.82	208.8	.45543	10.25	208.8	.44842
160	26.08	210.4	.47991	17.35	210.4	.46722	13.03	210.3	.45820	10.42	210.5	.45116
170	26.50	212.1	.48259	17.62	212.1	.46990	13.24	212.0	.46089	10.59	212.2	.45296
180	26.92	213.8	.48523	17.88	213.7	.47254	13.46	213.6	.46353	10.76	213.8	.45651
190	......	......	....	18.13	215.4	.47514	13.66	215.3	.46614	10.93	215.4	.45913
200	......	......	....	18.38	217.0	.47769	13.88	216.9	.46871	11.10	217.0	.46171

* David L. Fiske, Urbana, Ill., 1925.

Table 19. **Sulfur Dioxide** (SO_2), **Superheated Vapor**—(*Continued*)

Temp., °F.	Abs. pressure, 15 lb. per sq. in. Gage pressure, 0.30 lb. per sq. in. (Sat'n. temp., 14.43°F.)			Abs. pressure, 20 lb. per sq. in. Gage pressure, 5.30 lb. per sq. in. (Sat'n. temp., 26.44°F.)			Abs. pressure, 25 lb. per sq. in. Gage pressure, 10.30 lb. per sq. in. (Sat'n. temp., 36.33°F.)			Abs. pressure, 30 lb. per sq. in. Gage pressure, 15.30 lb. per sq. in. (Sat'n. temp., 44.76°F.)		
(*At sat'n*)	(*5.110*)	(*184.17*)	(*0.39091*)	(*3.878*)	(*184.86*)	(*0.38339*)	(*3.123*)	(*185.26*)	(*0.37754*)	(*2.614*)	(*185.48*)	(*0.37269*)
20	5.192	185.4	0.39270									
30	5.333	187.3	.39672									
40	5.470	189.2	.40054	4.035	187.8	0.38959	3.181	186.1	0.37927			
50	5.604	191.0	.40424	4.145	189.8	.39346	3.273	188.4	.38372			
60	5.734	192.8	.40777	4.251	191.8	.39719	3.363	190.6	.38795	2.747	189.3	0.37969
70	5.862	195.6	.41116	4.354	193.7	.40080	3.451	192.7	.39198	2.830	191.6	.38428
80	5.988	196.4	.41443	4.454	195.6	.40429	3.356	194.7	.39582	2.907	193.8	.38848
90	6.112	198.2	.41765	4.552	197.5	.40758	3.618	196.7	.39945	2.980	195.9	.39236
100	6.233	199.9	.42076	4.648	199.3	.41093	3.696	198.6	.40291	3.052	197.9	.39603
110	6.353	201.6	.42383	4.742	201.1	.41415	3.772	200.5	.40625	3.122	199.9	.39955
120	6.471	203.3	.42682	4.834	202.9	.41726	3.848	202.4	.40949	3.189	201.8	.40293
130	6.588	205.6	.42976	4.925	204.7	.42027	3.923	204.2	.41261	3.254	203.7	.40619
140	6.705	206.7	.43264	5.015	206.5	.42322	3.998	206.0	.41568	3.318	205.6	.40935
150	6.821	208.4	.43548	5.104	208.2	.42613	4.073	207.8	.41866	3.381	207.5	.41241
160	6.937	210.1	.43825	5.193	209.9	.42898	4.145	209.6	.42156	3.443	209.3	.41539
170	7.052	211.8	.44097	5.281	211.6	.43176	4.216	211.4	.42439	3.504	211.1	.41829
180	7.167	213.5	.44366	5.369	213.3	.43449	4.287	213.2	.42717	3.565	212.9	.42112
190	7.282	215.2	.44630	5.456	215.0	.43716	4.358	215.0	.42988	3.625	214.7	.42387
200	7.396	216.9	.44889	5.542	216.7	.43977	4.428	216.7	.43253	3.685	216.5	.42657
210	....	....	....	5.629	218.4	.44234	4.498	218.4	.43413	3.744	218.3	.42921
220	....	....	....	5.715	220.1	.44488	4.567	220.1	.43769	3.803	220.1	.43180
230	....	....	....	....	....	....	4.637	221.8	.44023	3.861	221.9	.43438
240	....	....	....	....	....	....	4.706	223.5	.44275	3.919	223.6	.43691
250	....	....	....	....	....	....	....	....	....	3.977	225.3	.43942
260	....	....	....	....	....	....	....	....	....	4.035	227.0	.44188

Table 19. Sulfur Dioxide (SO_2), Superheated Vapor—(*Continued*)

Temp., °F.	Abs. pressure, 40 lb. per sq. in. Gage pressure, 25.30 lb. per sq. in. (Sat'n. temp. 58.83°F.)			Abs. pressure, 50 lb. per sq. in. Gage pressure, 35.30 lb. per sq. in. (Sat'n. temp., 70.40°F.)			Abs. pressure, 60 lb. per sq. in. Gage pressure, 45.30 lb. per sq. in. (Sat'n. temp., 80.29°F.)			Abs. pressure, 70 lb. per sq. in. Gage pressure, 55.30 lb. per sq. in. (Sat'n. temp., 88.97°F.)		
	Volume, cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Volume, cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Volume, cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Volume, cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.
(*At sat'n*)	(*1.970*)	(*185.60*)	(*0.36470*)	(*1.577*)	(*185.45*)	(*0.35826*)	(*1.3144*)	(*185.16*)	(*0.35272*)	(*1.125*)	(*184.77*)	(*0.34789*)
60	1.980	185.9	0.36544									
70	2.064	188.7	.37064									
80	2.121	191.3	.37544	1.668	188.4	0.36366						
90	2.185	193.6	.37992	1.723	191.2	.36887						
100	2.246	196.1	.38415	1.775	193.9	.37369	1.288	191.4	0.36403	1.181	187.6	0.35443
110	2.304	198.3	.38810	1.825	196.4	.37815	1.346	194.3	.36906	1.228	191.6	.36020
120	2.360	200.4	.39183	1.872	198.8	.38234	1.403	197.0	.37375	1.272	194.8	.36545
130	2.413	202.5	.39541	1.917	201.1	.38627	1.459	199.5	.37810	1.313	197.6	.37028
140	2.465	204.6	.39881	1.961	203.3	.38998	1.514	201.9	.38217	1.352	200.3	.37478
150	2.515	206.5	.40209	2.003	205.4	.39353	1.563	204.2	.38603	1.389	202.9	.37897
160	2.565	208.5	.40525	2.044	207.5	.39691	1.608	206.5	.38963	1.424	205.3	.38291
170	2.614	210.4	.40831	2.084	209.6	.40015	1.650	208.6	.39310	1.457	207.6	.38662
180	2.662	212.3	.41127	2.123	211.6	.40327	1.689	210.7	.39639	1.489	209.9	.39014
190	2.709	214.2	.41416	2.161	213.4	.40628	1.726	212.8	.39956	1.521	212.0	.39348
200	2.755	216.0	.41694	2.199	215.4	.40919	1.751	214.8	.40260	1.551	214.1	.39670
210	2.800	217.9	.41966	2.237	217.3	.41200	1.785	216.8	.40554	1.580	216.1	.39978
220	2.845	219.7	.42233	2.274	219.2	.41477	1.819	218.7	.40839	1.608	218.1	.40275
230	2.889	221.5	.42494	2.311	221.1	.41748	1.853	220.7	.41118	1.636	220.1	.40564
240	2.933	223.3	.42751	2.347	223.0	.42015	1.885	222.6	.41391	1.664	222.1	.40845
250	2.977	225.1	.43007	2.383	224.9	.42275	1.917	224.5	.41657	1.691	224.1	.41120
260	3.021	227.0	.43262	2.418	226.7	.42535	1.948	226.4	.41917	1.718	226.0	.41389
270	....	....	....	2.454	228.5	.42791	1.979	228.2	.42175	1.745	227.9	.41653
280	....	....	....	2.489	230.3	.43045	2.010	230.1	.42431	1.771	229.8	.41912
290	....	....	....	....	....	....	2.040	232.0	.42685	1.798	231.7	.42167
300	....	....	....	....	....	....	2.070	233.8	.42935	1.824	233.5	.42418

Table 19. Sulfur Dioxide (SO_2), Superheated Vapor—(*Concluded*)

Temp., °F.	Abs. pressure, 80 lb. per sq. in. Gage pressure, 65.30 lb. per sq. in. (Sat'n. temp. 96.88°F.)			Abs. pressure, 100 lb. per sq. in. Gage pressure, 85.30 lb. per sq. in. (Sat'n. temp., 110.15°F.)			Abs. pressure, 120 lb. per sq. in. Gage pressure, 105.30 lb. per sq. in. (Sat'n. temp., 121.52°F.)			Abs. pressure, 140 lb. per sq. in. Gage pressure, 125.30 lb. per sq. in. (Sat'n. temp., 131.64°F.)		
(*At sat'n*)	(*0.9809*)	(*184.33*)	(*0.34357*)	(*0.7786*)	(*183.30*)	(*0.33603*)	(*0.6430*)	(*182.19*)	(*0.33954*)	(*0.5451*)	(*181.04*)	(*0.32388*)
100	0.993	185.6	0.34571									
110	1.040	189.1	.35214									
120	1.084	192.5	.35797	0.8190	187.3	0.34296						
130	1.125	195.7	.36330	0.8575	191.0	.34942						
140	1.163	198.6	.36819	0.8928	194.6	.35528	0.7085	190.1	0.34264	0.5734	185.1	0.33089
150	1.199	201.3	.37270	0.9255	197.9	.36061	0.7403	193.9	.34904	.6055	189.7	.33777
160	1.232	203.9	.37692	0.9561	200.9	.36558	0.7700	197.4	.35484	.6345	193.6	.34442
170	1.263	206.4	.38093	0.9848	203.7	.37009	0.7972	200.6	.36012	.6613	196.3	.35041
180	1.292	208.7	.38461	1.012	206.4	.37431	0.8228	203.7	.36494	.6861	200.8	.35588
190	1.320	211.0	.38813	1.038	209.0	.37829	0.8470	206.7	.36936	.7092	204.0	.36088
200	1.347	213.3	.39150	1.062	211.5	.38203	0.8699	209.4	.37348	.7309	207.1	.36548
210	1.374	215.5	.39471	1.086	213.8	.38556	0.8916	212.0	.37737	.7513	210.0	.36976
220	1.400	217.5	.39780	1.109	216.1	.38892	0.9124	214.5	.38104	.7707	212.7	.37379
230	1.426	219.6	.40079	1.131	218.4	.39214	0.9324	217.0	.38451	.7892	215.4	.37758
240	1.451	221.6	.40369	1.152	220.5	.39524	0.9515	219.3	.38785	.8070	217.9	.38118
250	1.476	223.6	.40651	1.173	222.6	.39824	0.9700	221.5	.39106	.8241	220.3	.38461
260	1.500	225.6	.40926	1.194	224.7	.40114	0.9880	223.7	.39416	.8405	222.6	.38789
270	1.524	227.6	.41195	1.213	226.8	.40397	1.006	225.9	.39713	.8564	224.9	.39105
280	1.547	229.5	.41459	1.232	228.8	.40673	1.023	228.0	.40002	.8720	227.1	.39408
290	1.570	231.5	.41719	1.251	230.8	.40944	1.040	230.1	.40284	.8970	229.3	.39701
300	1.593	233.4	.41974	1.268	232.8	.41207	1.056	233.2	.40558	.9017	231.5	.39985
310	....	....	....	1.284	234.8	.41464	1.072	234.3	.40825	.9161	233.6	.40261
320	....	....	....	1.299	236.7	.41716	1.088	236.3	.41085	.9302	235.7	.40529
330	....	....	....	....	....	....	1.104	238.3	.41338	.9441	237.7	.40791
340	....	....	....	....	....	....	1.120	240.3	.41583	.9579	239.7	.41049

Table 20. Methyl Chloride (CH_3Cl), Saturated Vapor†

Temp., °F.	Pressure		Volume		Density		Heat content from −40°			Entropy from −40°	
	Abs., lb. per sq. in.	Gage, lb. per sq. in.	Liquid, cu. ft. per lb.	Vapor, cu. ft. per lb.	Liquid, lb. per cu. ft.	Vapor, lb. per cu. ft.	Liquid, B.t.u. per lb.	Latent, B.t.u. per lb.	Vapor, B.t.u. per lb.	Liquid, B.t.u. per lb. per °F.	Vapor, B.t.u. per lb. per °F.
t	p	p_d	v_f	v_g	$1/v_f$	$1/v_g$	h_f	h	h_g	s_f	s_g
−40	6.878	15.92*	0.01553	12.72	64.39	0.07861	0.000	190.66	190.66	0.0000	0.4544
−30	9.036	11.52*	.01568	9.873	63.78	0.1013	3.562	188.52	192.08	.0084	.4472
−20	11.71	6.090*	.01583	7.761	63.17	0.1289	7.146	186.34	193.49	.0166	.4405
−10	14.96	0.266	.01598	6.176	62.58	0.1619	10.75	184.11	194.87	.0247	.4343
0	18.90	4.201	.01613	4.969	62.00	0.2013	14.39	181.85	196.23	.0327	.4284
5	21.15	6.455	.01622	4.471	61.65	0.2237	16.21	180.70	196.92	.0367	.4257
10	23.60	8.903	.01631	4.038	61.31	0.2477	18.04	179.53	197.58	.0406	.4229
20	29.16	14.46	.01647	3.312	60.72	0.3019	21.73	177.11	198.84	.0484	.4177
30	35.68	20.98	.01665	2.739	60.06	0.3650	25.44	174.59	200.03	.0560	.4126
40	43.25	28.56	.01684	2.286	59.38	0.4375	29.17	172.00	201.17	.0636	.4079
50	51.99	37.29	.01704	1.920	58.69	0.5208	32.93	169.35	202.28	.0710	.4034
60	62.00	47.30	.01724	1.624	58.00	0.6158	36.71	166.62	203.33	.0784	.3991
70	73.41	58.71	.01744	1.382	57.34	0.7234	40.52	163.82	204.34	.0856	.3950
80	86.26	71.56	.01764	1.183	56.69	0.8451	44.36	160.91	205.27	.0928	.3910
86	94.70	80.00	.01778	1.081	56.24	0.9253	46.67	159.13	205.80	.0970	.3887
90	100.6	85.95	.01786	1.018	55.99	0.9819	48.21	157.92	206.13	.0998	.3872
100	116.7	102.0	.01808	0.8814	55.31	1.135	52.09	154.85	206.94	.1069	.3836
110	134.5	119.8	.01833	0.7672	54.55	1.303	56.00	151.70	207.70	.1138	.3801
120	154.2	139.5	.01859	0.6710	53.79	1.490	59.93	148.46	208.39	.1206	.3768
130	175.9	161.1	.01887	0.5889	52.99	1.698	63.89	145.13	209.02	.1274	.3736
140	199.6	184.9	.01915	0.5189	52.22	1.927	67.87	141.71	209.58	.1341	.3705
150	225.4	210.7	.01945	0.4586	51.41	2.181	71.87	138.23	210.10	.1407	.3674
160	253.5	238.8	.01978	0.4070	50.56	2.457	75.90	134.66	210.56	.1473	.3646
170	283.9	269.2	.02015	0.3613	49.63	2.768	79.97	130.96	210.93	.1538	.3618

---

* In. Hg below 1 atm.

† Tanner, Banning, and Matthewson, *Ind. Eng. Chem.*, **31**, 878 (1939). Copyright, 1939, E. I. du Pont de Nemours & Co., Inc.

Table 21. Methyl Chloride, Superheated Vapor*

Temp., °F.	Abs. pressure, 6 lb. per sq. in., gage pressure, 17.7 in. vacuum (sat'n. temp., −44.8°F.)			Temp., °F.	Abs. pressure, 20 lb. per sq. in., gage pressure, 5.3 lb. per sq. in. (sat'n. temp., 2.5°F.)			Temp., °F.	Abs. pressure, 100 lb. per sq. in., gage pressure, 85.3 lb. per sq. in. (sat'n. temp., 89.6°F.)		
t	*V*	*H*	*S*	*t*	*V*	*H*	*S*	*t*	*V*	*H*	*S*
(At sat'n.)	*(14.45)*	*(189.96)*	*(0.4580)*	*(At sat'n.)*	*(4.710)*	*(196.58)*	*(0.4270)*	*(At sat'n.)*	*(1.025)*	*(206.11)*	*(0.3872)*
−40	14.62	190.77	0.4599	10	4.801	197.95	0.4300	90	1.026	206.21	0.3877
−30	14.99	192.52	.4640	20	4.917	199.90	.4341	100	1.055	208.58	.3920
−20	15.36	194.27	.4681	30	5.032	201.82	.4380	110	1.083	210.96	.3962
−10	15.72	196.06	.4721	40	5.146	203.75	.4420	120	1.111	213.33	.4003
0	16.09	197.84	.4760	50	5.260	205.71	.4458	130	1.138	215.74	.4044
10	16.45	199.66	.4799	60	5.373	207.66	.4496	140	1.165	218.15	.4084
20	16.82	201.48	.4838	70	5.486	209.66	.4534	150	1.191	220.55	.4124
30	17.18	203.34	.4876	80	5.599	211.65	.4572	160	1.217	222.94	.4163
40	17.55	205.19	.4914	90	5.711	213.67	.4608	170	1.243	225.33	.4201
50	17.91	207.10	.4952	100	5.823	215.69	.4645	180	1.268	227.71	.4239
60	18.27	209.01	.4989	110	5.935	217.75	.4681	190	1.293	230.10	.4276
70	18.63	210.95	.5025	120	6.046	219.80	.4717	200	1.318	232.50	.4312
80	18.99	212.88	.5061	130	6.157	221.90	.4753	210	1.343	234.91	.4349
90	19.35	214.85	.5097	140	6.268	223.99	.4788	220	1.367	237.32	.4384
100	19.71	216.82	.5133	150	6.379	226.12	.4823	230	1.391	239.76	.4420
110	20.07	218.83	.5169	160	6.489	228.24	.4858	240	1.415	242.20	.4455
120	20.42	220.84	.5204	170	6.599	230.40	.4892	250	1.439	244.63	.4489
130	20.78	222.89	.5239	180	6.709	232.56	.4927	260	1.463	247.06	.4523
140	21.14	224.94	.5274	190	6.819	234.75	.4961	270	1.487	249.51	.4557
150	21.50	227.03	.5308	200	6.929	236.94	.4994	280	1.511	251.96	.4591
160	21.86	229.11	.5342	210	7.038	239.17	.5028	290	1.534	254.44	.4624
170	22.21	231.24	.5376	220	7.147	241.40	.5061	300	1.557	256.92	.4657
180	22.57	233.36	.5410	230	7.256	243.68	.5094	310	1.580	259.43	.4689
190	22.93	235.52	.5443	240	7.365	245.96	.5127	320	1.603	261.93	.4722
200	23.29	237.69	.5476	250	7.474	248.26	.5159	330	1.626	264.45	.4754
210	23.64	239.89	.5510	260	7.583	250.55	.5192	340	1.649	266.97	.4786
220	24.00	242.09	.5542	270	7.692	252.89	.5224	350	1.672	269.52	.4817
230	24.35	244.34	.5575	280	7.801	255.23	.5256	360	1.695	272.07	.4849
240	24.71	246.60	.5607	290	7.910	257.60	.5287	370	1.717	274.66	.4880
250	25.06	248.88	.5640	300	8.019	259.96	.5319	380	1.739	277.25	.4911
260	25.42	251.15	.5672	310	8.128	262.36	.5350	390	1.761	279.87	.4942

---

* Tanner, Banning, and Matthewson, *Ind. Eng. Chem.*, **31**, 878 (1939). Copyright, 1939, E. I. duPont de Nemours & Co., Inc.

Table 21. Methyl Chloride, Superheated Vapor—(*Concluded*)

Temp., °F.	Abs. pressure, 10 lb. per sq. in., gage pressure, 9.6 in. vacuum (sat'n. temp., −26.1°F.)			Temp., °F.	Abs. pressure, 50 lb. per sq. in., gage pressure, 35.3 lb. per sq. in. (sat'n. temp., 47.8°F.)			Temp., °F.	Abs. pressure, 200 lb. per sq. in., gage pressure, 185.3 lb. per sq. in. (sat'n. temp., 140.3°F.)		
t	*V*	*H*	*S*	*t*	*V*	*H*	*S*	*t*	*V*	*H*	*S*
(*At sat'n.*)	(*8.993*)	(*192.64*)	(*0.4446*)	(*At sat'n.*)	(*1.992*)	(*202.09*)	(*0.4048*)	(*At sat'n.*)	(*0.517*)	(*209.60*)	(*0.3708*)
−20	9.124	193.67	0.4471	50	2.003	202.55	0.4053	150	0.533	212.41	0.3749
−10	9.346	195.50	.4512	60	2.054	204.65	.4094	160	.551	215.30	.3796
0	9.567	197.32	.4552	70	2.104	206.77	.4134	170	.566	218.09	.3842
10	9.788	199.18	.4591	80	2.154	208.89	.4174	180	.582	220.87	.3886
20	10.01	201.04	.4630	90	2.203	211.03	.4213	190	.597	223.61	.3929
30	10.23	202.91	.4669	100	2.252	213.18	.4252	200	.612	226.35	.3971
40	10.45	204.78	.4707	110	2.300	215.35	.4290	210	.626	229.05	.4012
50	10.67	206.70	.4745	120	2.348	217.52	.4328	220	.641	231.75	.4052
60	10.89	208.62	.4782	130	2.396	219.70	.4366	230	.654	234.43	.4091
70	11.11	210.58	.4819	140	2.443	221.88	.4402	240	.668	237.12	.4129
80	11.33	212.53	.4856	150	2.490	224.10	.4439	250	.682	239.75	.4167
90	11.55	214.51	.4892	160	2.537	226.32	.4475	260	.695	242.39	.4204
100	11.77	216.50	.4928	170	2.584	228.55	.4511	270	.708	245.02	.4240
110	11.99	218.52	.4964	180	2.630	230.79	.4546	280	.721	247.65	.4276
120	12.21	220.54	.5000	190	2.676	233.05	.4581	290	.734	250.29	.4311
130	12.43	222.61	.5035	200	2.722	235.32	.4616	300	.747	252.93	.4346
140	12.65	224.67	.5069	210	2.768	237.61	.4650	310	.760	255.58	.4380
150	12.86	226.77	.5104	220	2.813	239.90	.4684	320	.772	258.21	.4414
160	13.08	228.86	.5138	230	2.858	242.24	.4718	330	.785	260.86	.4448
170	13.30	231.00	.5172	240	2.903	244.58	.4752	340	.797	263.51	.4481
180	13.52	233.13	.5206	250	2.948	246.92	.4785	350	.810	266.17	.4515
190	13.74	235.30	.5240	260	2.993	249.27	.4818	360	.822	268.84	.4547
200	13.95	237.47	.5273	270	3.038	251.65	.4851	370	.834	271.51	.4580
210	14.17	239.68	.5306	280	3.083	254.02	.4884	380	.847	274.19	.4612
220	14.38	241.89	.5339	290	3.128	256.43	.4916	390	.859	276.88	.4644
230	14.60	244.15	.5372	300	3.173	258.83	.4948	400	.870	279.59	.4676
240	14.81	246.42	.5045	310	3.217	261.27	.4980	410	.883	282.31	.4707
250	15.03	248.70	.5437	320	3.261	263.71	.5011	420	.895	285.05	.4738
260	15.24	250.98	.5469	330	3.305	266.18	.5043	430	.906	287.79	.4769
270	15.46	253.32	.5501	340	3.349	268.65	.5074	440	.918	290.52	.4800
280	15.67	255.66	.5532	350	3.393	271.17	.5105				

Table 21. Ethyl Chloride (C_2H_5Cl)†

Temp., °F.	Pressure		Volume		Density		Heat content above 32°F.			Entropy from 32°F,		
	Abs., lb. per sq. in.	Gage, lb. per sq. in.	Liquid, cu. ft. per lb.	Vapor, cu. ft. per lb.	Liquid, lb. per cu. ft.	Vapor, lb. per cu. ft.	Liquid, B.t.u. per lb.	Latent, B.t.u. per lb.	Vapor, B.t.u. per lb.	Liquid, B.t.u. per lb. per °F.	Evapor., B.t.u. per lb. per °F.	Vapor, B.t.u. per lb. per °F.
−22	2.20	−12.5	0.01657	34.4	60.35	0.0291	−23.1	181.3	158.2	−0.0497	0.4139	0.3642
−13	2.85	−11.85	.01669	26.95	59.92	.0371	−19.2	179.9	160.7	− .0410	.4025	.3615
− 4	3.66	−11.04	.01682	21.33	59.45	.0469	−15.4	178.5	163.1	− .0324	.3915	.3591
+ 5	4.65	−10.05	.01695	17.06	59.00	.0586	−11.6	177.0	165.4	− .0241	.3807	.3566
14	5.85	− 8.85	.01708	13.77	58.55	.0726	− 7.7	175.5	167.8	− .0159	.3702	.3543
23	7.28	− 7.42	.01721	11.21	58.10	.0892	− 3.8	174.0	170.2	− .0079	.3602	.3523
32	8.99	− 5.71	.01735	9.21	57.64	.1086	0.0	172.5	172.5	.0000	.3506	.3506
41	11.01	− 3.69	.01749	7.62	57.18	.1311	+ 3.8	170.9	174.7	+ .0077	.3411	.3488
50	13.37	− 1.33	.01763	6.36	56.72	.1573	7.7	169.3	177.0	.0154	.3320	.3475
59	16.11	+ 1.41	.01777	5.34	56.27	.1873	11.6	167.7	179.3	.0228	.3231	.3459
68	19.29	4.50	.01792	4.51	55.80	.2215	15.4	166.0	181.4	.0302	.3144	.3446
77	22.94	8.24	.01807	3.84	55.34	.2604	19.2	164.3	183.5	.0374	.3059	.3433
*86	27.10	12.40	.01822	3.29	54.88	.3043	23.1	162.6	185.7	.0445	.2978	.3423
95	31.82	17.12	.01838	2.83	54.41	.3536	26.9	160.8	187.7	.0515	.2897	.3412
104	37.17	22.47	.01854	2.44	53.94	.4090	30.8	159.0	189.9	.0583	.2819	.3402
113	43.16	28.46	.01870	2.13	53.47	.4704	34.6	157.2	191.8	.0651	.2743	.3394
122	49.88	35.18	.01887	1.86	53.00	.5382	38.5	155.3	193.8	.0718	.2668	.3386
131	57.36	42.66	.01904	1.63	52.52	.6135	42.3	153.3	195.6	.0783	.2594	.3377

† *Am. Soc. Refrigerating Eng., Circ.* 9, 1926.
* Standard ton temperatures.

Table 23. Properties of Saturated Vapor of "F-12" (CCl_2F_2)‡

Temp., °F.	Pressure		Volume		Density		Heat content from −40°			Entropy from −40°	
	Abs. lb. per sq. in.	Gage, lb. per sq. in.	Liquid, cu. ft. per lb.	Vapor, cu. ft. per lb.	Liquid, lb. per cu. ft.	Vapor, lb. per cu. ft.	Liquid, B.t.u. per lb.	Latent, B.t.u. per lb.	Vapor, B.t.u. per lb.	Liquid, B.t.u. per lb. per °F.	Vapor, B.t.u. per lb. per °F.
−40	9.32	*10.92	0.0106	3.911	94.58	0.2557	0	73.50	73.50	0	0.17517
−38	9.82	* 9.91	.0106	3.727	94.39	0.2683	0.40	73.34	73.74	0.00094	.17490
−36	10.34	* 8.87	.0106	3.553	94.20	0.2815	0.81	73.17	73.98	.00188	.17463
−34	10.87	* 7.80	.0106	3.389	93.99	0.2951	1.21	73.01	74.22	.00282	.17438
−32	11.43	* 6.66	.0107	3.234	93.79	0.3092	1.62	72.84	74.46	.00376	.17412
−30	12.02	* 5.45	.0107	3.088	93.59	0.3238	2.03	72.67	74.70	.00471	.17387
−28	12.62	* 4.23	.0107	2.950	93.39	0.3390	2.44	72.50	74.94	.00565	.17364
−26	13.26	* 2.93	.0107	2.820	93.18	0.3546	2.85	72.33	75.18	.00659	.17340
−24	13.90	* 1.63	.0108	2.698	92.98	0.3706	3.25	72.16	75.41	.00753	.17317
−22	14.58	* 0.24	.0108	2.583	92.78	0.3871	3.66	71.98	75.64	.00846	.17296
−20	15.28	0.58	.0108	2.474	92.58	0.4042	4.07	71.80	75.87	.00940	.17275
−18	16.01	1.31	.0108	2.370	92.38	0.4219	4.48	71.63	76.11	.01033	.17253
−16	16.77	2.07	.0108	2.271	92.18	0.4403	4.89	71.45	76.34	.01126	.17232
−14	17.55	2.85	.0109	2.177	91.97	0.4593	5.30	71.27	76.57	.01218	.17291
−12	18.37	3.67	.0109	2.088	91.77	0.4789	5.72	71.09	76.81	.01310	.17124
−10	19.20	4.50	.0109	2.003	91.57	0.4993	6.14	70.91	77.05	.01403	.17175
− 8	20.08	5.38	.0109	1.922	91.35	0.5203	6.57	70.72	77.29	.01496	.17158
− 6	20.98	6.28	.0110	1.845	91.14	0.5420	6.99	70.53	77.52	.01589	.17140
− 4	21.91	7.21	.0110	1.772	90.93	0.5644	7.41	70.34	77.75	.01682	.17123
− 2	22.87	8.17	.0110	1.703	90.72	0.5872	7.83	70.15	77.98	.01775	.17107
0	23.87	9.17	.0110	1.637	90.52	0.6109	8.25	69.96	78.21	.01869	.17091
2	24.89	10.19	.0110	1.574	90.31	0.6352	8.67	69.77	78.44	.01961	.17075
4	25.96	11.26	.0111	1.514	90.11	0.6606	9.10	69.57	78.67	.02052	.17060
†5	26.51	11.81	.0111	1.485	90.00	0.6735	9.32	69.47	78.79	.02097	.17052
6	27.05	12.35	.0111	1.457	89.88	0.6864	9.53	69.37	78.90	.02143	.17045
8	28.18	13.48	.0111	1.403	89.68	0.7129	9.96	69.17	79.13	.02235	.17030
10	29.35	14.65	.0112	1.351	89.45	0.7402	10.39	68.97	79.36	.02328	.17015
12	30.56	15.86	.0112	1.301	89.24	0.7687	10.82	68.77	79.59	.02419	.17001
14	31.80	17.10	.0112	1.253	89.03	0.7981	11.26	68.56	79.82	.02510	.16987
16	33.08	18.38	.0112	1.207	88.81	0.8288	11.70	68.35	80.05	.02601	.16974
18	34.40	19.70	.0113	1.163	88.58	0.8598	12.12	68.15	80.27	.02692	.16961
20	35.75	21.05	.0113	1.121	88.37	0.8921	12.55	67.94	80.49	.02783	.16949
22	37.15	22.45	.0113	1.081	88.13	0.9251	13.00	67.72	80.72	.02873	.16938
24	38.58	23.88	.0113	1.043	87.91	0.9588	13.44	67.51	80.95	.02963	.16926
26	40.07	25.37	.0114	1.007	87.68	0.9930	13.88	67.29	81.17	.03053	.16913
28	41.59	26.89	.0114	.973	87.47	1.028	14.32	67.07	81.39	.03143	.16900
30	43.16	28.46	.0115	.939	87.24	1.065	14.76	66.85	81.61	.03233	.16887
32	44.77	30.07	.0115	.908	87.02	1.102	15.21	66.62	81.83	.03323	.16876
34	46.42	31.72	.0115	.877	86.78	1.140	15.65	66.40	82.05	.03413	.16865
36	48.13	33.43	.0116	.848	86.55	1.180	16.10	66.17	82.27	.03502	.16854
38	49.88	35.18	.0116	.819	86.33	1.221	16.55	65.94	82.49	.03591	.16843
40	51.68	36.98	.0116	.792	86.10	1.263	17.00	65.71	82.71	.03680	.16833
42	53.51	38.81	.0116	.767	85.88	1.304	17.46	65.47	82.93	.03770	.16823
44	55.40	40.70	.0117	.742	85.66	1.349	17.91	65.24	83.15	.03859	.16813
46	57.35	42.65	.0117	.718	85.43	1.393	18.36	65.00	83.36	.03948	.16803
48	59.35	44.65	.0117	.695	85.19	1.438	18.82	64.74	83.57	.04037	.16794

‡ From *Am. Soc. Refrigerating Eng., Circ.* 12, by permission.

* In. Hg. below 1 atm.

† Standard ton temperatures.

Table 23. Properties of Saturated Vapor of "F-12" (CCl_2F_2)—*(Continued)*

Temp., °F.	Pressure		Volume		Density		Heat content from −40°			Entropy from −40°	
	Abs. lb. per sq. in.	Gage, lb. per sq. in.	Liquid, cu. ft. per lb.	Vapor, cu. ft. per lb.	Liquid, lb. per cu. ft.	Vapor, lb. per cu. ft.	Liquid, B.t.u. per lb.	Latent, B.t.u. per lb.	Vapor, B.t.u. per lb.	Liquid, B.t.u. per lb. per °F.	Vapor, B.t.u. per lb. per °F.
50	61.39	46.69	0.0118	.673	84.94	1.485	19.27	64.51	83.78	0.04126	0.16785
52	63.49	48.79	.0118	.652	84.71	1.534	19.72	64.27	83.99	.04215	.16779
54	65.63	50.93	.0118	.632	84.50	1.583	20.18	64.02	84.20	.04304	.16767
56	67.84	53.14	.0119	.612	84.28	1.633	20.64	63.77	84.41	.04392	.16758
58	70.10	55.40	.0119	.593	84.04	1.686	21.11	63.51	84.62	.04480	.16749
60	72.41	57.71	.0119	.575	83.78	1.740	21.57	63.25	84.82	.04568	.16741
62	74.77	60.07	.0120	.557	83.57	1.795	22.03	62.99	85.02	.04657	.16733
64	77.20	62.50	.0120	.540	83.34	1.851	22.49	62.73	85.22	.04745	.16725
66	79.67	64.97	.0120	.524	83.10	1.909	22.95	62.47	85.42	.04833	.16717
68	82.24	67.54	.0121	.508	82.86	1.968	23.42	62.20	85.62	.04921	.16709
70	84.82	70.12	.0121	.493	82.60	2.028	23.90	61.92	85.82	.05009	.16701
72	87.50	72.80	.0121	.479	82.37	2.090	24.37	61.65	86.02	.05097	.16693
74	90.20	75.50	.0122	.464	82.12	2.153	24.84	61.38	86.22	.05185	.16685
76	93.00	78.30	.0122	.451	81.87	2.218	25.32	61.10	86.42	.05272	.16677
78	95.85	81.15	.0123	.438	81.62	2.284	25.80	60.81	86.61	.05359	.16669
80	98.76	84.06	.0123	.425	81.39	2.353	26.28	60.52	86.80	.05446	.16662
82	101.7	87.00	.0123	.413	81.12	2.423	26.76	60.23	86.99	.05534	.16655
84	104.8	90.1	.0124	.401	80.87	2.495	27.24	59.94	87.18	.05621	.16648
*86	107.9	93.2	.0124	.389	80.63	2.569	27.72	59.65	87.37	.05708	.16640
88	111.1	96.4	.0124	.378	80.37	2.645	28.21	59.35	87.56	.05795	.16632
90	114.3	99.6	.0125	.368	80.11	2.721	28.70	59.04	87.74	.05882	.16624
92	117.7	103.0	.0125	.357	79.86	2.799	29.19	58.73	87.92	.05969	.16616
94	121.0	106.3	.0126	.347	79.60	2.880	29.68	58.42	88.10	.06056	.16608
96	124.5	109.8	.0126	.338	79.32	2.963	30.18	58.10	88.28	.06143	.16600
98	128.0	113.3	.0126	.328	79.06	3.048	30.67	57.78	88.45	.06230	.16592
100	131.6	116.9	.0127	.319	78.80	3.135	31.16	57.46	88.62	.06316	.16584
102	135.3	120.6	.0127	.310	78.54	3.224	31.65	57.14	88.79	.06403	.16576
104	139.0	124.3	.0128	.302	78.27	3.316	32.15	56.80	88.95	.06490	.16568
106	142.8	128.1	.0128	.293	78.00	3.411	32.65	56.46	89.11	.06577	.16560
108	146.8	132.1	.0129	.285	77.73	3.509	33.15	56.12	89.27	.06663	.16551
110	150.7	136.0	.0129	.277	77.46	3.610	33.65	55.78	89.43	.06749	.16542
112	154.8	140.1	.0130	.269	77.18	3.714	34.15	55.43	89.58	.06836	.16533
114	158.9	144.2	.0130	.262	76.89	3.823	34.65	55.08	89.73	.06922	.16524
116	163.1	148.4	.0131	.254	76.60	3.934	35.15	54.72	89.87	.07008	.16515
118	167.4	152.7	.0131	.247	76.32	4.049	35.65	54.36	90.01	.07094	.16505
120	171.8	157.1	.0132	.240	76.02	4.167	36.16	53.99	90.15	.07180	.16495
122	176.2	161.5	.0132	.233	75.72	4.288	36.66	53.62	90.28	.07266	.16484
124	180.8	166.1	.0133	.227	75.40	4.413	37.16	53.24	90.40	.07352	.16473
126	185.4	170.7	.0133	.220	75.10	4.541	37.67	52.85	90.52	.07437	.16462
128	190.1	175.4	.0134	.214	74.78	4.673	38.18	52.46	90.64	.07522	.16450
130	194.9	180.2	.0134	.208	74.46	4.808	38.69	52.07	90.76	.07607	.16438
132	199.8	185.1	.0135	.202	74.13	4.948	39.19	51.67	90.86	.07691	.16425
134	204.8	190.1	.0135	.196	73.81	5.094	39.70	51.26	90.96	.07775	.16411
136	209.9	195.2	.0136	.191	73.46	5.247	40.21	50.85	91.06	.07858	.16396
138	215.0	200.3	.0137	.185	73.10	5.405	40.72	50.43	91.15	.07941	.16380
140	220.2	205.5	.0138	.180	72.73	5.571	41.24	50.00	91.24	.08024	.16363

* Standard ton temperatures.

Table 24. Properties of Superheated Vapor of "F-12" (CCl_2F_2) *

Temp., °F.	Abs. pressure, 8 lb. per sq. in., gage pressure, 13.6 in. vac. (sat'n. temp., −45.8°F.)			Abs. pressure, 10 lb. per sq. in., gage pressure, 9.6 in. vac. (sat'n. temp., −37.3°F.)			Abs. pressure, 12 lb. per sq. in., gage pressure, 5.5 in. vac. (sat'n. temp., −30.0°F.)			Abs. pressure, 14 lb. per sq. in., gage pressure, 1.4 in. vac. (sat'n. temp., −23.7°F.)		
	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.
(At sat'n.)	*(4.502)*	*(72.80)*	*(0.17596)*	*(3.652)*	*(73.80)*	*(0.17480)*	*(3.093)*	*(74.69)*	*(0.17389)*	*(2.677)*	*(75.45)*	*(0.17314)*
−40	4.569	73.56	0.17777									
−30	4.684	74.87	.18085	3.728	74.77	0.17704	3.093	74.69	0.17389			
−20	4.799	76.20	.18390	3.821	76.11	.18008	3.172	76.02	.17695	2.706	75.94	0.17427
−10	4.914	77.54	.18691	3.913	77.46	.18310	3.250	77.37	.17998	2.773	77.28	.17731
0	5.028	78.89	.18991	4.006	78.81	.18611	3.328	78.73	.18299	2.841	78.64	.18032
10	5.142	80.26	.19284	4.098	80.18	.18905	3.405	80.10	.18594	2.908	80.01	.18328
20	5.257	81.64	.19574	4.189	81.56	.19194	3.483	81.48	.18884	2.974	81.40	.18618
30	5.370	83.02	.19860	4.280	82.94	.19482	3.560	82.87	.19173	3.041	82.80	.18907
40	5.484	84.43	.20143	4.371	84.35	.19766	3.637	84.28	.19458	3.107	84.21	.19192
50	5.598	85.85	.20425	4.463	85.77	.20047	3.714	85.71	.19739	3.173	85.63	.19475
60	5.711	87.27	.20703	4.556	87.19	.20326	3.790	87.14	.20018	3.239	87.06	.19753
70	5.824	88.72	.20977	4.648	88.64	.20601	3.867	88.59	.20293	3.303	88.51	.20028
80	5.938	90.18	.21250	4.740	90.11	.20874	3.943	90.05	.20566	3.369	89.97	.20302
90	6.051	91.64	.21519	4.832	91.58	.21144	4.019	91.52	.20836	3.435	91.44	.20572
100	6.165	93.13	.21786	4.923	93.05	.21411	4.095	94.00	.21104	3.501	92.93	.20841
110	6.278	94.63	.22051	5.015	94.56	.21676	4.170	94.50	.21367	3.567	94.43	.21106
120	6.391	96.13	.22314	5.107	96.07	.21940	4.246	96.01	.21631	3.633	95.95	.21369
130	6.504	97.64	.22573	5.198	97.59	.22199	4.323	97.53	.21891	3.699	97.48	.21630
140	6.617	99.18	.22831	5.289	99.14	.22458	4.400	99.07	.22151	3.763	99.01	.21889
150	6.730	100.73	.23087	5.379	100.66	.22713	4.474	100.62	.22406	3.828	100.56	.22144
160	6.843	102.29	.23340	5.470	102.24	.22967	4.549	102.18	.22659	3.894	102.13	.22400
170	6.955	103.87	.23591	5.560	103.81	.23218	4.624	103.75	.22911	3.960	103.71	.22651
180	7.068	105.44	.23842	5.650	105.40	.23469	4.700	105.34	.23162	4.025	105.30	.22902
190	7.181	107.05	.24090	5.740	107.00	.23717	4.774	106.94	.23409	4.090	106.90	.23150
200	7.294	108.67	.24337	5.831	108.63	.23963	4.850	108.55	.23656	4.155	108.52	.23398
210	7.407	110.28	.24581	5.921	110.25	.24208	4.926	110.18	.23901	4.220	110.14	.23643
220	7.520	111.93	.24825	6.011	111.88	.24451	5.000	111.82	.24144	4.285	111.78	.23886
230	7.633	113.57	.25066	6.101	113.53	.24692	5.076	113.47	.24385	4.350	113.44	.24128
240	......	......	......	......	......	......	5.152	115.15	.24626	4.414	115.11	.24368
250	......	......	......	......	......	......	......	......	...	4.479	116.79	.24607

* From *Am. Soc. Refrig. Eng., Circ.* 12, by permission.

Table 24. Properties of Superheated Vapor of "F-12" (CCl_2F_2)—(*Continued*)

Temp., °F.	Abs. pressure, 16 lb. per sq. in., gage pressure, 1.3 lb. per sq. in. (sat'n. temp., −18.0°F.)			Abs. pressure, 18 lb. per sq. in., gage pressure, 3.3 lb. per sq. in. (sat'n. temp., −12.9°F.)			Abs. pressure, 20 lb. per sq. in., gage pressure, 5.3 lb. per sq. in. (sat'n. temp., −8.2°F.)			Abs. pressure, 24 lb. per sq. in., gage pressure, 9.3 lb. per sq. in. (sat'n. temp., 0.3°F.)		
	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.
(At sat'n.)	*(2.370)*	*(76.11)*	*(0.17254)*	*(2.123)*	*(76.70)*	*(0.17302)*	*(1.925)*	*(77.27)*	*(0.17160)*	*(1.633)*	*(78.24)*	*(0.17089)*
−10	2.417	77.20	0.17498	2.137	77.09	0.17289						
0	2.476	78.56	.17800	2.191	78.47	.17592	1.965	78.39	0.17407			
10	2.535	79.94	.18095	2.244	79.85	.17889	2.013	79.76	.17704	1.664	79.59	0.17378
20	2.594	81.33	.18387	2.297	81.24	.18181	2.060	81.14	.17996	1.705	80.99	.17673
30	2.652	82.73	.18676	2.350	82.65	.18471	2.107	82.55	.18286	1.745	82.40	.17964
40	2.710	84.13	.18962	2.402	84.06	.18757	2.155	83.97	.18573	1.785	83.82	.18251
50	2.768	85.55	.19244	2.454	85.49	.19041	2.203	85.40	.18858	1.825	85.26	.18536
60	2.827	86.99	.19524	2.506	86.93	.19321	2.250	86.85	.19138	1.865	86.71	.18817
70	2.886	88.45	.19800	2.558	88.38	.19597	2.297	88.31	.19415	1.905	88.17	.19096
80	2.944	89.91	.20075	2.610	89.84	.19872	2.343	89.78	.19688	1.944	89.64	.19371
90	3.002	91.39	.20345	2.662	91.32	.20142	2.390	91.26	.19959	1.984	91.13	.19643
100	3.059	92.88	.20613	2.713	92.81	.20411	2.437	92.75	.20229	2.023	92.63	.19912
110	3.117	94.38	.20879	2.764	94.32	.20676	2.483	94.26	.20494	2.062	94.13	.20179
120	3.175	95.89	.21143	2.815	95.83	.20941	2.530	95.78	.20759	2.101	95.64	.20444
130	3.232	97.42	.21403	2.867	97.36	.21201	2.577	97.31	.21020	2.140	97.17	.20704
140	3.290	98.95	.21662	2.918	98.90	.21462	2.623	98.85	.21280	2.179	98.72	.20966
150	3.347	100.50	.21917	2.969	100.45	.21717	2.669	100.40	.21537	2.218	100.28	.21223
160	3.404	102.07	.22172	3.020	102.02	.21971	2.716	101.97	.21792	2.257	101.86	.21479
170	3.461	103.65	.22425	3.072	103.60	.22223	2.762	103.56	.22045	2.296	103.45	.21732
180	3.519	105.24	.22677	3.124	105.19	.22477	2.808	105.15	.22297	2.335	105.05	.21984
190	3.576	106.85	.22925	3.175	106.80	.22725	2.854	106.76	.22545	2.373	106.66	.22233
200	3.633	108.47	.23173	3.226	108.42	.22973	2.901	108.38	.22794	2.412	108.28	.22481
210	3.690	110.10	.23418	3.277	110.05	.23219	2.947	110.01	.23039	2.450	109.92	.22727
220	3.747	111.74	.23662	3.327	111.69	.23461	2.992	111.65	.23283	2.489	111.57	.22972
230	3.803	113.40	.23903	3.378	113.34	.23703	3.038	113.31	.23524	2.528	113.23	.23214
240	3.860	115.07	.24143	3.428	115.00	.23944	3.084	114.98	.23766	2.566	114.90	.23455
250	3.917	116.76	.24382	3.478	116.68	.24182	3.130	116.67	.24005	2.605	116.58	.26395
260	....	....	....	3.528	118.36	.24420	3.177	118.36	.24242	2.644	118.28	.23933
270	....	....	....	....	....	....	3.223	120.07	.24477	2.682	120.00	.24169

Table 24. Properties of Superheated Vapor of "F-12" (CCl_2F_2)—(*Continued*)

Temp., °F.	Abs. pressure, 28 lb. per sq. in., gage pressure, 13.3 lb. per sq. in. (sat'n. temp., 7.7°F.)			Abs. pressure, 32 lb. per sq. in., gage pressure, 17.3 lb. per sq. in. (sat'n. temp., 14.3°F.)			Abs. pressure, 36 lb. per sq. in., gage pressure, 21.3 lb. per sq. in. (sat'n. temp., 20.4°F.)			Abs. pressure, 40 lb. per sq. in., gage pressure, 25.3 lb. per sq. in. (sat'n. temp., 25.9°F.)		
	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.
(*At sat'n.*)	(*1.409*)	(*79.10*)	(*0.17032*)	(*1.245*)	(*79.84*)	(*0.16985*)	(*1.113*)	(*80.54*)	(*0.16947*)	(*1.009*)	(*81.16*)	(*0.16914*)
10	1.415	79.41	0.17099									
20	1.450	80.81	.17393	1.262	80.67	0.17152						
30	1.485	82.23	.17685	1.293	82.08	.17445	1.140	81.90	0.17227	1.019	81.76	0.17030
40	1.520	83.66	.17975	1.323	83.51	.17734	1.168	83.35	.17518	1.044	83.20	.17322
50	1.555	85.11	.18261	1.354	84.96	.18022	1.196	84.81	.17806	1.070	84.65	.17612
60	1.590	86.56	.18544	1.384	86.41	.18304	1.223	86.27	.18089	1.095	86.11	.17896
70	1.625	88.03	.18823	1.414	87.88	.18583	1.250	87.74	.18369	1.120	87.60	.18178
80	1.659	89.51	.19097	1.444	89.36	.18860	1.278	89.22	.18647	1.144	89.09	.18455
90	1.693	90.99	.19371	1.474	90.85	.19133	1.305	90.71	.18921	1.169	90.58	.18731
100	1.727	92.49	.19642	1.504	92.35	.19404	1.332	92.22	.19193	1.194	92.09	.19004
110	1.761	94.01	.19909	1.535	93.87	.19673	1.359	93.75	.19462	1.218	93.62	.19272
120	1.795	95.53	.20174	1.565	95.40	.19940	1.386	95.28	.19729	1.242	95.15	.19538
130	1.828	97.07	.20436	1.595	96.94	.20202	1.412	96.82	.19991	1.267	96.70	.19803
140	1.862	98.62	.20698	1.624	98.50	.20463	1.439	98.37	.20254	1.291	98.26	.20066
150	1.896	100.18	.20956	1.654	100.06	.20721	1.465	99.93	.20512	1.315	99.83	.20325
160	1.930	101.75	.21212	1.683	101.64	.20977	1.492	101.51	.20770	1.340	101.42	.20583
170	1.963	103.33	.21466	1.713	103.23	.21232	1.518	103.11	.21024	1.364	103.02	.20838
180	1.997	104.93	.21719	1.743	104.83	.21486	1.545	104.72	.21278	1.388	104.63	.21092
190	2.030	106.55	.21967	1.772	106.45	.21735	1.571	106.34	.21528	1.412	106.25	.21343
200	2.063	108.17	.22216	1.802	108.08	.21985	1.597	107.97	.21778	1.435	107.88	.21592
210	2.096	109.81	.22462	1.831	109.72	.22231	1.623	109.61	.22024	1.459	109.52	.21840
220	2.129	111.46	.22706	1.860	111.36	.22476	1.650	111.27	.22270	1.482	111.17	.22085
230	2.163	113.12	.22949	1.889	113.03	.22718	1.676	112.94	.22513	1.506	112.84	.22329
240	2.196	114.80	.23191	1.918	114.72	.22960	1.702	114.62	.22756	1.530	114.52	.22572
250	2.229	116.49	.23430	1.948	116.41	.23200	1.728	116.31	.22996	1.554	116.21	.22813
260	2.262	118.19	.23669	1.977	118.11	.23439	1.754	118.02	.23235	1.577	117.92	.23052
270	2.295	119.91	.23905	2.006	119.82	.23676	1.780	119.74	.23472	1.601	119.65	.23289
280	2.329	121.65	.24141	2.035	121.55	.23912	1.807	121.47	.23708	1.625	121.40	.23526
290	....	....	....	2.065	123.30	.24146	1.833	123.22	.23942	1.649	123.15	.23760
300	....	....	....	....	....	....	....	....	....	1.673	124.92	.23994

Table 24. Properties of Superheated Vapor of "F-12" (CCl_2F_2)—(*Continued*)

Temp., °F.	Abs. pressure, 50 lb. per sq. in., gage pressure, 35.3 lb. per sq. in. (sat'n. temp., 38.3°F.)			Abs. pressure, 60 lb. per sq. in., gage pressure, 45.3 lb. per sq. in. (sat'n. temp., 48.7°F.)			Abs. pressure, 70 lb. per sq. in., gage pressure, 55.3 lb. per sq. in. (sat'n. temp., 57.9°F.)			Abs. pressure, 80 lb. per sq. in., gage pressure, 65.3 lb. per sq. in. (sat'n. temp., 66.3°F.)		
	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.
(*At sat'n.*)	(*0.817*)	(*82.52*)	(*0.16841*)	(*0.688*)	(*83.65*)	(*0.16791*)	(*0.594*)	(*84.61*)	(*0.16749*)	(*0.521*)	(*85.45*)	(*0.16716*)
40	0.821	82.76	0.16895									
50	.842	84.24	.17187	0.690	83.83	0.16829						
60	.863	85.72	.17475	.708	85.33	.17120	0.597	84.94	0.16810			
70	.884	87.22	.17760	.726	86.84	.17407	.612	86.44	.17097	0.526	86.01	0.16819
80	.904	88.72	.18040	.743	88.35	.17689	.628	87.96	.17382	.540	87.56	.17108
90	.924	90.23	.18317	.760	89.87	.17968	.643	89.49	.17665	.554	89.12	.17394
100	.944	91.75	.18591	.778	91.41	.18246	.658	91.03	.17943	.568	90.68	.17675
110	.964	93.29	.18862	.795	92.96	.18519	.673	92.59	.18219	.582	92.26	.17954
120	.984	94.83	.19132	.812	94.51	.18789	.689	94.16	.18493	.596	93.84	.18229
130	1.004	96.39	.19397	.829	96.07	.19056	.704	95.75	.18763	.609	95.43	.18500
140	1.024	97.96	.19662	.846	97.65	.19323	.719	97.34	.19030	.623	97.03	.18771
150	1.044	99.54	.19923	.863	99.24	.19585	.733	98.94	.19293	.636	98.64	.19035
160	1.064	101.14	.20182	.880	100.84	.19846	.748	100.54	.19555	.649	100.26	.19298
170	1.084	102.75	.20439	.897	102.45	.20104	.763	102.16	.19814	.662	101.88	.19558
180	1.103	104.36	.20694	.913	104.07	.20360	.777	103.80	.20071	.675	103.52	.19817
190	1.123	105.98	.20946	.930	105.71	.20613	.792	105.45	.20325	.688	105.18	.20073
200	1.142	107.62	.21196	.946	107.36	.20865	.806	107.10	.20579	.701	106.84	.20328
210	1.162	109.28	.21444	.962	109.02	.21113	.820	108.76	.20829	.714	108.51	.20580
220	1.181	110.95	.21691	.979	110.69	.21361	.835	110.43	.21079	.726	110.19	.20828
230	1.200	112.62	.21935	.995	112.37	.21607	.849	112.13	.21325	.739	111.88	.21076
240	1.220	114.31	.22179	1.012	114.06	.21853	.863	113.83	.21570	.751	113.58	.21321
250	1.239	116.00	.22419	1.028	115.77	.22094	.878	115.55	.21815	.764	115.30	.21566
260	1.258	117.71	.22660	1.044	117.49	.22334	.892	117.28	.22057	.777	117.03	.21809
270	1.277	119.44	.22898	1.060	119.23	.22573	.906	119.02	.22296	.789	118.78	.22049
280	1.296	121.18	.23134	1.076	120.97	.22810	.920	120.76	.22534	.802	120.54	.22289
290	1.314	122.93	.23367	1.092	122.73	.23045	.934	122.52	.22770	.814	122.30	.22525
300	1.332	124.69	.23600	1.108	124.50	.23280	.948	124.29	.23006	.826	124.08	.22760
310	1.350	126.45	.23831	1.124	126.28	.23513	.961	126.07	.23239	.839	125.88	.22995
320	....	....	....	1.140	128.07	.23745	.975	127.88	.23471	.851	127.70	.23229
330	....	....	....	....	....	....	.989	129.70	.23702	.864	129.52	.23461

Table 24. Properties of Superheated Vapor of "F-12" (CCl_2F_2)—(*Continued*)

Temp., °F.	Abs. pressure 90 lb. per sq. in., gage pressure, 75.3 lb. per sq. in. (sat'n. temp., 73.9°F.)			Abs. pressure, 100 lb. per sq. in., gage pressure, 85.3 lb. per sq. in. (sat'n. temp., 80.9°F.)			Abs. pressure, 110 lb. per sq. in., gage pressure, 95.3 lb. per sq. in. (sat'n. temp., 87.3°F.)			Abs. pressure, 120 lb. per sq. in., gage pressure, 105.3 lb. per sq. in. (sat'n. temp., 93.4°F.)		
	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.
(*At sat'n.*)	(*0.465*)	(*86.21*)	(*0.16685*)	(*0.419*)	(*86.89*)	(*0.16659*)	(*0.382*)	(*87.50*)	(*0.16635*)	(*0.350*)	(*88.05*)	(*0.16610*)
80	0.473	87.18	0.16862									
90	.486	88.74	.17149	0.430	88.32	0.16926	0.385	87.91	0.16711			
100	.499	90.31	.17433	.442	89.93	.17210	.396	89.51	.17001	0.357	89.13	0.16803
110	.511	91.89	.17713	.454	91.54	.17493	.407	91.12	.17287	.367	90.75	.17090
120	.523	93.48	.17990	.465	93.15	.17773	.417	92.74	.17568	.377	92.38	.17374
130	.535	95.08	.18262	.477	94.76	.18049	.428	94.37	.17845	.387	94.01	.17654
140	.547	96.69	.18533	.488	96.37	.18321	.438	96.01	.18122	.397	95.65	.17932
150	.559	98.31	.18799	.499	97.99	.18590	.449	97.66	.18394	.407	97.30	.18207
160	.571	99.94	.19065	.510	99.63	.18856	.459	99.31	.18660	.417	98.96	.18474
170	.584	101.58	.19327	.521	101.28	.19120	.469	100.97	.18924	.426	100.63	.18743
180	.596	103.23	.19588	.531	102.94	.19381	.479	102.64	.19187	.436	102.31	.19011
190	.607	104.89	.19845	.542	104.61	.19638	.489	104.32	.19447	.445	104.00	.19271
200	.619	106.56	.20101	.553	106.29	.19894	.499	106.01	.19706	.454	105.70	.19529
210	.630	108.24	.20353	.563	107.98	.20148	.509	107.71	.19962	.463	107.41	.19785
220	.642	109.93	.20603	.574	109.68	.20401	.519	109.42	.20216	.472	109.13	.20041
230	.653	111.63	.20852	.585	111.39	.20650	.528	111.14	.20464	.482	110.86	.20294
240	.665	113.35	.21100	.595	113.11	.20899	.538	112.87	.20712	.491	112.60	.20545
250	.676	115.08	.21345	.606	114.84	.21145	.548	114.61	.20959	.500	114.35	.20792
260	.688	116.82	.21589	.616	116.58	.21389	.557	116.36	.21205	.508	116.11	.21035
270	.699	118.57	.21831	.626	118.33	.21631	.567	118.12	.21448	.517	117.88	.21279
280	.710	120.33	.22070	.636	120.10	.21870	.576	119.89	.21690	.526	119.66	.21521
290	.721	122.10	.22306	.646	121.88	.22108	.586	121.68	.21930	.534	121.45	.21760
300	.732	123.88	.22542	.657	123.67	.22347	.595	123.48	.22167	.543	123.25	.22000
310	.743	125.67	.22776	.667	125.47	.22583	.605	125.29	.22405	.552	125.07	.22238
320	.754	127.48	.23008	.677	127.28	.22817	.614	127.11	.22639	.560	126.90	.22472
330	.765	129.31	.23240	.687	129.10	.23050	.623	128.94	.22872	.569	128.74	.22707
340	....	....	....	.697	130.94	.23281	.632	130.78	.23103	.578	130.59	.22940
350	....	....	....	.707	132.80	.23510	.641	132.63	.23333	.586	132.45	.23171
360	....	....	....	.718	134.68	.23738	.651	134.50	.23562	.595	134.32	.23400

Table 24. Properties of Superheated Vapor of "F-12" (CCl_2F_2)—(*Continued*)

Temp., °F.	Abs. pressure, 130 lb. per sq. in., gage pressure, 115.3 lb. per sq. in. (sat'n. temp., 99.1°F.)			Abs. pressure, 140 lb. per sq. in., gage pressure, 125.3 lb. per sq in. (sat'n. temp., 104.5°F.)			Abs. pressure, 160 lb. per sq. in., gage pressure, 145.3 lb. per sq. in. (sat'n. temp., 114.5°F.)		
	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.
(*At sat'n.*)	(*0.323*)	(*88.547*)	(*0.16588*)	(*0.298*)	(*88.99*)	(*0.16566*)	(*0.259*)	(*89.77*)	(*0.16522*)
100	0.324	88.69	0.16615						
110	.333	90.33	.16905	0.304	89.92	0.16725			
120	.343	91.98	.17193	.314	91.60	.17021	0.264	90.68	0.16682
130	.353	93.64	.17476	.323	93.28	.17306	.273	92.40	.16977
140	.362	95.30	.17756	.332	94.96	.17590	.282	94.12	.17269
150	.371	96.97	.18030	.341	96.65	.17868	.290	95.84	.17553
160	.380	98.65	.18302	.350	98.34	.18142	.298	97.57	.17832
170	.389	100.34	.18571	.358	100.03	.18412	.306	99.31	.18106
180	.398	102.04	.18839	.366	101.72	.18678	.313	101.05	.18377
190	.407	103.74	.19102	.374	103.42	.18941	.321	102.80	.18646
200	.416	105.45	.19362	.383	105.14	.19205	.329	104.55	.18913
210	.424	107.16	.19620	.391	106.86	.19466	.336	106.31	.19175
220	.433	108.89	.19877	.399	108.59	.19724	.344	108.07	.19435
230	.442	110.62	.20130	.407	110.33	.19976	.351	109.83	.19693
240	.450	112.36	.20382	.415	112.09	.20229	.358	111.60	.19949
250	.458	114.11	.20629	.423	113.85	.20479	.366	113.38	.20203
260	.467	115.87	.20876	.431	115.63	.20728	.373	115.17	.20453
270	.475	117.64	.21122	.439	117.42	.20974	.380	116.97	.20700
280	.483	119.42	.21364	.447	119.22	.21219	.387	118.78	.20946
290	.492	121.21	.21605	.455	121.03	.21461	.394	120.60	.21189
300	.500	123.01	.21846	.462	122.85	.21701	.401	122.43	.21432
310	.508	124.82	.22084	.470	124.67	.21939	.408	124.27	.21672
320	.516	126.64	.22320	.477	126.50	.22174	.414	126.12	.21909
330	.524	128.47	.22554	.485	128.33	.22411	.421	127.98	.22145
340	.531	130.31	.22786	.492	130.17	.22646	.428	129.85	.22381
350	.539	132.17	.23016	.500	132.02	.22880	.435	131.73	.22616
360	.547	134.05	.23246	.507	133.89	.23109	.442	133.63	.22849
370	.555	135.94	.23475	.515	135.78	.23336	.448	135.55	.23079

Table 24. Properties of Superheated Vapor of "F-12" (CCl_2F_2)—(*Concluded*)

Temp., °F.	Abs. pressure, 180 lb. per sq. in., gage pressure, 165.3 lb. per sq. in. (sat'n. temp., 123.7°F.)			Abs. pressure, 200 lb. per sq. in., gage pressure, 185.3 lb. per sq. in. (sat'n. temp., 132.1°F.)			Abs. pressure, 220 lb. per sq. in., gage pressure, 205.3 lb. per sq. in. (sat'n. temp., 139.9°F.)		
	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	Vol., cu. ft. per lb.	Heat content, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.
(At sat'n.)	*(0.228)*	*(90.38)*	*(0.16476)*	*(0.202)*	*(90.86)*	*(0.16424)*	*(0.181)*	*(91.50)*	*(0.16375)*
130	0.233	91.47	0.16665						
140	.241	93.23	.16964	0.208	92.30	0.16661	0.181	91.52	0.16378
150	.249	94.99	.17254	.216	94.10	.16966	.188	93.32	.16685
160	.257	96.75	.17541	.224	95.90	.17262	.195	95.13	.16986
170	.265	98.52	.17823	.231	97.70	.17551	.202	96.94	.17282
180	.272	100.29	.18102	.238	99.51	.17838	.209	98.75	.17576
190	.280	102.07	.18377	.245	101.32	.18115	.216	100.57	.17861
200	.287	103.85	.18648	.252	103.13	.18388	.223	102.39	.18142
210	.294	105.63	.18912	.258	104.94	.18659	.229	104.22	.18420
220	.301	107.42	.19174	.265	106.76	.18927	.236	106.05	.18694
230	.307	109.21	.19433	.272	108.58	.19192	.242	107.89	.18962
240	.314	111.01	.19693	.278	110.40	.19455	.248	109.74	.19226
250	.321	112.81	.19947	.284	112.23	.19713	.254	111.59	.19487
260	.327	114.62	.20199	.290	114.06	.19967	.259	113.44	.19745
270	.334	116.44	.20449	.296	115.89	.20217	.265	115.30	.20001
280	.340	118.26	.20698	.302	117.73	.20467	.271	117.16	.20255
290	.347	120.09	.20944	.308	119.58	.20715	.277	119.03	.20504
300	.353	121.92	.21187	.314	121.44	.20961	.282	120.91	.20753
310	.359	123.76	.21428	.320	123.31	.21204	.288	122.80	.21002
320	.365	125.61	.21665	.326	125.19	.21445	.293	124.70	.21244
330	.371	127.47	.21904	.331	127.08	.21685	.299	126.60	.21485
340	.377	129.34	.22140	.337	128.98	.21923	.304	128.51	.21724
350	.383	131.23	.22374	.343	130.89	.22159	.309	130.42	.21960
360	.390	133.13	.22608	.348	132.81	.22392	.314	132.34	.22195
370	.396	135.05	.22840	.354	134.74	.22624	.320	134.27	.22430
380	.402	136.98	.23072	.360	136.68	.22856	.325	136.21	.22665
390	.408	138.91	.23301	.365	138.63	.23085	.330	138.16	.22895
400	.414	140.85	.23529	.370	140.59	.23314	.335	140.12	.23124

Table 25. Properties of Saturated Vapor.† Dichloromonofluoromethane ($CHCl_2F$)—"F-21"

Temp., °F.	Pressure		Volume		Density		Heat content from −40°			Entropy from −40°	
	Abs., lb. per sq. in.	Gage, lb. per sq. in.	Liquid, cu. ft. per lb.	Vapor, cu. ft. per lb.	Liquid, lb. per cu. ft.	Vapor, lb. per cu. ft.	Liquid, B.t.u. per lb.	Latent, B.t.u. per lb.	Vapor, B.t.u. per lb.	Liquid B.t.u. per lb. °F.	Vapor B.t.u. per lb. °F.
t	p	p_d	v_f	v_g	$1/v_f$	$1/v_g$	h_f	h	h_g	s_f	s_g
−40	1.358	27.16*	0.01058	32.09	94.52	0.03116	0.00	114.56	114.56	0.0000	0.2730
−30	1.888	26.08*	.01066	23.61	93.79	.04236	2.36	113.40	115.76	.0055	.2695
−20	2.578	24.67*	.01075	17.66	93.04	.05661	4.71	112.25	116.96	.0109	.2663
−10	3.463	22.87*	.01084	13.43	92.28	.07449	7.07	111.10	118.17	.0162	.2633
0	4.582	20.59*	.01093	10.35	91.52	.09662	9.44	109.93	119.37	.0214	.2606
5†	5.243	19.25*	.01097	9.132	91.13	.1095	10.63	109.34	119.97	.0240	.2593
10	5.978	17.75*	.01102	8.085	90.74	.1237	11.81	108.76	120.57	.0265	.2581
20	7.699	14.25*	.01112	6.392	89.96	.1565	14.21	107.57	121.78	.0316	.2559
30	9.793	9.98*	.01122	5.112	89.16	.1956	16.61	106.37	122.98	.0365	.2538
40	12.32	4.84*	.01132	4.130	88.35	.2421	19.04	105.15	124.19	.0414	.2519
50	15.33	0.63	.01142	3.370	87.54	.2968	21.49	103.90	125.39	.0463	.2502
60	18.90	4.20	.01153	2.773	86.71	.3606	23.98	102.62	126.60	.0511	.2486
70	23.08	8.38	.01164	2.300	85.87	.4346	26.49	101.30	127.79	.0559	.2471
80	27.96	13.26	.01176	1.923	85.03	.5201	29.03	99.95	128.98	.0606	.2458
86†	31.23	16.53	.01183	1.733	84.52	.5770	30.56	99.12	129.68	.0634	.2450
90	33.58	18.88	.01188	1.619	84.17	.6177	31.59	98.55	130.14	.0652	.2446
100	40.04	25.34	.01200	1.371	83.31	.7292	34.18	97.11	131.29	.0699	.2434
110	47.40	32.70	.01213	1.169	82.43	.8557	36.79	95.63	132.42	.0745	.2424
120	55.75	41.05	.01226	1.001	81.54	.9986	39.46	94.08	133.53	.0791	.2414
130	65.15	50.45	.01240	0.8623	80.65	1.160	42.13	92.48	134.61	.0837	.2405
140	75.72	61.02	.01254	.7457	79.74	1.341	44.86	90.80	135.66	.0882	.2396
150	87.51	72.81	.01269	.6476	78.82	1.544	47.62	89.06	136.68	.0927	.2388
160	100.6	85.91	.01284	.5646	77.90	1.771	50.43	87.26	137.69	.0972	.2381

Note: Melting point, −211°F.
Boiling point, 48°F.
Critical temperature, 353.3°F.
Critical pressure, 750 lb. per sq. in. abs.

‡ Courtesy Kinetic Chemicals, Inc.

* In. Hg below 1 atm.

† Standard ton temperature.

Table 26. Properties of Superheated Vapor.* Dichloromonofluoromethane ($CHCl_2F$)—"F-21"

Temp., °F.	Abs. pressure 1.2 lb. per sq. in., gage pressure 27.48 in. vac. (sat'n. temp. −43.6°F.)			Temp., °F.	Abs. pressure 2 lb. per sq. in., gage pressure 25.85 in. vac. (sat'n. temp. −28.2°F.)			Temp., °F.	Abs. pressure 4 lb. per sq. in., gage pressure 21.78 in. vac. (sat'n. temp. −4.9°F.)			Temp., °F.	Abs. pressure 10 lb. per sq. in., gage pressure 9.37 in. vac. (sat'n. temp. 30.9°F.)		
t	*V*	*H*	*S*	*t*	*V*	*H*	*S*	*t*	*V*	*H*	*S*	*t*	*V*	*H*	*S*
(At sat'n.)	*(36.02)*	*(114.13)*	*(0.2744)*	*(At sat'n.)*	*(22.37)*	*(115.98)*	*(0.2689)*	*(At sat'n.)*	*(11.74)*	*(118.78)*	*(0.2619)*	*(At sat'n.)*	*(5.014)*	*(123.10)*	*(0.2536)*
−40	36.34	114.57	0.2754	−20	22.80	116.99	0.2712								
−30	37.21	115.80	.2783	−10	23.33	118.24	.2740	0	11.87	119.40	0.2633				
−20	38.08	117.03	.2812	0	23.85	119.49	.2768	10	12.14	120.66	.2660	40	5.112	124.30	0.2561
−10	38.95	118.28	.2840	10	24.38	120.76	.2795	20	12.40	121.95	.2687	50	5.219	125.63	.2587
0	39.83	119.54	.2867	20	24.90	122.04	.2822	30	12.66	123.25	.2714	60	5.326	126.99	.2613
10	40.70	120.80	.2894	30	25.43	123.34	.2849	40	12.93	124.57	.2741	70	5.433	128.36	.2639
20	41.57	122.08	.2921	40	25.95	124.65	.2876	50	13.19	125.90	.2767	80	5.540	129.74	.2665
30	42.44	123.38	.2948	50	26.47	125.98	.2902	60	13.45	127.25	.2793	90	5.646	131.13	.2691
40	43.32	124.69	.2975	60	27.00	127.33	.2928	70	13.72	128.61	.2819	100	5.753	132.54	.2716
50	44.19	126.01	.3001	70	27.52	128.69	.2954	80	13.98	129.99	.2845	110	5.860	133.95	.2741
60	45.06	127.35	.3027	80	28.05	130.07	.2979	90	14.24	131.37	.2870	120	5.966	135.38	.2766
70	45.93	128.71	.3053	90	28.57	131.45	.3005	100	14.51	132.77	.2895	130	6.072	136.82	.2791
80	46.80	130.09	.3079	100	29.09	132.85	.3030	110	14.77	134.18	.2920	140	6.178	138.28	.2815
90	47.67	131.48	.3104	110	29.62	134.26	.3055	120	15.03	135.61	.2945	150	6.285	139.75	.2840
100	48.54	132.88	.3129	120	30.14	135.68	.3080	130	15.29	137.04	.2970	160	6.391	141.24	.2864
110	49.42	134.29	.3154	130	30.66	137.12	.3104	140	15.56	138.49	.2994	170	6.497	142.75	.2888
120	50.29	135.71	.3179	140	31.19	138.57	.3129	150	15.82	139.96	.3019	180	6.603	144.27	.2912
130	51.16	137.14	.3203	150	31.71	140.03	.3153	160	16.08	141.45	.3043	190	6.709	145.79	.2936
140	52.03	138.59	.3228	160	32.23	141.52	.3177	170	16.35	142.95	.3067	200	6.815	147.33	.2959
150	52.90	140.06	.3252	170	32.76	143.02	.3201	180	16.61	144.46	.3091	210	6.921	148.89	.2983
160	53.77	141.55	.3276	180	33.28	144.53	.3225	190	16.87	145.98	.3114	220	7.027	150.46	.3006
170	54.64	143.05	.3300	190	33.80	146.05	.3249	200	17.13	147.52	.3138	230	7.133	152.05	.3029
180	55.51	144.56	.3324	200	34.33	147.59	.3272	210	17.39	149.08	.3161	240	7.239	153.65	.3052
190	56.38	146.08	.3348	210	34.85	149.14	.3296	220	17.66	150.65	.3184	250	7.344	155.26	.3075
200	57.25	147.61	.3371	220	35.37	150.71	.3319	230	17.92	152.23	.3207	260	7.450	156.88	.3098
210	58.12	149.16	.3394	230	35.89	152.29	.3342	240	18.18	153.82	.3230	270	7.555	158.52	.3120
220	58.99	150.73	.3418	240	36.42	153.88	.3365	250	18.44	155.43	.3253	290	7.761	160.17	.3143
230	59.86	152.31	.3441	250	36.94	155.49	.3388	260	18.70	157.05	.3276	290	7.767	161.84	.3165
240	60.73	153.90	.3464	260	......	......	......	270	18.97	158.68	.3298	300	7.872	163.52	.3187
250	61.60	155.51	.3486	...	......	......	......	...	......	......	......	310	7.978	165.21	.3210

* Courtesy Kinetic Chemicals, Inc.

Table 26. Properties of Superheated Vapor. Dichloromonofluoromethane ($CHCl_2F$)—"**F-21**"—(*Concluded*)

Temp., °F.	Abs. pressure 20 lb. per sq. in., gage pressure 5.3 lb. per sq. in. (sat'n. temp. 62.8°F.)			Temp., °F.	Abs. pressure 50 lb. per sq. in., gage pressure 35.3 lb. per sq. in. (sat'n. temp. 113.3°F.)			Temp., °F.	Abs. pressure 100 lb. per sq. in., gage pressure 85.3 lb. per sq. in. (sat'n. temp. 159.6°F.)		
t	*V*	*H*	*S*	*t*	*V*	*H*	*S*	*t*	*V*	*H*	*S*
(*At sat'n.*)	(*2.630*)	(*126.94*)	(*0.2483*)	(*At sat'n.*)	(*1.111*)	(*132.79*)	(*0.2421*)	(*At sat'n.*)	(*0.5680*)	(*137.66*)	(*0.2381*)
60	.....	......	......	100	......	......	......	160	0.5685	137.72	0.2382
70	2.670	127.94	0.2501	110	......	......	......	170	.5815	139.32	.2408
80	2.725	129.33	.2527	120	1.127	133.78	0.2438	180	.5945	140.92	.2433
90	2.780	130.73	.2552	130	1.150	135.26	.2463	190	.6073	142.53	.2458
100	2.834	132.14	.2578	140	1.173	136.75	.2488	200	.6200	144.16	.2483
110	2.889	133.56	.2603	150	1.197	138.26	.2513	210	.6326	145.80	.2508
120	2.943	135.00	.2628	160	1.220	139.79	.2538	220	.6450	147.45	.2532
130	2.998	136.45	.2653	170	1.243	141.33	.2563	230	.6574	149.10	.2556
140	3.052	137.91	.2678	180	1.265	142.88	.2587	240	.6698	150.76	.2580
150	3.106	139.39	.2702	190	1.288	144.43	.2612	250	.6820	152.44	.2604
160	3.160	140.90	.2727	200	1.311	146.00	.2636	260	.6942	154.13	.2628
170	3.214	142.41	.2751	210	1.334	147.59	.2659	270	.7063	155.83	.2651
180	3.268	143.93	.2775	220	1.356	149.19	.2683	280	.7183	157.54	.2674
190	3.322	145.46	.2799	230	1.379	150.80	.2707	290	.7303	159.26	.2698
200	3.376	147.01	.2822	240	1.401	152.42	.2730	300	.7423	161.00	.2721
210	3.430	148.58	.2846	250	1.424	154.06	.2753	310	.7542	162.75	.2743
220	3.484	150.16	.2869	260	1.446	155.71	.2776	320	.7660	164.51	.2766
230	3.537	151.75	.2892	270	1.469	157.37	.2799	330	.7777	166.28	.2789
240	3.591	153.36	.2915	280	1.491	159.05	.2822	340	.7894	168.06	.2811
250	3.645	154.97	.2938	290	1.513	160.74	.2845	350	.8011	169.85	.2833
260	3.698	156.60	.2961	300	1.535	162.44	.2867	360	.8128	171.66	.2856
270	3.752	158.24	.2984	310	1.558	164.16	.2890	370	.8244	173.48	.2878
280	3.805	159.90	.3007	320	1.580	165.89	.2912	380	.8360	175.31	.2900
290	3.859	161.57	.3029	330	1.602	167.63	.2934	390	.8475	177.16	.2922
300	3.912	163.25	.3051	340	1.624	169.38	.2956	400	.8590	179.02	.2943
310	3.965	164.95	.3074	350	1.646	171.14	.2978				
320	4.019	166.67	.3096	360	1.668	172.92	.3000				
330	4.072	168.40	.3118	370	1.690	174.72	.3022				
340	4.125	170.14	.3140	380	1.712	176.52	.3044				
...	.....	......	......	390	1.734	178.34	.3065				

Table 27. Properties of Saturated Vapor.‡ Trichloromonofluoromethane (CCl_3F)—"F-11"

Temp.	Pressure		Volume		Density		Heat content from −40°			Entropy from −40°	
°F.	Abs., lb. per sq. in.	Gage, lb. per sq. in.	Liquid, cu. ft. per lb.	Vapor, cu. ft. per lb.	Liquid lb. per cu. ft.	Vapor lb. per cu. ft.	Liquid B.t.u. per lb.	Latent B.t.u. per lb.	Vapor B.t.u. per lb.	Liquid B.t.u. per lb. °F.	Vapor B.t.u. per lb. °F.
t	p	p_d	v_f	v_g	$1/v_f$	$1/v_g$	h_f	h	h_g	s_f	s_g
−40	0.7391	28.42*	0.00988	44.21	101.25	0.02262	0.00	87.48	87.48	0.0000	0.2085
−30	1.034	27.81*	.00995	32.33	100.52	.03093	1.97	86.70	88.67	.0046	.2064
−20	1.420	27.03*	.01002	24.06	99.77	.04157	3.94	85.93	89.87	.0091	.2046
−10	1.920	26.01*	.01010	18.17	99.03	.05503	5.91	85.16	91.07	.0136	.2030
5†	2.931	23.95*	.01022	12.27	97.88	.08152	8.88	84.00	92.88	.0201	.2009
10	3.352	23.10*	.01026	10.83	97.50	.09233	9.88	83.60	93.48	.0222	.2003
20	4.342	21.08*	.01034	8.519	96.72	.1174	11.87	82.82	94.69	.0264	.1991
30	5.557	18.61*	.01042	6.776	95.94	.1476	13.88	82.03	95.91	.0306	.1981
40	7.032	15.61*	.01051	5.447	95.14	.1836	15.89	81.22	97.11	.0346	.1972
50	8.804	12.00*	.01060	4.421	94.34	.2262	17.92	80.40	98.32	.0386	.1964
60	10.90	7.73*	.01069	3.626	93.53	.2758	19.96	79.57	99.53	.0426	.1958
70	13.40	2.64*	.01079	2.993	92.71	.3342	22.02	78.71	100.73	.0465	.1951
80	16.31	1.61	.01088	2.492	91.88	.4012	24.09	77.84	101.93	.0504	.1947
86†	18.28	3.58	.01094	2.242	91.38	.4461	25.34	77.31	102.65	.0527	.1944
90	19.69	4.99	.01098	2.091	91.04	.4783	26.18	76.95	103.12	.0542	.1942
100	23.60	8.90	.01109	1.765	90.19	.5666	28.27	76.03	104.30	.0580	.1938
110	28.09	13.39	.01119	1.499	89.34	.6671	30.40	75.08	105.47	.0617	.1935
120	33.20	18.50	.01130	1.281	88.47	.7808	32.53	74.10	106.63	.0654	.1933
130	38.96	24.26	.01142	1.101	87.60	.9080	34.67	73.11	107.78	.0691	.1931
140	45.50	30.80	.01154	0.9505	86.69	1.052	36.84	72.07	108.91	.0727	.1929
150	52.85	38.15	.01166	.8240	85.78	1.214	39.02	71.00	110.02	.0763	.1927
160	61.04	46.34	.01179	.7176	84.85	1.394	41.23	69.89	111.12	.0798	.1926

Note: Melting point, −168°F.
Boiling point, 74.7°F.
Critical temperature, 388.4°F.
Critical pressure, 635 lb. per sq. in. abs.

‡ Courtesy Kinetic Chemicals, Inc.

* In. Hg below 1 atm.

† Standard ton temperature.

Table 28. Properties of Superheated Vapor.* Trichloromonofluoromethane (CCl_3F)—**"F-11"**

Temp., °F.	Abs. pressure 0.7 lb. per sq. in., gage pressure 28.50 in. vac. (sat'n. temp. −41.6°F.)			Temp., °F.	Abs. pressure 2.0 lb. per sq. in., gage pressure 25.85 in. vac. (sat'n. temp. −8.6°F.)			Temp., °F.	Abs. pressure 5.0 lb. per sq. in., gage pressure 19.74 in. vac. (sat'n. temp. 25.6°F.)			Temp., °F.	Abs. pressure 10.0 lb. per sq. in., gage pressure 9.57 in. vac. (sat'n. temp. 55.9°F.)		
t	*V*	*H*	*S*	*t*	*V*	*H*	*S*	*t*	*V*	*H*	*S*	*t*	*V*	*H*	*S*
(*At sat'n.*)	(*46.54*)	(*87.89*)	(*0.2090*)	(*At sat'n.*)	(*17.50*)	(*91.24*)	(*0.2088*)	(*At sat'n.*)	(*7.475*)	(*95.38*)	(*0.1985*)	(*At sat'n.*)	(*3.928*)	(*99.04*)	(*0.1960*)
−40	46.69	87.49	0.2093	−10	......	......	......	20	......	......	......	50			
−30	47.81	88.69	.2121	0	17.83	92.31	0.2052	30	7.543	95.94	0.1997	60	3.961	99.57	0.1970
−20	48.93	89.91	.2149	10	18.23	93.56	.2079	40	7.703	97.22	.2023	70	4.042	100.89	.1996
−10	50.06	91.14	.2177	20	18.62	94.82	.2105	50	7.863	98.51	.2048	80	4.123	102.23	.2021
0	51.18	92.38	.2204	30	19.02	96.09	.2131	60	8.023	99.82	.2074	90	4.204	103.57	.2045
10	52.30	93.63	.2231	40	19.41	97.37	.2157	70	8.182	101.14	.2099	100	4.285	104.93	.2070
20	53.42	94.89	.2258	50	19.81	98.66	.2183	80	8.341	102.47	.2124	110	4.366	106.29	.2094
30	54.54	96.16	.2284	60	20.20	99.97	.2208	90	8.500	103.80	.2148	120	4.447	107.67	.2118
40	55.66	97.44	.2310	70	20.59	101.28	.2233	100	8.659	105.16	.2173	130	4.528	109.05	.2141
50	56.77	98.73	.2335	80	20.99	102.61	.2258	110	8.818	106.51	.2197	140	4.609	110.44	.2165
60	57.89	100.03	.2361	90	21.38	103.94	.2282	120	8.977	107.88	.2220	150	4.690	111.85	.2188
70	59.01	101.34	.2386	100	21.77	105.28	.2307	130	9.135	109.26	.2244	160	4.771	113.26	.2211
80	60.13	102.67	.2411	110	22.17	106.64	.2331	140	9.294	110.65	.2267	170	4.851	114.69	.2234
90	61.25	104.00	.2435	120	22.56	108.01	.2354	150	9.452	112.05	.2290	180	4.932	116.12	.2257
100	62.37	105.34	.2459	130	22.95	109.38	.2378	160	9.610	113.45	.2313	190	5.012	117.56	.2279
110	63.49	106.69	.2483	140	23.35	110.76	.2401	170	9.769	114.87	.2336	200	5.092	119.01	.2301
120	64.60	108.06	.2507	150	23.74	112.17	.2424	180	9.927	116.30	.2359	210	5.171	120.48	.2323
130	65.72	109.43	.2531	160	24.13	113.57	.2447	190	10.09	117.74	.2381	220	5.251	121.95	.2345
140	66.84	110.82	.2554	170	24.52	114.98	.2470	200	10.25	119.19	.2403	230	5.331	123.43	.2367
150	67.96	112.22	.2577	180	24.92	116.41	.2492	210	10.40	120.65	.2425	240	5.411	124.92	.2388
160	69.08	113.62	.2600	190	25.31	117.85	.2515	220	10.56	122.12	.2447	250	5.491	126.43	.2409
170	70.20	115.04	.2622	200	25.70	119.30	.2537	230	10.72	123.60	.2469	260	5.571	127.95	.2431
180	71.32	116.47	.2645	210	26.10	120.76	.2559	240	10.88	125.09	.2490	270	5.650	129.48	.2452
190	72.43	117.90	.2667	220	26.49	122.22	.2581	250	11.04	126.59	.2511	280	5.730	131.01	.2473
200	73.55	119.35	.2689	230	26.88	123.70	.2602	260	11.19	128.11	.2533	290	5.809	132.56	.2494
210	74.67	120.81	.2711	240	27.27	125.19	.2624	270	11.35	129.63	.2554	300	5.889	134.12	.2514
220	75.79	122.27	.2733	250	27.66	126.69	.2645	280	11.51	131.16	.2574	310	5.969	135.68	.2535
230	76.90	123.75	.2754	260	28.06	128.21	.2666	290	11.67	132.71	.2595	320	6.048	137.26	.2555
240	78.02	125.24	.2776	270	28.45	129.73	.2687	300	11.82	134.27	.2616	330	6.127	138.84	.2575
250	79.14	126.74	.2797	280	28.84	131.26	.2708	310	11.98	135.83	.2636	340	6.206	140.43	.2595
260	80.26	128.25	.2818	290	29.23	132.80	.2728	320	12.14	137.40	.2656	350	6.285	142.04	.2615
270	......	......	......	300	29.62	134.35	.2749	330	12.30	138.98	.2677	360	6.365	143.66	.2635

* Courtesy Kinetic Chemicals, Inc.

Table 28. Properties of Superheated Vapor. Trichloromonofluoromethane (CCl$_3$F)—"F-11"—(*Concluded*)

Temp., °F.	Abs. pressure 20 lb. per sq. in., gage pressure 5.3 lb. per sq. in. (sat'n. temp. 90.8°F.)			Temp., °F.	Abs. pressure 40 lb. per sq. in., gage pressure 25.3 lb. per sq. in. (sat'n. temp. 131.6°F.)			Temp., °F.	Abs. pressure 60 lb. per sq. in., gage pressure 45.3 lb. per sq. in. (sat'n. temp. 158.8°F.)		
t	*V*	*H*	*S*	*t*	*V*	*H*	*S*	*t*	*V*	*H*	*S*
(*At sat'n.*)	(*2.061*)	(*103.22*)	(*0.1942*)	(*At sat'n.*)	(*1.074*)	(*107.96*)	(*0.1930*)	(*At sat'n.*)	(*0.7297*)	(*110.99*)	(*0.1926*)
80	......	......	......	130	......	......	......	160	0.7314	111.17	0.1929
90	......	......	......	140	1.092	109.17	0.1950	170	.7471	112.65	.1953
100	2.099	104.47	0.1964	150	1.115	110.60	.1974	180	.7627	114.14	.1976
110	2.140	105.84	.1989	160	1.137	112.04	.1997	190	.7782	115.63	.1999
120	2.182	107.23	.2013	170	1.159	113.49	.2021	200	.7937	117.13	.2022
130	2.224	108.63	.2036	180	1.181	114.95	.2044	210	.8090	118.64	.2045
140	2.266	110.03	.2060	190	1.202	116.42	.2067	220	.8241	120.16	.2068
150	2.307	111.44	.2083	200	1.224	117.89	.2089	230	.8391	121.68	.2090
160	2.349	112.87	.2107	210	1.246	119.38	.2112	240	.8541	123.21	.2112
170	2.390	114.31	.2130	220	1.268	120.88	.2134	250	.8691	124.76	.2134
180	2.431	115.75	.2152	230	1.289	122.39	.2156	260	.8839	126.31	.2156
190	2.473	117.19	.2175	240	1.311	124.91	.2178	270	.8987	127.87	.2177
200	2.514	118.65	.2197	250	1.332	125.44	.2199	280	.9134	129.44	.2198
210	2.555	120.12	.2219	260	1.353	126.97	.2221	290	.9281	131.01	.2220
220	2.596	121.60	.2241	270	1.374	128.52	.2242	300	.9427	132.60	.2241
230	2.637	123.09	.2263	280	1.396	130.08	.2263	310	.9574	134.20	.2262
240	2.678	124.59	.2285	290	1.417	131.64	.2284	320	.9719	135.81	.2282
250	2.719	126.11	.2306	300	1.438	133.22	.2305	330	.9863	137.42	.2303
260	2.760	127.63	.2328	310	1.459	134.80	.2326	340	1.001	139.05	.2323
270	2.800	129.16	.2349	320	1.480	136.40	.2347	350	1.015	140.69	.2344
280	2.841	130.70	.2370	330	1.501	138.00	.2367	360	1.030	142.33	.2364
290	2.881	132.25	.2391	340	1.522	139.61	.2387	370	1.044	143.98	.2384
300	2.922	133.82	.2411	350	1.543	141.24	.2407	380	1.059	145.65	.2404
310	2.962	135.39	.2432	360	1.564	142.88	.2427	390	1.073	147.32	.2424
320	3.003	136.97	.2452	370	1.584	144.52	.2447	400	1.087	149.01	.2443
330	3.043	138.56	.2472	380	1.605	146.18	.2467	410	1.101	150.70	.2463
340	3.084	140.16	.2493	390	1.626	147.83	.2487	420	1.115	152.41	.2483
350	3.124	141.77	.2513	400	1.647	149.51	.2506	430	1.130	154.12	.2502
360	3.164	143.40	.2532	410	1.668	151.19	.2526	440	1.144	155.84	.2521
370	3.205	145.03	.2552	420	1.688	152.89	.2545	450	1.158	157.57	.2540
380	3.245	146.68	.2572	430	1.709	154.59	.2564	460	1.172	159.31	.2559
390	3.286	148.33	.2592	440	1.729	156.30	.2583				
400	3.326	149.99	.2611								

Table 29. Properties of Propane[†]

Temp., °F.	Pressure, lb. per sq. in.		Specific volume, cu. ft. per lb.		Density, lb. per cu. ft.		Heat content, B.t.u. per lb.		Latent heat, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	
	Absolute	Gage	Liquid	Vapor	Liquid	Vapor	Liquid	Vapor		Liquid	Vapor
−75	6.37	*17.0	0.02660	14.5	37.59	0.0690	−39.5	151.0	190.5	−0.092	0.404
−70	7.37	*14.9	.02674	12.9	37.40	0.0775	−37.0	152.5	189.5	− .086	.400
−65	8.48	*12.7	.02688	11.3	37.20	0.0885	−34.5	153.5	188.0	− .080	.397
−60	9.72	*10.1	.02703	9.93	37.00	0.101	−32.0	155.0	187.0	− .074	.393
−55	11.1	* 7.3	.02717	8.70	36.80	0.115	−29.0	156.5	185.5	− .067	.391
−50	12.6	* 4.3	.02732	7.74	36.60	0.129	−26.5	158.0	184.5	− .061	.389
−45	14.4	* 0.6	.02748	6.89	36.39	0.145	−24.0	159.0	183.0	− .055	.386
−40	16.2	1.5	.02763	6.13	36.19	0.163	−21.5	160.0	181.5	− .049	.384
−35	18.1	3.4	.02779	5.51	35.99	0.181	−19.0	161.0	180.0	− .042	.382
−30	20.3	5.6	.02795	4.93	35.78	0.203	−16.0	163.0	179.0	− .036	.380
−25	22.7	8.0	.02811	4.46	35.58	0.224	−13.5	164.0	177.5	− .030	.378
−20	25.4	10.7	.02827	4.00	35.37	0.250	−11.0	165.0	176.0	− .024	.377
−15	28.3	13.6	.02844	3.60	35.16	0.278	− 8.0	167.0	175.0	− .018	.375
−10	31.4	16.7	.02860	3.26	34.96	0.307	− 5.5	168.0	173.5	− .012	.374
− 5	34.7	20.0	.02878	2.97	34.75	0.337	− 2.5	169.5	172.0	− .006	.372
0	38.2	23.5	.02895	2.71	34.54	0.369	0	170.5	170.5	.000	.371
+ 5	41.9	27.2	.02913	2.48	34.33	0.403	+ 3.0	172.0	169.5	+ .006	.370
10	46.0	31.3	.02931	2.27	34.12	0.441	5.5	173.5	168.0	.012	.370
15	50.6	35.9	.02950	2.07	33.90	0.483	8.5	175.0	166.5	.018	.369
20	55.5	40.8	.02970	1.90	33.67	0.526	11.0	176.0	165.0	.024	.368
25	60.9	46.2	.02991	1.74	33.43	0.575	14.0	177.5	163.5	.030	.368
30	66.3	51.6	.03012	1.60	33.20	0.625	17.0	179.0	162.0	.035	.366
35	72.0	57.3	.03033	1.48	32.97	0.676	20.0	180.5	160.5	.041	.366
40	78.0	63.3	.03055	1.37	32.73	0.730	23.0	182.0	159.0	.047	.366
45	84.6	69.9	.03078	1.27	32.49	0.787	26.0	183.5	157.5	.053	.365
50	91.8	77.1	.03102	1.18	32.24	0.847	29.0	185.0	156.0	.059	.365
55	99.3	84.6	.03125	1.10	32.00	0.909	32.0	186.5	154.5	.065	.365
60	107.1	92.4	.03150	1.01	31.75	0.990	35.0	188.0	153.0	.070	.364
65	115.4	100.7	.03174	0.945	31.50	1.06	38.0	189.5	151.5	.076	.364
70	124.0	109.3	.03201	0.883	31.24	1.13	41.0	190.5	149.5	.082	.364
75	133.2	118.5	.03229	0.825	30.97	1.21	44.0	192.0	148.0	.088	.364
80	142.8	128.1	.03257	0.770	30.70	1.30	47.5	193.5	146.0	.093	.364
85	153.1	138.4	.03287	0.722	30.42	1.39	50.5	195.0	144.5	.099	.364
86	155.3	140.5	.03292	0.717	30.38	1.40	51.0	195.5	144.0	.100	.364
90	164.0	149.0	.03317	0.673	30.15	1.49	54.0	196.5	142.5	.105	.364
95	175.0	160.0	.03348	0.632	29.87	1.58	57.0	197.5	140.5	.111	.364
100	187.0	172.0	.03381	0.591	29.58	1.69	60.5	199.0	138.5	.116	.363
105	200.0	185.0	.03416	0.553	29.27	1.81	63.5	200.0	136.5	.122	.363
110	212.0	197.0	.03481	0.519	28.85	1.96	67.0	201.0	134.0	.128	.363
115	226.0	211.0	.03493	0.488	28.63	2.05	70.5	202.0	131.5	.134	.363
120	240.0	225.0	.03534	0.459	28.30	2.18	73.5	202.5	129.0	.140	.363
125	254.0	239.0	.03575	0.432	27.97	2.31	77.0	203.5	126.5	.145	.362

[†] *Am. Soc. Refrigerating Eng., Circ.* 9, by permission.

[*] In. Hg below 1 atm. (29.92 in.).

Table 30. Properties of Butane†

Temp., °F.	Pressure, lb. per sq. in.		Specific volume, cu. ft. per lb.		Density, lb. per cu. ft.		Heat content, B.t.u. per lb.		Latent heat, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	
	Absolute	Gage	Liquid	Vapor	Liquid	Vapor	Liquid	Vapor		Liquid	Vapor
0	7.3	*15.0	0.02591	11.1	38.59	0.0901	0	170.5	170.5	0.000	0.370
5	8.2	*13.2	.02603	9.98	38.41	.100	2.5	172.0	169.5	.006	.370
10	9.2	*11.1	.02615	8.95	38.24	.112	5.5	174.0	168.5	.011	.370
15	10.4	* 8.8	.02627	8.05	38.07	.124	8.0	176.0	168.0	.017	.370
20	11.6	* 6.3	.02639	7.23	37.89	.138	10.5	177.5	167.0	.022	.370
25	13.0	* 3.6	.02651	6.55	37.72	.153	13.0	179.0	166.0	.028	.371
30	14.4	* 0.6	.02664	5.90	37.54	.169	16.0	181.5	165.5	.033	.371
35	16.0	1.3	.02676	5.37	37.37	.186	19.0	183.5	164.5	.039	.371
40	17.7	3.0	.02689	4.88	37.19	.205	21.5	185.0	163.5	.044	.371
45	19.6	4.9	.02703	4.47	37.00	.224	24.5	187.0	162.5	.050	.372
50	21.6	6.9	.02716	4.07	36.82	.246	27.0	188.5	161.5	.056	.373
55	23.8	9.1	.02730	3.73	36.63	.268	30.0	190.5	160.5	.061	.373
60	26.3	11.6	.02743	3.40	36.45	.294	33.0	192.5	159.5	.067	.374
65	28.9	14.2	.02759	3.12	36.24	.321	36.0	194.5	158.5	.072	.374
70	31.6	16.9	.02773	2.88	36.06	.347	38.5	196.0	157.5	.078	.375
75	34.5	19.8	.02789	2.65	35.86	.377	41.5	198.0	156.5	.083	.375
80	37.6	22.9	.02805	2.46	35.65	.407	44.5	199.5	155.0	.089	.376
85	40.9	24.2	.02821	2.28	35.45	.439	47.5	201.5	154.0	.094	.376
86	41.6	26.9	.02825	2.24	35.40	.446	48.5	202.0	153.5	.095	.376
90	44.5	29.8	.02838	2.10	35.24	.476	51.0	203.0	152.0	.100	.377
95	48.2	33.5	.02854	1.96	35.04	.510	54.0	205.0	151.0	.105	.377
100	52.2	37.5	.02870	1.81	34.84	.552	57.0	206.5	149.5	.111	.378
105	56.4	41.7	.02889	1.70	34.62	.588	60.5	208.5	148.0	.117	.380
110	60.8	46.1	.02906	1.58	34.41	.633	63.5	210.5	147.0	.122	.380
115	65.6	50.9	.02925	1.48	34.19	.676	66.5	212.0	145.5	.128	.381
120	70.8	56.1	.02945	1.38	33.96	.725	70.0	213.5	143.5	.134	.382
125	76.0	61.3	.02966	1.30	33.72	.769	73.5	215.5	142.0	.139	.382
130	81.4	66.7	.02986	1.21	33.49	.826	76.5	217.0	140.5	.145	.384
135	87.0	72.3	.03009	1.14	33.23	.877	80.0	219.0	139.0	.151	.385
140	92.6	77.9	.03032	1.07	32.98	.934	83.5	221.0	137.5	.157	.386

† *Am. Soc. Refrigerating Eng., Circ.* 9, by permission.

* In. Hg below 1 atm. (29.92 in.).

Table 31. Properties of Isobutane†

Temp., °F.	Pressure, lb. per sq. in.		Specific volume, cu. ft. per lb.		Density, lb. per cu. ft.		Heat content, B.t.u. per lb.		Latent heat, B.t.u. per lb.	Entropy, B.t.u. per lb. per °F.	
	Absolute	Gage	Liquid	Vapor	Liquid	Vapor	Liquid	Vapor		Liquid	Vapor
−20	7.50	*14.6	0.02610	10.5	38.35	0.0952	−9.0	156.5	165.5	−0.020	0.356
−15	8.30	*13.0	.02620	9.90	38.15	0.101	−7.0	157.0	164.0	− .015	.354
−10	9.28	*11.1	.02635	8.91	37.95	0.112	−4.5	158.5	163.0	− .010	.353
− 5	10.4	* 8.8	.02645	7.99	37.80	0.125	−2.5	159.5	162.0	− .005	.351
0	11.6	* 6.3	.02660	7.17	37.60	0.139	0	160.5	160.5	.000	.350
+ 5	13.1	* 3.3	.02675	6.41	37.40	0.156	+2.5	162.0	159.5	.005	.348
10	14.6	* 0.2	.02690	5.75	37.20	0.174	5.0	163.5	158.5	.011	.348
15	16.3	1.6	.02705	5.18	37.00	0.193	7.5	164.5	157.0	.016	.347
20	18.2	3.5	.02715	4.68	36.80	0.214	10.0	166.0	156.0	.021	.346
25	20.2	5.5	.02730	4.24	36.60	0.236	13.0	167.5	154.5	.027	.346
30	22.3	7.6	.02745	3.86	36.40	0.259	15.5	169.0	153.5	.032	.346
35	24.6	9.9	.02760	3.52	36.20	0.284	18.0	170.5	152.5	.038	.346
40	26.9	12.2	.02780	3.22	36.00	0.311	21.0	172.0	151.0	.044	.346
45	29.5	14.8	.02795	2.96	35.80	0.338	24.0	174.0	150.0	.049	.346
50	32.5	17.8	.02810	2.71	35.60	0.369	27.0	175.5	148.5	.055	.346
55	35.5	20.8	.02825	2.49	35.40	0.402	30.0	177.5	147.5	.061	.347
60	38.7	24.0	.02840	2.28	35.20	0.439	33.0	179.0	146.0	.067	.348
65	42.2	27.5	.02855	2.10	35.00	0.476	36.5	181.0	144.5	.073	.349
70	45.8	31.1	.02875	1.94	34.80	0.515	39.5	183.0	143.5	.079	.350
75	49.7	35.0	.02890	1.79	34.60	0.559	43.0	185.0	142.0	.086	.351
80	53.9	39.2	.02910	1.66	34.35	0.602	46.5	187.0	140.5	.092	.352
85	58.6	43.9	.02930	1.54	34.10	0.649	50.0	189.0	139.0	.098	.353
86	59.5	44.8	.02935	1.52	34.10	0.658	50.5	189.5	139.0	.099	.354
90	63.3	48.6	.02950	1.42	33.90	0.704	53.5	191.0	137.5	.105	.356
95	68.4	53.7	.02965	1.32	33.70	0.758	57.5	193.5	136.0	.112	.358
100	73.7	59.0	.02990	1.23	33.45	0.813	61.0	195.5	134.5	.118	.359
105	79.3	64.6	.03005	1.14	33.25	0.877	65.0	198.0	133.0	.125	.360
110	85.1	70.4	.03030	1.07	33.00	0.935	69.0	200.0	131.0	.132	.362
115	91.4	76.7	.03050	0.990	32.80	1.01	73.0	202.5	129.5	.139	.364
120	98.0	83.3	.03075	0.926	32.50	1.08	77.0	204.5	127.5	.147	.367
125	104.8	90.1	.03095	0.867	32.30	1.15	81.5	207.5	126.0	.154	.369
130	112.0	97.3	.03125	0.811	32.00	1.23	86.0	210.0	124.0	.161	.371
135	119.3	104.6	.03145	0.760	31.80	1.32	90.5	212.5	122.0	.169	.375
140	126.8	112.1	.03175	0.716	31.80	1.32	95.0	215.5	120.5	.176	.377

† *Am. Soc. Refrigerating Eng., Circ.* 9, by permission.

* In. Hg below 1 atm. (29.92 in.).

Table 32. Ethane (C_2H_6), Saturated Vapor[1]

Temp., °F.	Pressure		Volume		Density		Heat content, above −40°F.			Entropy, from −40°F.	
	Abs., lb. per sq. in.	Gage, lb. per sq. in.	Liquid, cu. ft. per lb.	Vapor, cu. ft. per lb.	Liquid, lb. per cu. ft.	Vapor, lb. per cu. ft.	Liquid, B.t.u. per lb.	Latent, B.t.u. per lb.	Vapor, B.t.u. per lb.	Liquid, B.t.u. per lb. per °F.	Vapor, B.t.u. per lb. per °F.
−150	7.0	15.6*	0.02849	16.7	35.10	0.060	....	242			
−145	8.0	13.6*	.02865	14.1	34.90	0.071	....	240			
−140	9.7	10.1*	.02888	12.1	34.63	0.083	....	238			
−135	11.2	7.1*	.02901	10.5	34.47	0.095	....	236			
−130	13.2	3.0*	.02924	8.85	34.20	0.113	....	235			
−125	15.5	0.8	.02939	7.69	34.02	0.130	....	233			
−120	18.2	3.5	.02961	6.89	33.77	0.145	....	231			
−115	21.4	6.7	.02976	5.88	33.60	0.170	....	229			
−110	24.8	10.1	.03001	5.27	33.32	0.190	....	227			
−105	28.5	13.8	.03018	4.55	33.13	0.220	....	225			
−100	32.4	17.7	.0305	4.13	32.8	0.242	....	224			
−95	36.4	21.7	.0307	3.57	32.6	0.280	....	222			
−90	41.0	26.3	.0309	3.23	32.4	0.310	....	220			
−85	46.0	31.3	.0311	2.86	32.2	0.350	....	218			
−80	51.2	36.5	.0313	2.56	31.9	0.390	....	216			
−75	56.8	42.1	.0315	2.35	31.7	0.425	....	214			
−70	63.0	48.3	.0318	2.10	31.5	0.477	....	212			
−65	70.3	55.6	.0320	1.94	31.3	0.515	....	210			
−60	78.2	63.5	.0322	1.75	31.0	0.570	....	208			
−55	80.6	75.9	.0325	1.63	30.8	0.615	....	206			
−50	95.9	81.2	.0327	1.50	30.5	0.666	....	204			
−45	105.0	90.3	.0330	1.39	30.3	0.720	....	201			
−40	114.5	99.8	.0333	1.28	30.0	0.780	....	199			
−35	124.5	109.8	.0336	1.18	29.8	0.845	....	196			
−30	135.0	120.3	.0339	1.13	29.5	0.875	....	194			
−25	146.7	132.0	.0342	1.05	29.2	0.950	....	192			
−20	159.5	144.8	.0345	0.976	28.9	1.03	....	190			
−15	172	157	.0350	0.855	28.6	1.17	....	187			
−10	187	172	.0353	0.819	28.3	1.22	....	185			
−5	202	187	.0357	0.730	28.0	1.37	....	182			
0	219	204	.0361	0.689	27.7	1.45	....	179			
+5‡	236	221	.0365	0.629	27.4	1.59	....	176			
10	254	239	.0370	0.581	27.0	1.72	....	174			
15	272	257	.0375	0.538	26.7	1.86	....	171			
20	292	277	.0379	0.495	26.3	2.02	....	168			
25	307	292	.0385	0.457	26.0	2.19	....	165			
30	335	320	.0390	0.422	25.6	2.37	....	162			
35	358	343	.0397	0.389	25.2	2.57	....	158			
40	383	368	.0403	0.360	24.8	2.78	....	155			
45	405	390	.0410	0.330	24.4	3.03	....	150			
50	428	413	.0417	0.305	24.0	3.28	....	146			
55	453	438	.0426	0.279	23.5	3.58	....	141			
60	481	466	.0435	0.256	23.0	3.90	....	136			
65	511	496	.0444	0.238	22.5	4.20	....	130			
70	543	528	.0461	0.214	21.7	4.67	....	124			
75	584	569	.0478	0.182	20.9	5.50	....	115			
80	625	610	.0508	0.163	19.7	6.14	....	107			
85	672	657	.0549	0.128	18.2	7.80	....	78			
86‡	681	666	.0569	0.122	17.8	8.2	....	70			
89.8	718	703	.0775	0.0775	12.9	12.9	....	0			

---

[1] Linde Air Products Company, Compilers.

Pressure: 7 to 32 lb. abs., Maass and Wright;[1] 31 to 347 lb. abs., Kuenen and Robson;[2] 162 to 734 lb. abs., A. Heinlen.[3]

Density:† −162 to −101°F. (experimental data on liquid), Maass and Wright;[4] remainder of liquid, vapor, and latent heat data (calculated), Linde Air Products Co. Laboratory.

[1] *J. Am. Chem. Soc.*, **43**, 1098 (1921). [2] *Phil. Mag.*, (6) **3**, 149 (1902). [3] *Lieb. Ann.*, **282**, 229 (1894). [4] *J. Am. Chem. Soc.*, p. 1104 (1921).

* In. Hg below 1 atm. (29.92 in. = 14,696 lb. per sq. in. abs.).

† Probable accuracy: Density, liquid 1, vapor 3, latent heat 10 per cent.

‡ Standard ton temperatures.

Table 28. Methyl Bromide (CH_3Br)†

Temperature		Pressure			.emperature		Pressure			General properties
°F.	°C.	Atmos., sq. in.	Abs., lb. per sq. in.	Gage, lb. per sq. in.	°F.	°C.	Atmos., sq. in.	Abs., lb. per sq. in.	Gage, lb. per sq. in.	
− 4	−20	0.35	5.15	*19.43	86	30	2.40	35.28	20.58	Specific gravity (at 32°F.)........... 1.732
+14	−10	0.55	8.09	*13.45	104	40	3.44	45.56	30.86	Specific heat........................ 0.2
32	0	0.82	12.05	*5.39	140	60	5.90	86.75	72.05	Specific volume (at 32°F.).. 4.6 cu. ft. per lb.
50	10	1.20	17.64	2.94	167	75	8.40	148.68	133.98	Boiling point (at 760 mm.)........ 40.1°F.
68	20	1.75	25.43	10.73	212	100	14.50	213.15	198.45	Latent heat (at 40.1°F.)... 106 B.t.u. per lb.

† *Am. Soc. Refrigerating Eng., Circ.* 9, by permission.

* In. Hg below 1 atm.

Table 34. Nitrous Oxide (N_2O)†

Temp., °F.	Pressure		Volume		Density		Heat latent, B.t.u. per lb.
	Abs., lb. per sq. in.	Gage, lb. per sq. in.	Liquid, cu. ft. per lb.	Vapor, cu. ft. per lb.	Liquid, lb. per cu. ft.	Vapor, lb. per cu. ft.	
−130	14.2	0.5	0.01232	5.940	81.17	0.165	172.3
−121	19.6	4.9	.01248	4.370	80.13	.23	168.9
−112	26.8	12.0	.01264	3.200	79.11	.30	165.6
−103	35.5	20.8	.01280	2.480	78.12	.40	162.3
−94	47.3	32.6	.01296	1.880	77.16	.53	158.9
−85	59.6	44.9	.01312	1.510	76.22	.66	155.0
−76	75.0	60.3	.01314	1.220	76.10	.82	150.7
−67	92.3	77.6	.01376	0.9990	72.67	1.00	148.9
−58	113.0	98.3	.01408	0.8270	71.02	1.20	145.8
−49	135.0	120.3	.01440	0.6960	69.44	1.40	142.5
−40	160.0	145.3	.01472	0.6000	67.93	1.65	139.1
−31	190.0	175.3	.01504	0.5120	66.49	1.95	135.6
−22	223.0	208.3	.01536	0.4430	65.10	2.25	132.3
−13	257.0	242.3	.01568	0.3950	63.77	2.50	129.0
−4	295.0	280.3	.01600	0.3470	62.50	2.85	125.2
+5*	333.0	318.3	.01632	0.3080	61.27	3.25	121.4
14	375.0	360.3	.01680	0.2690	59.52	3.70	116.8
23	422.0	405.3	.01728	0.2340	57.87	4.25	111.9
32	471.0	456.3	.01776	0.2017	56.30	4.95	107.5
41	528.0	513.3	.01845	0.1744	54.19	5.70	103.2
50	592.0	577.3	.01920	0.1496	52.08	6.65	95.8
59	663.0	648.3	.02016	0.1276	46.73	7.80	88.2
68	745.0	730.3	.02140	0.1076	45.60	9.30	73.6
77	832.0	817.3	.02300	0.0896	43.48	11.20	66.9
†86	930.0	915.3	.02560	0.0726	39.06	13.80	51.1
95	1035.0	1020.3	.03136	0.0634	31.88	18.70	24.4
96	1055.0	1040.3	.03498	0.0537	[illegible].58	23.50	13.2
97	1065.0	1040.3	.04080	0.0408	24.51	24.50	0.0

---

† *Am. Soc. Refrigerating Eng., Circ.* 9, by permission.

* Standard ton temperatures.

Table 35. Ethyl Ether [$(C_2H_5)_2O$]†

Temp., °F.	Pressure		Volume of vapor, cu. ft. per lb.	Density of vapor, cu. ft. per lb.	Heat content above 32°F.		
	Abs., lb. per sq. in.	Gage, in. vac.			Liquid, B.t.u. per lb.	Latent, B.t.u. per lb.	Vapor, B.t.u. per lb.
0	1.3	27.28	38.0	0.0263	−18.00	171.0	153.00
*5	1.5	26.87	35.0	.0285	−15.00	170.8	155.80
10	1.8	26.26	32.5	.0332	−12.00	170.4	158.43
15	2.2	25.46	30.0	.0352	−9.50	170.2	161.70
20	2.5	24.84	27.0	.0372	−6.50	170.0	163.50
25	2.9	24.03	24.3	.0417	−4.00	169.6	165.60
30	3.4	23.00	21.4	.0468	−1.50	169.4	167.90
35	3.9	22.00	19.3	.0518	+1.40	168.8	170.20
40	4.4	21.09	17.0	.0588	4.00	168.4	172.40
45	4.9	19.97	15.0	.0666	6.60	168.0	174.60
50	5.5	18.72	13.2	.0757	9.57	167.6	177.17
70	8.8	12.05	7.8	.1280	20.04	165.4	185.44
75	9.8	10.02	7.0	.1430	23.40	164.8	188.20
80	10.9	7.33	6.2	.1620	26.40	164.2	190.60
85	12.2	5.09	5.5	.1860	29.00	163.8	192.80
86*	12.3	4.62	5.4	.1880	29.50	163.5	193.00
90	13.4	2.72	5.1	.1960	31.50	163.0	194.50
95	14.7	0.00	4.8	.2130	34.00	162.2	196.20
100	16.0	1.3†	4.5	.2220	36.50	161.5	197.50

* Standard ton-temperatures.

† *Am. Soc. Refrig. Eng., Circ.* 9, by permission.

CONDENSERS

The primary function of the condenser is to transfer to the environment (usually water or air) the heat extracted at the lower temperature. If c is the specific heat of water (or air), W the weight of water sent through the condenser per minute, Δt the temperature rise in degrees Fahrenheit, and n the number of tons refrigeration produced, then

$$W = \frac{200n}{c\Delta t}$$

Usually Δt will not be more than 30°F., from which it follows that W is approximately 7 lb. per min. per ton refrigeration or about 1100 to 1200 gal. per day per ton refrigeration. In considering any refrigeration installation it is always important to determine whether the water requirements can be met from the available sources. In large metropolitan centers this may often be a serious problem. The question of cooling-water economy is discussed by Sherwood (*Refrig. Eng.*, February, 1927) and Waterfill (*op. cit.*, September, 1927).

Types of Condensers. Condensers may be divided into the following types: (1) submerged coil; (2) atmospheric; (3) atmospheric counterflow; (4) double pipe; (5) multicoil; (6) flooded; (7) vertical shell and tube; (8) horizontal shell and tube.

Submerged-coil Condensers. The early condensers consisted of pipe coils placed in a tank. Ammonia flowed downward through the coil while water was introduced at the lower part of the tank and overflowed through a connection near the tank top. This was a cheap condenser to build but had several objectionable features. The most serious fault was that only a small part of the water touched the pipe coil. As a consequence, the amount of heat removed per gallon of water was very low, and to obtain a given capacity a large condenser tank was necessary. In addition the water tended to remain stagnant along the coil, and this interfered with the transfer of heat from the ammonia vapor to the coil, so that 30 B.t.u./(sq. ft.)/(deg. F.) (per hr.) was about the maximum rate.

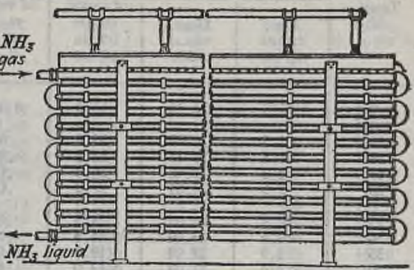


FIG. 17.—Atmospheric condenser.

Atmospheric Condensers. Atmospheric condensers are made of a vertical row of horizontal pipe lengths, from 8 to 20 in number, with the ends connected by return bends forming a continuous pipe through which the ammonia passes. Ammonia enters at the top from a header connecting two or more of the rows of pipes and flows downward, losing its heat to the water. Liquid ammonia settles in the bottom coils and passes off to the receiver or storage tank. Water is fed into a trough placed over the top pipe and, overflowing, drops into the top pipe and thence successively flows over each pipe into a basin beneath the coil. This type is illustrated in Fig. 17. It will be noted that both the ammonia and water flows are cross current. The condensed ammonia cannot be lower in temperature than the temperature of the discharged water.

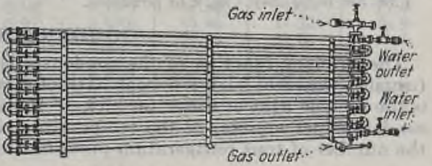


FIG. 18.—Double-pipe condenser.

Bottom-inlet Atmospheric Condensers. In bottom-inlet condensers the flow is made countercurrent by feeding the vapor at the bottom and by taking liquid off at the top, but, unless bleeder connections are taken off every few coils, the liquid tends to flow back toward the bottom. The condition prevails in those condensers where, after passing upward through the two lower pipes, the ammonia vapor is conveyed by an outside connection to the top pipe and then flows downward through the remainder of the bank. Both designs are erratic in operation and are subject to flooding and slugging.

Countercurrent Atmospheric Condensers. As stated above, the coldest water in the ordinary atmospheric condenser is at the top in contact with the hottest gases, while the hottest water at the bottom is in contact with the coldest ammonia. To reduce the ammonia temperature on leaving the condenser, condensers are designed with the gas inlet at the bottom where it meets the hottest water and the liquid discharge higher where the water is cooler. In its passage upward through the coils the condensing ammonia will have a tendency to trickle downward against the entering vapor. To prevent the bottom coils from filling up with liquid, trap drains are provided at several points.

Double-pipe Condensers. The desirability of having a condenser suitable for locations where the splashing of water by an atmospheric condenser could not be permitted led to the use of the double-pipe condenser, Fig. 18. In this design, two pipes are placed one within the other. The inner one is usually 1¼ in. in diameter and is filled with cooling water. The outer pipe is 2 in. in diameter, and the ammonia gas flows between the inner and outer pipes. Suitable connections permit the water to flow from one inner pipe to the next upper one, while at the same time the ammonia gas can pass in the opposite direction. The inner pipe extends through the fitting and through two stuffing boxes which seal the connection against leakage.

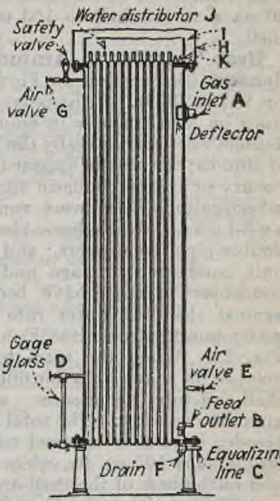


Fig. 19.—Vertical shell-and-tube condenser.

Shell-and-tube Condensers. A type of construction of the vertical single-pass multitube condenser is illustrated in Fig. 19. This consists of a welded shell with flared ends riveted to heavy tube sheets and a number of charcoal-iron tubes. The ammonia gas inlet is at the point *A*. In order to provide for a more even distribution of the gas at the inlet, a special deflector is attached to the inlet nozzle as shown. The ammonia is condensed on the tubes, collects in the bottom part of the shell, and is drained off at the feed outlet *B*. An equalizing line is provided at *C*. A drain is installed at *F*, while purge connections may be made at points *E* and *F*. A circular water box *H* is attached to the top sheet, and a special water baffle *I*, having serrated edges *K*, is placed within the water box as shown. Water-distributing devices *J* are placed on the top end of each tube. These distributors are made of cast iron and are hollow and with spiral grooves, and cause the water to flow in a corkscrew motion down the tube. A certain amount of air is drawn in through the hollow cores of the device. The vertical shell-and-tube condenser has the advantage that a greater amount of cooling surface can be obtained for a given floor space than with any other type.

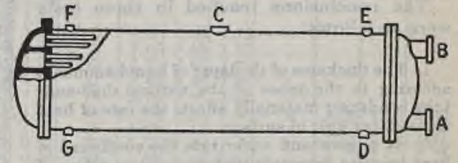


Fig. 20.—Horizontal multipass condenser. (*Courtesy Struthers Wells.*)

Horizontal Multipass Condenser. A construction of a horizontal multipass condenser is illustrated in Fig. 20. Water enters at the inlet *A* and passes through a bank of 1¼-in. tubes in the lower part of the shell, then flows backward and forward several times through the tubes in the upper part and finally out through the exit *B*. Refrigerant vapors enter at the inlet *C* placed on the top of the shell near the middle. The vapors condense, and the condensate is removed at *D* where it meets the coldest cooling water. A purge for non-condensable gases is provided at E. A safety valve is installed at F, and a drain valve at G. The number of passes depends on the diameter of the shell, being 2 to 6 on a 12-in. shell and 8 to 12 on a 24-in. shell. The total number of tubes varies from about 30 on a 12-in. shell to 160 on a 24-in. shell.

Heat Transfer in Ammonia Condensers (See section on Flow of Heat, pp. 947 to 1029). Exhaustive investigations on heat transfer in ammonia condensers were carried on by the University of Illinois; the results appear in the University of Illinois Bulletin 25. In these investigations tests were run on shell-and-tube, atmospheric-bleeder, and double-pipe condensers, and both the unit condenser tonnage and the total condenser tonnage have been plotted against the unit water rate for initial water temperature of 68°F. as shown in Fig. 21. When the total tonnage developed is taken into consideration the shell-and-tube condenser shows the greatest capacity. The total area of the bleeder condenser exposed to saturated ammonia is 105 sq. ft., while that of the double-pipe condenser is 92 sq. ft. and in the case of the shell-and-tube condenser it is 251 sq. ft.

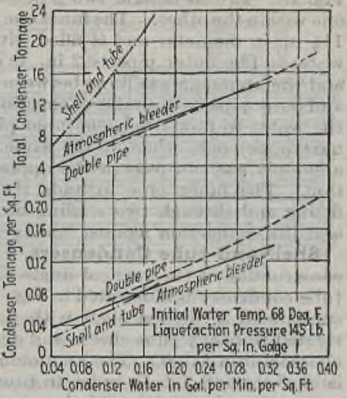


FIG. 21.—Comparative performances of various condensers.

As reported by the University of Illinois Bulletin 186, the coefficient of heat transfer of a 16-ft. vertical shell-and-tube condenser having thirty 2-in. tubes is found to vary with the water velocity, or water flow, as shown in Fig. 22 and Table 36. The variations in heat transfer with varying water flows and tonnages are shown in Fig. 23.

The conclusions reached in these tests were as follows:

1. The thickness of the layer of liquid ammonia adhering to the tubes of the vertical shell-and-tube condenser materially affects the rate of heat transfer per unit of surface.

2. At a constant water rate the coefficient of heat transfer decreases with increasing values of the mean temperature difference between the ammonia and the water in the vertical shell-and-tube condenser.

3. Irrespective of the proportions of the vertical shell-and-tube condenser no appreciable subcooling of the liquid occurs.

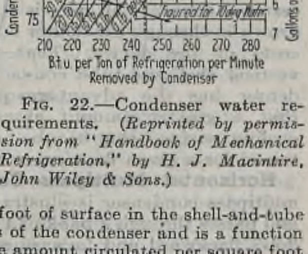


FIG. 22.—Condenser water requirements. (*Reprinted by permission from "Handbook of Mechanical Refrigeration," by H. J. Macintire, John Wiley & Sons.*)

4. The condenser tonnage developed per square foot of surface in the shell-and-tube condenser is independent of the size or proportions of the condenser and is a function only of the initial temperature of the water and the amount circulated per square foot of surface per unit of time.

5. The condenser tonnage developed per square foot of surface in the condensers when the effective surface is altered by raising the liquid level or by plugging pumps, as

was done in the investigation, is approximately independent of the proportions or arrangement of surface if a given amount of water at a given initial temperature is circulated per square foot of surface in a given time.

6. One square foot of surface is approximately eight times as effective in transferring heat from saturated ammonia vapor to water as it is in transferring heat from superheated ammonia gas to water.*

7. For conditions of viscous flow on the water side, the coefficient of heat transfer in a superheat remover is a linear function of the water velocity.*

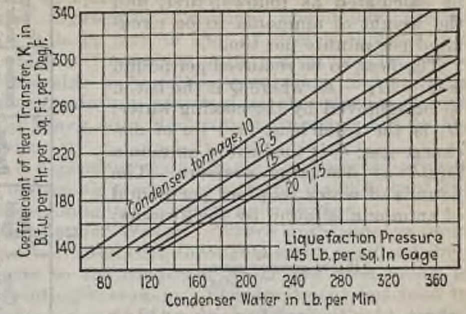


FIG. 23.—Variation of heat transfer with water rate for shell-and-tube condensers.

Condenser Water. The amount of water required in the condenser depends upon the quantity of heat which must be removed

Table 36. Gallons Condenser Water per Ton Refrigeration

Condenser pressure, lb. per sq. in. gage	Corresponding temp., °F.	60°F. water				70°F. water				80°F. water			
		Range, °F.	Water per ton of refrigeration, gal. per min. Suction pressure, lb. gage			Range, °F.	Water per ton of refrigeration, gal. per min. Suction pressure, lb. gage			Range, °F.	Water per ton of refrigeration gal. per min. Suction pressure, lb. gage		
			15	20	25		15	20	25		15	20	25
126.4	75	10	2.90	2.85	2.80								
131.4	77	12	2.40	2.35	2.30								
136.6	79	14	2.05	2.00	1.95								
141.8	81	16	1.85	1.80	1.75	6	4.87	4.80	4.70				
147.2	83	18	1.65	1.60	1.55	8	3.67	3.60	3.55				
152.7	85	20	1.47	1.45	1.41	10	2.95	2.90	2.85				
158.3	87	22	1.35	1.32	1.30	12	2.45	2.40	2.35				
164.1	89	24	1.25	1.22	1.20	14	2.10	2.05	2.00				
170.1	91	26	1.15	1.13	1.10	16	1.90	1.85	1.80	6	5.00	4.90	4.80
176.2	93	28	1.08	1.05	1.03	18	1.68	1.63	1.60	8	3.75	3.70	3.65
182.6	95	30	1.00	0.99	0.97	20	1.50	1.48	1.45	10	3.00	2.95	2.90
189.1	97	32	0.95	0.93	0.90	22	1.38	1.35	1.32	12	2.52	2.48	2.43
195.7	99	..	....	....	....	24	1.28	1.25	1.22	14	2.13	2.10	2.06
202.5	101	..	....	....	....	26	1.17	1.15	1.13	16	1.92	1.85	1.83
209.5	103	..	....	....	....	28	1.10	1.07	1.05	18	1.70	1.67	1.63
216.5	105	..	....	....	....	30	1.02	1.00	0.98	20	1.53	1.50	1.48
223.7	107	..	....	....	....	32	0.95	0.93	0.92	22	1.40	1.37	1.35
231.1	109	..	....	....	....	..	....	....	....	24	1.30	1.28	1.25
238.7	111	..	....	....	....	..	....	....	....	26	1.20	1.16	1.15
246.5	113	..	....	....	....	..	....	....	....	28	1.12	1.08	1.06
254.5	115	..	....	....	....	..	....	....	....	30	1.05	1.02	1.00
262.7	117	..	....	....	....	..	....	....	....	32	1.00	0.98	0.95

---

* If the superheated vapor is condensing, the rate of heat transfer is probably as good as for saturated vapor. The above conclusions apply to the case in which superheated vapor goes to a lower degree of superheat.

to liquefy the ammonia. This may be calculated as follows: first, find the weight of ammonia to be circulated per minute per ton.

The heat to be removed per pound is $Q = H_s - h_l$, where Q is the B.t.u. to be removed by the cooling water; H_s is the total heat in 1 lb. of discharge gas; h_l is the heat of ammonia liquid at discharge pressure. The pounds of water required per pound of ammonia is found by the formula:

$$W = \frac{Q}{(t_o - t_i)c_w}$$

where t_o = temperature of outlet water.

t_i = temperature of inlet water.

c_w = specific heat of water = 1.

The condensing pressure depends on the temperatures and quantity of the cooling water. With 60°F. water in large quantities, it is possible to get a condensing pressure as low as 145 lb. per sq. in.; with 75°F. water, 165 lb. per sq. in.; with 85°F. water, 185 lb. per sq. in. In ordinary practice the pressure may be somewhat higher.

Table 37 gives the performances obtained on a multipass condenser.

The temperature of the condenser water should be as low as possible. If water costs are high, it may be more economical to provide cooling towers and use less water. Carbon dioxide condensers cannot operate with water above 88°F., the critical temperature. Before installing carbon dioxide machines, the maximum temperature of the available water should accordingly be ascertained.

Cooling-water Towers. (See Cooling Towers Sec. 9.) In most places water is too costly to be wasted after use over the condenser and is recovered either by a cooling tower or pond.

Towers are usually figured for a small range of water cooling, about 5°F. and are designed to handle ap-

Table 37. Condenser Performance

8-in. multitube, seven-pass, condenser*

Test No.	Date	Duration, hr.	Surface, sq. ft.	Temp. of water to condenser, °F.	Temp. of water from condenser, °F.	Total quantity of water circulated, gal.	Water circulated, gal. per min.	Total weight of ammonia liquefied, lb.	Weight of ammonia liquefied per min., lb.	Temp. of gas to condenser, °F.	Condenser pressure, lb. per sq. in., gage	Condensing temp., °F.	Temp. of liquid ammonia from condenser, °F.	Rating, tons refriger.	Surface per ton refriger., sq. ft.	Cooling water per ton refriger., gal. per min.
66	1-19-25	5	132	55.1	67.7	20,420	68.07	4,004	13.35	101.9	123.4	73.9	64.3	32.92	4.00	2.06
60	12-26-24	5	132	60.0	72.6	20,170	67.23	3,915	13.05	102.0	131.8	77.4	69.9	31.88	4.14	2.10
57	12-18-24	5	132	65.0	76.6	20,820	69.70	3,829	12.76	113.7	140.6	80.8	75.7	31.00	4.23	2.25
54	12-15-24	5	132	70.9	80.9	21,870	72.90	3,763	12.54	108.9	152.6	85.3	79.3	30.00	4.40	2.43
50	12- 9-24	5	132	75.3	84.6	23,480	78.26	3,363	11.21	102.5	160.6	88.18	82.3	27.00	4.88	2.90
44	10-16-24	5	132	80.4	88.8	24,150	80.50	2,412	10.19	108.7	170.9	91.7	86.8	24.00	5.50	3.35
47	11-12-24	5	132	84.4	92.0	24,470	81.60	2,843	9.48	107.6	177.7	94.0	90.7	22.04	6.00	3.70
65	1-15-25	5	132	90.1	96.1	24,650	82.20	2,490	8.30	107.9	193.3	98.8	93.8	19.40	6.72	4.24

* Tested by H. Vogt Machine Company.

proximately 6 gal. water per ton of refrigeration and cool this to within 5°F. of the wet-bulb temperature with an assumed air velocity of 5 m.p.h.

Corrosion in Condenser Systems. Ammonia condensers present a severe corrosion problem. A large area of iron or steel is exposed to a continuous flow of water, which is frequently quite corrosive. To obtain efficient heat transfer it is desirable to have the metal bare, and ordinary methods of protection, such as painting, are used at the expense of condensing efficiency. Similarly heavy rust films retard corrosion but to the detriment of operating efficiency.

Corrosion of steel in water is dependent upon a supply of dissolved oxygen. Except in closed condenser or cooling systems, where the water is kept out of contact with the air, it is uneconomical to remove the oxygen.

Silicate Treatment of Condenser Water. Where fresh water is recirculated, passing over the condensers and then through a spray pond or tower, the addition of sodium silicate to the condenser water has been found to be a practical means of preventing corrosion. This treatment has been in use in some plants and its behavior warrants continued trial.

COOLING SYSTEMS

Cooling Systems (see Cooling Towers, pp. 1108–1117). Two systems of removing heat from cold-storage rooms are used, the **direct-evaporation** and the **indirect** or **brine** systems.

With the **direct-expansion** system, where ammonia is allowed to boil in the cold-storage rooms, there is danger from ammonia leakage at all times. The pipe lines may become corroded or may split due to imperfect welding, or fittings may be broken accidentally with resulting damage to life or commodities. In a large system with long supply and return pipes, or extensive refrigerating piping, the amount of the initial charge has to be very large and constant care to maintain the piping tight at all times is required. The result is that, except under certain particular conditions, the so-called **brine system** is best. This has the further advantage that very appreciable amounts of refrigeration may be stored up in the brine to take up peak loads or provide for refrigeration during closing-down periods.

In the **brine system of indirect refrigeration,** the high-pressure side is the same as in the direct-expansion system. The low-pressure side consists of a brine cooler, usually of the shell-and-tube type, similar in construction to a steam condenser. The brine is a non-freezing solution of sodium chloride or calcium chloride, of such concentration as will not freeze at the temperature carried in the cooling system. The brine system then is really an additional unit, in which the brine is kept cool by boiling ammonia and the cold-storage rooms or other refrigerating applications are kept cold by the brine.

In the direct-expansion system one less heat transfer is necessary, and consequently the expansion coils can be maintained at a higher temperature and pressure so that the compression work is less. Moreover, the refrigerant extracts heat by evaporation (latent heat) so it can be distributed at room temperatures, whereas brine extracts heat only by being colder than the surroundings (sensible heat) and must be distributed cold in well-insulated piping. The direct-expansion system is therefore much more efficient in theory, and the only obstacle to its general adoption is the difficulty of constructing leakproof systems.

Evaporation or Expansion Coils.[*] Just as the function of the condenser is to dissipate heat, so the function of the evaporating coil is to collect heat

---

[*] For a more general discussion see Consley, Heat Transfer in Ammonia Shell-and-tube Brine Coolers as Affected by Operating Conditions, *Refrig. Eng.*, **35,** 409 (1938).

at the lower temperature of the refrigerating system. It is quite possible to inject the liquid refrigerant into the material to be cooled and then cause it to evaporate, but this is in general undesirable for two reasons: first, the refrigerant will have a deleterious effect on the substance cooled, and, second, its vapor pressure will be lower because of dilution and the power expenditure in compression will be greater. However, in case the refrigerant can also be used as a solvent, this system may prove very satisfactory, and one notable application is found in the Edeleanu process of extracting asphaltic materials from mineral oils with sulfur dioxide.

The simplest method of cooling brine is to thrust a pipe coil into the brine tank with one end connected to the liquid refrigerant supply and the other to the compressor. An older form of construction with a series of continuous coils attached to headers is shown in Fig. 24. Liquid ammonia is fed into the bottom and drawn off on top. Under these conditions liquid ammonia is present on the inside surface of the coils, and if this is accidentally drawn into the compressor it may fill up the clearance space in the cylinder and blow off the head on the compression stroke. This difficulty is overcome by placing an accumulator (shown by dotted lines in Fig. 25) between the expansion coil and the compressor. The accumulator may serve the further purpose of precooling the liquid ammonia returning to the expansion coil. This arrangement of coils with accumulator is known as the **flooded system.** In order to facilitate transfer of heat, brine tanks may be built with partitions and bulkheads so arranged that the brine may be agitated and swirled around and through the coils.

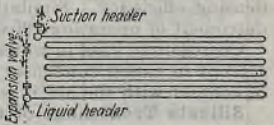


FIG. 24.—Continuous-coil evaporator.

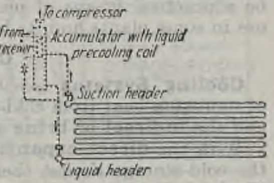


FIG. 25.—Evaporator coil with accumulator.

A modern short evaporator coil is shown in Fig. 26. This is sometimes made with three sets of pipes between the upper and lower headers and placed in a covered brine compartment. The herringbone coil, shown in Fig. 27, can be placed in a trough through which the brine is circulated at high velocity.

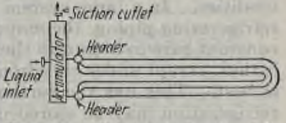


FIG. 26.—Short evaporator coil.

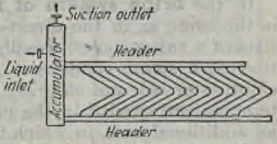


FIG. 27.—Herringbone coil.

Besides the **submerged** types of coils, which are used chiefly in ice making, shell-and-tube brine coolers, and double-pipe brine coolers with brine flowing through the inner tube are also used. Shell-and-tube coolers are largely displacing other types in more recent installations (Fig. 28). Brine flows through the tubes, and a surface of 8 to 15 sq. ft. is usually allowed per ton of refrigeration. They may be placed either upright or horizontal and require small floor space. A float valve can be placed in the liquid receiver to regulate the amount of ammonia fed to the expansion coils.

Heat Transfer in Evaporators. (See Section 7, pp. 947–1029.) The York Ice Machinery Corporation conducted a series of tests on heat transfer in various evaporators. For shell-and-tube evaporators with tube arrangements and liquid levels shown in Fig. 28 the results were as given in Table 38.

For short flooded coils, the results obtained are shown in Table 39.

Liquid Controls. In order to prevent evaporating coils from overflowing with liquid refrigerant, some form of automatic control valve is usually employed. These may be classified into four groups (see Sec. 17, pp. 2009–2089):

1. The diaphragm-operated pressure-reducing valves controlling the refrigerant feed to the evaporator according to the pressure in the evaporator.
2. Diaphragm-operated pressure-reducing valves controlling the refrigerant feed to the evaporator by means of the temperature of the refrigerant gas leaving the evaporator.
3. Thermostatically controlled shut-off valves actuated by change of temperature in room or tank and used in series with hand-adjusted expansion valves or valves under group 1 or 2.
4. Float-controlled valves which function according to the change in liquid refrigerant level in the evaporators, or used as liquid traps to pass the refrigerant from the high side to the evaporator as it is condensed.

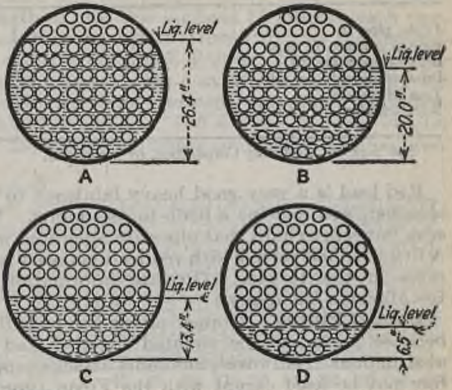


FIG. 28.—Liquid levels used in shell-and-tube coolers.

Brine Piping. Brine is seldom allowed to warm more than 20°F. in performing refrigeration, and the heat absorbed per pound is much smaller than that absorbed by the evaporation of an equal weight of refrigerant. Consequently brine piping is larger than refrigerant piping. Standard-weight piping is sufficient for all ordinary purposes.

Table 38. Results of Tests on Shell-and-Tube Cooler*

	17.9° brine in				34° brine in			
	A	B	C	D	A	B	C	D
Brine temp., °F.	17.6	17.9	17.9	17.8	34.3	34.0	33.9	33.6
Suction pressure at cooler, lb. gage	14.9	13.9	10.6	6.3	24.6	22.7	18.0	10.5
Superheat at cooler, °F.	3.4	7.3	13.3	23.0	3.5	7.8	14.5	28.7
Tons of refrigeration	61.4	59.0	51.3	41.0	79.2	75.5	66.2	49.8
Tube surface below liquid level, %.	87.5	62.5	37.5	12.5	87.5	62.5	37.5	12.5
Height of ammonia, in.	26.4	20.0	13.4	6.5	26.4	20.0	13.4	6.5
Mean temp. difference	15.5	17.3	22.7	30.6	19.3	21.3	27.5	38.9
B.t.u. transfer per sq. ft. per hr. per degree mean temp. difference	62.5	54.5	36.4	21.5	68.4	56.8	38.6	20.5
Brine velocity, ft. per min.	129	131	134	133	133	133	135	133

* The York Ice Machinery Corporation, by permission.

The same rule for threads applies as in any threaded pipe work. Threads should not be shouldered and should be entered three or four by hand. A 5-in. pipe ought to enter the fitting about ¾ or ⅞ in. and have two or three threads sticking out to make certain that it is not shouldered.

Table 39. Results with Short Flooded Coils*

	A	B	C	D
1¼-in. pipe, ft.	500	500	500	500
Suction pressure, lb. gage	19.6	21.2	18.9	21.5
Brine temperature, °F.	14.0	14.0	14.0	14.0
Brine velocity, ft. per min.	133	133	133	133
Tons refrigeration	7.3	9.0	12.0	15.7
B.t.u. transfer	43	68	66	120

* The York Ice Machinery Corporation, by permission.

Red lead is a very good heavy lubricant to be used for screwing the pipe together, and it helps a little to seal joints. White lead will make a better seal, but sets so hard that pipes are separated with difficulty and is little used. A little asphalt mixed with red lead makes a better joint in case of imperfections in the threads. (Private information from Torrance, Carbondale Ice Machine Company.)

Copper Piping. Copper piping has recently been introduced extensively because joints can be sweated together and can readily be bent around obstructions. However, ammonia attacks copper and such tubing can therefore not be used except with the Freon type of refrigerants or methyl and ethyl chlorides. Small commercial installations of 1 to 5 tons capacity are tending to go over to copper piping with these refrigerants because of the ease of installation.

Refrigeration Pipe Lines. There are two general systems of pipe-line refrigeration in use in the United States—those circulating ammonia and those circulating brine solutions as refrigerating mediums. The latter seems to be the preferred method. A brine system operated by the Merchants Refrigerating Company, New York, serving the dairy products, poultry, and produce trades, carries 400 refrigerator boxes, varying in size from 200 cu. ft. to 15,000 cu. ft. aggregating a total of 2,750,000 cu. ft. refrigerated space (Oakley, Am. Soc. Refrigerating Eng., annual meeting, 1924).

Brine is cooled to 0°F. by two horizontal multipass shell-and-tube brine coolers and is circulated through the system by means of four 12-in. mains, making a dual system on the trunk lines so that in the event of a break in one pair of trunk lines the other pair will carry the load. These mains are carried under the street at a depth of 4 to 7 ft. and are properly supported and insulated with three or more layers of 1-in. hair felt. The mains are full-weight wrought-iron pipe with flanged joints and ring gaskets, alternate joints being fitted with heavy cast-iron sleeves and calked with lead. Expansion joints of the corrugated type are introduced to take up the initial movement of the pipe. Great care must be exercised in construction to keep the losses at a minimum. The normal brine losses average about ½ bbl. a day, while the accidental losses over which the distributor has no control are about twice this amount. The specific gravity of the brine is maintained at about 1.23. The solution is tested several times a year and kept slightly alkaline. No trouble is experienced with deterioration of the mains, some of which have been in service over 30 years.

Centrifugal pumps are used to circulate the brine throughout the system, which during the summer conditions requires about 3000 gal. per min. giving a brine velocity in the mains of about 250 ft. per min. The pumps are designed to give maximum efficiency over a rather wide range of head to take care of increasing business and changing load.

In St. Louis and Los Angeles, liquid ammonia is distributed for refrigeration from a central compressor station, and the vapors drawn back by the compressor suction. The total length of pipe line is about 15 miles in each city.

In a brine-circulating system there are power losses due to the indirect method which are absent in a direct-expansion system, such as the lower operating back pressure to the compressor, the cost of brine pumping, and the consequent heat delivered to the brine through pumping. However, this is offset by features of safety, simplicity, and a more uniform suction pressure to the compressor by having the ammonia confined to bring coolers under immediate control.

The general application of pipe-line refrigeration is most successful in market districts where the demand for service is in confined territory. Power is the greatest item of expense, and, therefore, the cost per ton of refrigeration delivered to the system should be kept to a minimum (Fig. 29).

Pipe-line systems should be equipped with instruments to obtain data properly, and records should be kept of such items as tons of refrigeration delivered to the system, cost of power to deliver a ton, etc.

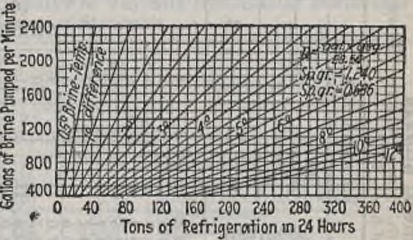


FIG. 29.—Brine flow required for varying tonnages of refrigeration.

Insulation Values of Building Materials (see Flow of Heat, Sec. 7, pp. 951*ff*. and 996).

An important factor in insulation is the perfection of construction, which is subject to wide variations. Unless the joints are carefully made and airtight, moisture will get in and freeze, causing the ultimate deterioration of the insulation. This is probably the factor most responsible for the divergence in laboratory experiments and practical results. A liberal factor of safety should be allowed to ensure good operating results.

It is customary to assume an average maximum temperature during the period of peak refrigerating loads. The average temperatures in the United States may be obtained from government reports. The average temperature experienced during a 24-hr. period for conditions that are likely to prevail for a week's time should be used in making estimates. The actual choice of the thickness of insulation for any particular case will be decided by the relative costs of the insulation itself and the book value of a ton of refrigeration.

The **standard insulation of the American Warehouse Association** is: For the walls, ceilings, floors, partitions, etc., of cold-storage buildings, the insulation consists of corkboard of good quality and of medium density with both outer surfaces sealed by either dipping or coating with an asphaltic mastic, or by applying a waterproof portland cement plaster as a finish. An additional ½ in. of waterproof portland cement is used between the layers of the cork to seal the voids between the cork granules against atmospheric or other moisture. The standard thickness for temperatures down to 32°F. is two layers of 2-in. corkboard; to this is added 1 in. for each 15°F. below 32°F. Piping, fittings, etc., should be covered with molded cork covering,

Table 40. Piping for General Storage

Cu. ft. of space cooled per linear foot of 1¼-in. and of 2-in. pipe. 15.67 lb. suction pressure. Temperature of ammonia expansion, 0°F. Mean temperature of brine, 10°F.

Size of room, cu. ft.	Room temp., 40°F.					Room temp., 36°F.					Room temp., 32°F.					Room temp., 28°F.				
	Tons refriger.	Direct expansion, in.		Brine, in.		Tons refriger.	Direct expansion, in.		Brine, in.		Tons refriger.	Direct expansion, in.		Brine, in.		Tons refriger.	Direct expansion, in.		Brine, in.	
		1¼	2	1¼	2		1¼	2	1¼	2		1¼	2	1¼	2		1¼	2	1¼	2
1,000	0.7	8	12	6	9	0.75	7	10	4.5	6	0.8	6	8	4	5	1.1	4	5	2	3
2,000	1.1	10	15	7	11	1.3	8	12	7	7	1.4	7	9	5	6	1.9	4.5	6	2.5	3.5
3,000	1.5	11	16	8	12	1.6	9	14	6	8	1.7	8	12	5.5	7	2.3	5	7	3	4
4,000	1.8	12	18	9	13	2.0	10	15	6.5	10	2.2	8.5	12.5	6	8	3.0	5.5	7.5	3.5	4.5
5,000	2.2	13	19	10	14	2.4	11	16	7	11	2.6	9	14	6.5	9	3.5	6	8	4	5
7,000	2.8	14	20	11	15	3.0	12	17	8	13	3.2	10	16	7	11	4.3	6.5	9	4.5	5.5
10,000	3.6	16	23	12	17	3.9	13	19	9	14	4.2	11	18	7.5	12	5.7	7	10	5	6
15,000	4.8	18	26	13	19	5.2	15	21	10	15	5.6	12	19	8	13	7.5	8	11	5.5	7
20,000	6.0	20	28	15	21	6.5	16	25	11	16	7.0	13	23	9	15	9.5	9	12	6	8
30,000	7.5	23	32	17	24	8.0	19	28	13	20	8.7	15	25	10	17	11.7	10	15	6.5	9
40,000	9.3	25	35	19	26	10.0	21	30	15	21	10.8	17	26	11	17.5	14.6	11	16	7	10
60,000	12.8	27	37	20	28	13.8	22	32	15.5	22	15.0	18	27	12	18	20.2	12	17	7.5	10.5
80,000	16.6	28	39	21	29	17.9	23	33	16	23	19.4	18.5	27.5	12.5	18.5	26.2	12.5	17.5	8	11
100,000	20.3	29	40	22	30	22.0	24	34	17	24	23.8	19	28	13	19	32.1	13	18	8.5	11.5

For other ammonia and brine temperatures multiply the number of feet of pipe found from the above table by the constants in the following table.

Suction pressure abs.	Ammonia temp., °F.	Brine temp., °F.	Room temperatures, °F.							
			Direct expansion				Brine			
			40	36	32	28	40	36	32	28
19.46	5	15	1.2	1.2	1.2	1.2	1.2	1.2	1.3	1.4
23.64	10	20	1.3	1.4	1.5	1.6	1.5	1.6	1.8	2.3
28.24	15	25	1.6	1.7	1.9	2.2	2.0	2.4	3.0	
32.25	20	30	2.0	2.3	2.7	3.5	3.0	4.3		

which should have its outer and inner surfaces sealed with a rubber or asphaltic mastic. Standard brine-pipe covering for temperatures down to 0°F. varies from 2 to 3 in. thickness with the diameter of the pipe. Table 40 (taken from Marks's Handbook, 4th ed., p. 2170) shows the cubic feet of space cooled per linear foot of pipe in general storage. Table 41 gives pertinent data on the thermal conductivities and densities of various insulating materials.

Pipe Lines in CO_2 Plants. The present tendency is to weld all connections in carbon dioxide plants, making provisions for expansion by means of bends. All pipe connections should be bent, wherever necessary, with a long radius. This gives the gas more chance to flow smoothly and minimizes vibration in the pipe lines. In the discharge lines the gas may attain a velocity of from 800 to 1,200 ft. per min. at a pressure of from 1000 to 1400 lb. Should this gas meet a right-angled bend, it would cause a severe shock, which would pass to the hangers and be transmitted to the building.

Table 41. Conductivity and Density of Various Insulating Materials*

U. S. Bureau of Standards

Material	Thermal conductivity, B.t.u./(sq. ft.) (in.) (hr. per °F.)	Density, lb. per cu. ft.	Description of material
Air	0.175	0.08	Ideal air space
Air cell, ½ in.	0.458	8.80	Asbestos paper and air spaces
Air cell, 1 in.	0.500	8.80	Asbestos paper and air spaces
Asbestos mill board	0.830	61.0	Pressed asbestos
Asbestos wood	3.700	123.0	Asbestos and cement
Balsawood	0.350	7.5	Light and soft across grain
Calorax	0.221	4.0	Fluffy, finely divided mineral matter
Cork	0.337	5.3	Granulated ⅛ to 3/16 in.
Cork	0.330	10.0	Regranulated 1/16 to ⅛ in.
Cork board	0.279	6.9	No artificial binder—low density
Cork board	0.308	11.3	No artificial binder—high density
Cotton wool	0.292	......	Loosely packed
Fibrofelt	0.329	11.3	Felted vegetable fibers
Fire-felt wool	0.625	43.0	Asbestos sheet coated with cement
Fire-felt sheet	0.583	26.0	Soft, flexible asbestos sheet
Flaxlinum	0.329	11.3	Felted vegetable fibers
Hair felt	0.246	17.0	
Hard maple wood	1.125	44.0	Across grain
Infusorial earth	0.583	43.0	Natural blocks
Insulite	0.296	11.9	Pressed wool pulp—rigid
Kapok	0.238	0.88	Vegetable fiber—loosely packed
Keystone hair	0.271	19.0	Hair felt combined with building paper
Linofelt	0.300	11.3	Vegetable fiber combined with paper
Lithboard	0.379	12.5	Mineral wool and vegetable fibers
Mineral wool	0.275	12.5	Medium packed
Mineral wool	0.288	18.0	Felted in blocks
Oak wood	1.000	38.0	Across grain
Planer shavings	0.417	8.8	Various
Pulp board	0.458	......	Stiff pasteboard
Pure wool	0.263	5.0	
Rock cork	0.346	21.0	Mineral wool and binder—rigid
Slag wool	0.750	15.0	
Tar roofing	0.707	55.0	
Virginia pine wood	0.958	34.0	Across grain
White pine wood	0.791	32.0	Across grain
Wool felt	0.363	21.0	Flexible paper stock

---

* See Physical and Chemical Data, Sec. 3, pp. 453–455.

Table 42. Heat Content of Calcium Chloride Brines, B.t.u. per Lb.

From value at 32°F., where it equals 200 B.t.u. minus the heat of solution

Temp., °F.	Specific gravity, 60°/60°F.												
	1.08	1.10	1.12	1.14	1.16	1.18	1.20	1.22	1.24	1.26	1.28	1.30	1.303
	Freezing point, °F.												
	22.5	19.1	15.1	10.6	5.4	−0.4	−6.4	−13.2	−20.7	−29.6	−41.0	−56.8	−59.8
	Lb. $CaCl_2$ per 100 lb. brine												
	9.2	11.4	13.6	15.7	17.8	19.8	21.8	23.7	25.6	27.5	29.4	31.2	31.4
−85	33.63	40.58	47.37	54.02	60.54	66.89	73.23	79.39	85.44	91.31	97.13	102.83	103.55
−80	35.47	42.40	49.17	55.80	62.30	68.63	74.95	81.09	87.12	92.97	98.77	104.46	105.18
−75	37.34	44.24	50.99	57.60	64.08	70.39	76.69	82.81	88.82	94.65	100.43	106.10	106.82
−70	39.23	46.11	52.83	59.42	65.87	72.16	78.44	84.54	90.54	96.35	102.11	107.76	108.47
−65	41.14	48.00	54.70	61.26	67.69	73.96	80.22	86.30	92.27	98.06	103.80	109.44	110.14
−60	43.07	49.91	56.59	63.13	69.54	75.78	82.01	88.07	94.02	99.79	105.52	111.13	111.83
−59.8°	43.15	49.98	56.65	63.19	69.61	75.85	82.08	88.13	94.08	99.85	105.58	111.20	111.90
−59.8°	60.60	71.56	82.30	92.82	103.13	113.19	123.16	132.88	142.43	151.73	160.93	169.94	171.10
−55	63.34	74.49	85.40	96.08	106.57	116.79	126.93	136.81	146.52	155.97	165.30	173.73	174.10
−50	66.25	77.60	88.67	99.56	110.23	120.63	130.94	140.99	150.87	160.49	169.97	176.86	177.23
−45	69.31	80.89	92.20	103.28	114.18	124.79	135.30	145.56	155.64	165.45	175.13	179.98	180.36
−40	72.57	84.40	95.96	107.32	118.44	129.30	140.04	150.53	160.82	170.86	180.24	183.12	183.50
−35	76.01	88.15	100.02	111.66	123.08	134.23	145.25	155.97	166.58	176.89	183.41	186.27	186.63
−30	79.71	92.21	104.43	116.39	128.15	139.62	150.96	162.04	172.90	183.49	186.59	189.41	189.77
−25	83.72	96.62	109.24	121.61	133.77	145.62	157.34	168.79	180.01	187.01	189.77	192.55	192.91
−20	88.07	101.50	114.63	127.49	140.13	152.45	164.64	176.55	187.57	190.24	192.95	195.71	196.05
−15	93.01	107.06	120.76	134.21	147.41	160.30	173.04	185.48	190.87	193.49	196.15	198.86	199.20
−10	98.64	113.44	127.91	142.11	156.04	169.63	183.06	191.60	194.19	196.73	199.35	202.01	202.35
− 5	105.07	120.88	136.36	151.53	166.42	180.94	192.45	195.00	197.51	199.99	202.55	205.18	205.51
0	112.60	129.62	146.29	162.62	178.67	193.48	195.95	198.40	200.84	203.26	205.77	208.34	208.67

Table 42. Heat Content of Calcium Chloride Brines, B.t.u. per Lb.—(*Concluded*)

Temp., °F.	Specific gravity, 60°/60°F.												
	1.08	1.10	1.12	1.14	1.16	1.18	1.20	1.22	1.24	1.26	1.28	1.30	1.303
	Freezing point, °F.												
	22.5	19.1	15.1	10.6	5.4	−0.4	−6.4	−13.2	−20.7	−29.6	−41.0	−56.8	−59.8
	Lb. $CaCl_2$ per 100 lb. brine												
	9.2	11.4	13.6	15.7	17.8	19.8	21.8	23.7	25.6	27.5	29.4	31.2	31.4
5	121.63	140.24	158.44	176.31	193.87	197.08	199.45	201.82	204.18	206.54	208.99	211.52	211.84
10	133.07	153.84	174.13	194.07	198.38	200.69	202.97	205.24	207.53	209.83	212.21	214.70	215.01
15	149.39	173.51	197.04	199.88	202.11	204.32	206.49	208.68	210.89	213.12	215.45	217.88	218.18
20	176.58	199.43	201.63	203.75	205.86	207.95	210.03	212.13	214.27	216.42	218.70	221.07	221.37
25	201.49	203.58	205.63	207.62	209.62	211.60	213.58	215.59	217.75	219.74	221.94	224.27	224.56
30	205.79	207.74	209.65	211.51	213.38	215.26	217.14	219.06	221.04	223.06	225.20	227.47	227.75
35	210.12	211.90	213.67	215.41	217.15	218.93	220.70	222.54	224.44	226.39	228.46	230.69	230.96
40	214.44	216.08	217.70	219.31	220.94	222.60	224.28	226.03	227.85	229.73	231.74	233.90	234.17
45	218.78	220.26	221.74	223.22	224.74	226.29	227.87	229.53	231.27	233.08	235.02	237.13	237.39
50	223.11	224.44	225.79	227.14	228.54	229.98	231.47	233.04	234.69	236.44	238.31	240.36	240.62
55	227.46	228.64	229.85	231.07	232.35	233.69	235.08	236.56	238.13	239.81	241.62	243.60	243.85
60	231.82	232.86	233.91	235.00	236.17	237.40	238.70	240.09	241.58	243.19	244.93	246.85	247.09
65	236.18	237.05	237.98	238.95	240.00	241.12	242.32	243.62	245.03	246.57	248.24	250.10	250.35
70	240.55	241.27	242.07	242.90	243.83	244.85	245.95	247.17	248.50	249.97	251.57	253.38	253.61
75	244.92	245.50	246.17	246.86	247.67	248.58	249.59	250.73	251.97	253.38	254.86	256.66	256.88

Table 43. Heat Content of Sodium Chloride Brines, B.t.u. per Lb.

Temp., °F.	Specific gravity, 60°/60°F.															
	1.03	1.04	1.05	1.06	1.07	1.08	1.09	1.10	1.11	1.12	1.13	1.14	1.15	1.16	1.17	1.178
	Freezing point, °F.															
	27.6	26.0	24.4	22.7	20.8	19.0	17.0	14.9	12.7	10.4	8.0	5.4	2.7	−0.3	−3.4	−6.0
	%, NaCl															
	4.14	5.51	6.86	8.21	9.56	10.88	12.20	13.51	14.81	16.09	17.36	18.63	19.87	21.11	22.33	23.30
−6.03	62.26	70.26	78.21	86.12	93.97	101.77	109.50	117.16	124.76	132.29	139.75	147.11	154.42	161.65	168.81	174.45
−4	64.12	72.42	80.67	88.88	97.03	105.12	113.14	121.09	128.97	136.79	144.53	152.17	159.75	167.26	174.69	176.03
−2	66.03	74.65	83.21	91.74	100.20	108.60	116.93	125.19	133.37	141.48	149.52	157.45	165.33	173.13	177.49	177.59
0	68.07	77.05	85.98	94.86	103.67	112.43	121.11	129.71	138.24	146.70	155.07	163.34	171.55	178.98	179.07	179.16
2	70.27	79.66	88.99	98.28	107.50	116.67	125.73	134.74	143.65	152.49	161.25	169.90	178.48	180.56	180.64	180.73
4	72.64	82.50	92.31	102.06	111.74	121.35	130.86	140.33	149.70	158.98	168.18	177.25	182.03	182.15	182.22	182.29
6	75.22	85.61	95.95	106.22	116.43	126.57	136.61	146.57	156.44	166.23	175.90	183.47	183.64	183.74	183.80	183.86
8	78.08	89.09	100.04	110.95	121.75	132.50	143.15	153.73	164.19	174.56	184.83	185.09	185.24	185.33	185.38	185.43
10	81.32	93.10	104.81	116.45	128.00	139.48	150.88	162.16	173.33	184.41	186.56	186.71	186.84	186.92	186.96	187.00
12	85.04	97.72	110.32	122.90	135.35	147.70	159.98	172.13	184.21	188.00	188.20	188.33	188.45	188.51	188.54	188.58
14	89.36	103.14	116.84	130.47	144.00	157.44	170.76	183.96	189.44	189.66	189.84	189.96	190.06	190.11	190.12	190.15
16	94.46	109.60	124.65	139.62	154.48	169.24	183.87	190.85	191.11	191.31	191.48	191.58	191.67	191.71	191.71	191.72
18	100.74	117.62	134.40	151.09	167.68	184.13	192.26	192.54	192.78	192.97	193.12	193.21	193.28	193.30	193.29	193.30
20	108.77	127.97	147.06	166.04	184.90	193.64	193.98	194.24	194.46	194.63	194.76	194.83	194.89	194.90	194.88	194.88
22	119.54	141.96	164.26	186.43	195.03	195.39	195.70	195.94	196.14	196.29	196.40	196.46	196.50	196.50	196.46	196.45
24	135.41	162.72	189.89	196.38	196.80	197.14	197.42	197.63	197.81	197.94	198.04	198.09	198.12	198.10	198.05	198.03
26	160.83	196.19	197.70	198.18	198.57	198.88	199.14	199.33	199.49	199.61	199.69	199.72	199.73	199.70	199.64	199.61
28	198.38	199.00	199.53	199.98	200.34	200.63	200.86	201.03	201.17	201.27	201.33	201.34	201.34	201.30	201.23	201.19
30	200.27	200.85	201.35	201.77	202.11	202.38	202.59	202.74	202.86	202.94	202.98	202.98	202.96	202.91	202.82	202.77

Table 43. Heat Content of Sodium Chloride Brines, B.t.u. per Lb.—(*Concluded*)

Temp., °F.	Specific gravity, 60°/60°F.															
	1.03	1.04	1.05	1.06	1.07	1.08	1.09	1.10	1.11	1.12	1.13	1.14	1.15	1.16	1.17	1.178
	Freezing point, °F.															
	27.6	26.0	24.4	22.7	20.8	19.0	17.0	14.9	12.7	10.4	8.0	5.4	2.7	−0.3	−3.4	−6.0
	%, NaCl															
	4.14	5.51	6.86	8.21	9.56	10.88	12.20	13.51	14.81	16.09	17.36	18.63	19.87	21.11	22.33	23.30
32	202.16	202.71	203.18	203.57	203.88	204.12	204.31	204.44	204.54	204.60	204.63	204.61	204.58	204.51	204.41	204.35
34	204.05	204.57	205.01	205.37	205.65	205.87	206.04	206.14	206.22	206.26	206.28	206.24	206.20	206.11	206.00	205.93
36	205.94	206.43	206.84	207.17	207.43	207.62	207.76	207.85	207.91	207.93	207.93	207.88	207.82	207.72	207.60	207.51
38	207.84	208.29	208.67	208.97	209.20	209.37	209.49	209.56	209.60	209.60	209.58	209.51	209.44	209.33	209.19	209.10
40	209.73	210.15	210.50	210.78	210.98	211.12	211.22	211.26	211.28	211.27	211.23	211.15	211.06	210.94	210.79	210.68
42	211.62	212.01	212.33	212.57	212.75	212.87	212.95	212.97	212.97	212.94	212.89	212.79	212.68	212.54	212.38	212.27
44	213.52	213.87	214.16	214.38	214.53	214.62	214.68	214.68	214.66	214.61	214.54	214.43	214.31	214.16	213.98	213.86
46	215.41	215.73	215.99	216.18	216.31	216.38	216.41	216.39	216.35	216.29	216.20	216.07	215.93	215.77	215.58	215.44
48	217.30	217.59	217.82	217.99	218.09	218.14	218.15	218.11	218.05	217.96	217.85	217.71	217.56	217.38	217.17	217.03
50	219.20	219.46	219.66	219.80	219.87	219.89	219.88	219.82	219.74	219.63	219.51	219.34	219.18	218.99	218.77	218.62
52	221.09	221.32	221.49	221.60	221.65	221.65	221.62	221.53	221.43	221.31	221.17	220.99	220.81	220.60	220.37	220.20
54	222.99	223.19	223.33	223.41	223.44	223.41	223.35	223.25	223.13	222.99	222.83	222.63	222.44	222.22	221.97	221.79
56	224.88	225.04	225.16	225.22	225.22	225.17	225.09	224.96	224.83	224.67	224.49	224.27	224.06	223.82	223.57	223.38
58	226.78	226.92	227.00	227.03	227.00	226.93	226.83	226.68	226.52	226.34	226.15	225.92	225.69	225.44	225.17	224.97
60	228.68	228.79	228.84	228.84	228.79	228.69	228.57	228.40	228.22	228.02	227.81	227.56	227.32	227.05	226.77	226.56
62	230.80	230.65	230.68	230.65	230.57	230.46	230.31	230.12	229.92	229.70	229.47	229.20	228.95	228.67	228.37	228.15
64	232.47	232.51	232.51	232.46	232.36	232.22	232.05	231.84	231.62	231.38	231.13	230.85	230.58	230.29	229.98	229.74
66	234.37	234.38	234.35	234.27	234.15	233.98	233.79	233.56	233.32	233.07	232.80	232.50	232.21	231.90	231.58	231.33
68	236.27	236.25	236.19	236.09	235.94	235.74	235.53	235.28	235.02	234.75	234.46	234.14	233.84	233.52	233.18	232.92
70	238.17	238.12	238.03	237.90	237.72	237.51	237.27	237.00	236.72	236.43	236.13	235.80	235.47	235.13	234.79	234.51
72	240.07	239.99	239.87	239.71	239.51	239.27	239.01	238.72	238.42	238.11	237.79	237.44	237.10	236.75	236.39	236.10
74	241.97	241.86	241.72	241.53	241.30	241.04	240.76	240.44	240.12	239.79	239.45	239.09	238.74	238.37	237.99	237.70
76	243.87	243.74	243.56	243.34	243.09	242.81	242.50	242.16	241.82	241.47	241.12	240.74	240.37	239.99	239.60	239.29
78	245.77	245.61	245.40	245.15	244.88	244.57	244.25	243.89	243.53	243.16	242.79	242.39	242.00	241.61	241.21	240.88
80	247.67	247.47	247.24	246.98	246.68	246.34	245.99	245.61	245.23	244.84	244.45	244.04	243.64	243.23	242.81	242.47

In all large carbon dioxide pipe work, it is customary to anchor the lines to the floor and to provide cork pads so the shock will not be passed to the building. Elimination of all vibration is a very important factor in air-conditioning work.

Brines (see also Tables 44 to 116 on specific gravity, Sec. 3, pp. 412–427). The brines commonly employed in refrigeration are calcium and sodium chlorides. The latter is cheaper but cannot be used below its eutectic point of −6.03°F. Calcium chloride of commercial grade does not operate very satisfactorily below −40°F. The specific gravity, freezing point, composition by weight, and heat content per pound of solution are given in Tables 42 and 43 which have been compiled by Jessup [*Refrig. Eng.*, **22**, 168–169 (1931)].

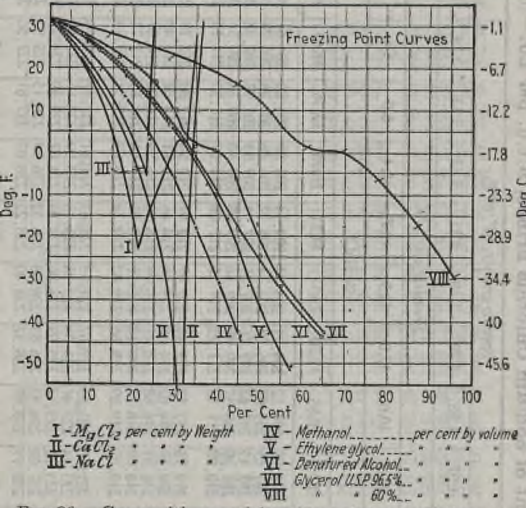


Fig. 30.—Compositions and freezing points of solutions.

While these brines have the great advantage of low cost, they have the disadvantage of being extremely corrosive and the calcium brines that of throwing down insoluble precipitates with untreated waters. Corrosion in closed systems can be largely overcome by the addition of sodium dichromate, $Na_2Cr_2O_7 \cdot 2H_2O$, 100 lb. and 200 lb. being required per 1000 cu. ft. of calcium and sodium brines, respectively. Enough caustic is added to make the brines slightly alkaline. In open systems using sodium brines, disodium phosphate, $Na_2HPO_4 \cdot 12H_2O$, may be used at the rate of 100 lb. per 1000 cu. ft., but the solutions should be colorless to phenolphthalein. Open calcium-brine systems may be protected by adding zinc dust at the rate of 60 lb. per 1000 cu. ft., a little at a time.

New brines have recently been developed in which corrosion is prevented by the use of buffers maintaining the proper acidity and protective colloids [Reinhartin Brines, *Refrig. Eng.*, **23**, 30 (1932)].

Magnesium chloride brines are used to some extent, but their eutectic temperature is not low enough to give them much advantage over sodium chloride.

Other materials which may be used are methanol, denatured alcohol, ethylene glycol, and glycerin. These have been much used for protection of automobile radiators and are no more corrosive than water. Their main disadvantage is cost, but they have been used in place of brines for household refrigerators. A comparison of freezing points is shown in Fig. 30.

The eutectic for the alcohols has not been determined, but it lies well below that of calcium chloride. The alcohols will give no difficulty from crystallization, but if the concentrations of the sodium and calcium brines are greater than 23 or 33 per cent, respectively, solid salt may deposit out. When solutions begin to freeze, their fluidity persists until some lower temperature is

Table 44. Freezing Points and Flow Points of Ethylene Glycol, Methanol, Denatured Alcohol, and Glycerol*

Composition				Freezing point		Flow point	
Weight, %		Volume, %		°C.	°F.	°C.	°F.
			Ethylene Glycol				
10		9.10		− 3.3	26.1	− 4.3	24.3
20		18.39		− 7.8	18.0	− 8.6	16.6
30		27.86		−13.5	7.7	−14.0	+ 6.8
35		32.67		−17.1	1.2	−19.1	− 2.4
40		37.53		−22.1	− 7.8	−24.5	−12.1
45		42.44		−26.7	−16	−34.5	−30.1
50		47.41		−35.4	−31.7	−43.9	−47.4
55		52.42		−41.7	−43.0	−47	−52.6
60		57.48		−46	−50.8	−53	−63.4
			Synthetic Methanol				
10		12.24		− 6.8	19.8	− 8.5	16.7
15		18.13		−10.5	13.1	−13.9	7.0
20		23.89		−16.2	2.8	−18.4	− 1.1
25		29.50		−21.2	− 6.2	−24	−11.2
30		34.98		−28	−18.4	−34	−19.2
35		40.33		−35.2	−31.4	−40	−40
40		45.56		−42	−43.6	−49.5	−57.1
			Denatured Alcohol				
10		11.9		− 4.3	24.26	− 6	21.2
15		17.75		− 6.9	19.58	− 8.5	16.7
20		23.37		−10	14	−14	6.8
25		28.25		−14	6.8	−16	3.2
30		34.33		−18.1	− 0.2	−21.4	− 6.5
35		39.65		−23.5	−10.3	−25.5	−13.9
40		44.85		−28	−18.4	−34.5	−30.1
45		49.95		−31.6	−24.9	−39.9	−39.9
50		54.95		−35.5	−31.9	−42.4	−44.3
60		64.66		−42.0	−43.6	−46.4	−52.5
			Glycerol				
96.5%	100%	96.5%	60%				
10	9.65	8.14	14.23	− 2.2	28.0	− 3	26.6
20	19.30	16.62	29.09	− 5.3	22.46	− 9.0	15.8
30	28.95	25.46	44.65	− 8.8	16.3	−15.8	3.5
35	33.78	30.03	52.71	−12.4	9.5	−18.9	− 2.0
40	38.60	34.70	60.96	−17.2	1.04	−19.5	− 3.1
45	43.43	39.47	69.40	−18.0	0.4	−20.5	− 4.9
50	48.25	44.36	78.03	−21.4	− 6.5	−28	−18.4
55	53.08	49.35	86.91	−27.5	−17.5	−35.9	−32.6
60	57.90	54.46	95.98	−34.0	−29.2	−41.9	−43.6
70	67.55	65.03	.....	−41.5	−42.7		

* From *Ind. Eng. Chem.*, **22**, 1316 (1930).

reached. The temperature at which the slush of ice crystals and liquid ceases to flow through a ¼-in. pipe has arbitrarily been defined as the **flow point** [Olsen, Brunjes, and Olsen, *Ind. Eng. Chem.*, **22,** 1316 (1930)]. A comparison of freezing points and flow points is given in Table 44.

Table 45. Specific Heats (Extrapolated) of Non-freezing Solutions

Weight, %	+23°F. −5°C.	+14°F. −10°C.	+5°F. −15°C.	−4°F. −20°C.	−13°F. −25°C.	−22°F. −30°C.	−31°F. −35°C.	−40°F. −40°C.
Methanol [Bose, *Z. phys. Chem.*, **58,** 585 (1907)]								
10	1.02							
20	0.98	0.98	0.98					
30	0.92	.92	.92	0.91	0.91			
40	0.87	.87	.86	.86	.85	0.85	0.84	0.83
50	0.81	.80	.79	.79	.78	.77	.76	.76
60	0.76	.76	.75	.74	.73	.73	.72	.71
70	0.71	.70	.69	.69	.68	.67	.66	.66
80	0.66	.66	.65	.64	.64	.63	.62	.62
90	0.62	.61	.60	.60	.59	.58	.58	.57
100	0.57	.56	.56	.55	.55	.54	.54	.53
Ethanol [Bose, *Z. phys. Chem.*, **58,** 585 (1907)]								
10	1.04							
20	1.04	1.04						
30	0.99	0.99	0.99	0.99				
40	0.93	0.92	.92	.91	0.91	0.90		
50	0.86	0.86	.85	.84	.84	.83	0.82	
60	0.80	0.79	.78	.77	.76	.75	.75	0.74
70	0.73	0.72	.71	.70	.69	.68	.67	.66
80	0.66	0.64	.63	.62	.61	.60	.59	.57
90	0.59	0.58	.57	.56	.54	.53	.52	.51
100	0.54	0.52	.51	.50	.49	.48	.47	.46
Ethylene Glycol (*Tech. Papers*, Carbide and Carbon Chemicals Corporation)								
10	1.00							
20	0.94	0.94						
30	0.87	.87	0.86	0.86				
40	0.84	.84	.83	.83	0.82	0.82		
50	0.78	.78	.77	.76	.75	.75	0.74	0.74
60	0.73	.72	.72	.71	.71	.70	.70	.69
70	0.70	.70	.69	.69	.68	.68	.67	.67
80	0.68	.68	.67	.67	.66	.66	.65	.65
90	0.65	.65	.64	.64	.63			
100	0.52	.50						
Glycerine ("International Critical Tables" and Landolt-Börnstein Tables, 5th ed.)								
10	0.96							
20	.93	0.93						
30	.89	.89	0.89					
40	.85	.85	.85	0.85				
50	.79	.79	.79	.79	0.79			
60	.76	.77	.77	.77	.77	0.77		
70	.71	.71	.71	.72	.72	.72	0.72	0.72

It seems very probable that future developments in low-temperature refrigeration will require the adoption of methanol or ethanol solutions with eutectic points of about −200°F. for solutions containing 80 per cent by weight.

Brine to Be Circulated. The brine required for a given condition may be calculated easily. The heat capacity of a certain volume of brine depends on the product of the density (specific gravity) and its specific heat (heat units required to raise 1 lb. 1°F.). For instance, the cooling effect of 100 gal. brine in raising 4°F. would be

$$\underset{\substack{\text{Degrees}\\ \text{of range}}}{4} \times \underset{\substack{\text{Number of}\\ \text{gallons}}}{100} \times \underset{\substack{\text{Weight of}\\ \text{gallons of}\\ \text{water}}}{8\tfrac{1}{3}} \times \underset{\substack{\text{Specific}\\ \text{gravity of}\\ \text{brine}}}{1.2} \times \underset{\substack{\text{Specific}\\ \text{heat of}\\ \text{brine}}}{0.7} = 2800 \text{ B.t.u.}$$

assuming 1.2 for the specific gravity and 0.7 for the specific heat of the brine. These values change with each concentration of brine solution.

If it is required to find the amount of brine per minute necessary to provide 1 ton of refrigeration with a 4° range of temperature, the calculation becomes

$$200 = \text{number of gallons} \times 8.33 \times 1.2 \times 0.7 \times 4$$

$$\text{Number of gallons} = \frac{200}{8.33 \times 1.2 \times 0.7 \times 4} = \frac{200}{28} = 7.14 \text{ gal. per min.}$$

The concentration or density of calcium or sodium chloride brines is determined by a hydrometer, the Baumé hydrometer being largely used for the purpose. The density is also measured by a salinometer, which gives a hydrometric reading four times that on the corresponding Baumé scale. For methanol, alcohol, or glycol solutions, hydrometers such as are used for testing antifreeze in automobile radiators may be used.

Nessler's Solution. To determine ammonia leakage into the brine Nessler's solution is used, which is prepared as follows: Dissolve 17 g. mercuric chloride in about 300 cc. distilled water; dissolve 35 g. potassium iodide in 100 cc. water. Add the former solution to the latter, with constant stirring, until a slight permanent red precipitate is formed. Next dissolve 120 g. potassium hydrate in about 200 cc. water; allow the solution to cool and then add to the previous solution and make up with water to 1 l. Add mercuric chloride solution until a permanent precipitate again forms. Allow to stand till settled and decant off the clear solution for use. Store in glass-stoppered blue bottles in a dark place.

In calcium chloride brine, Nessler's solution will form a yellow precipitate, but if no ammonia is present the precipitate will be almost white. In water or brine the precipitate will be yellow if there is but a trace of ammonia present, and a reddish brown if there is considerable ammonia in the sample.

Pressure Drop and Power Required in Brine Circulation. The pressure drop in brine pipes is determined by the rate of flow, the internal diameter and length of the pipe, and the viscosity of the brine. The method for finding the pressure drop with water has already been discussed in Sec. 6, pp. 803 to 826 and Fig. 5. The pressure drop for brines may be found by multiplying the values for water by the ratio of the kinematic viscosities of brine and water. This factor f is given by the formula

$$f = \frac{\text{viscosity of brine}}{\text{viscosity of water}} \times \frac{\text{density of water}}{\text{density of brine}}$$

The viscosities of most solutions have not been experimentally determined at low temperatures. However, the curve obtained by plotting the logarithm

Table 46. Viscosities (Extrapolated) of Refrigerating Solutions (Centipoises)

Weight, %	0°C. 32°F.	−5°C. 13°F.	−10°C. 14°F.	−15°C. 5°F.	−20°C. −4°F.	−25°C. −13°F.	−30°C. −22°F.	−35°C. −31°F.	−40°C. −40°F.	Freezing point
Sodium Chloride [Jessup, *Refrig. Eng.*, **12**, 171 (1925)]										
10.5	2.1	2.4	....	....	.....	.....	.....	....	....	+20°F.
16.8	2.4	2.8	3.2	....	.....	.....	.....	....	....	+10°F.
21.0	2.7	3.1	3.6	4.2	.....	.....	.....	....	....	0°F.
Calcium Chloride [Jessup, *Refrig. Eng.*, **12**, 171 (1925)]										
11.0	2.1	2.4	....	....	.....	.....	.....	....	....	+20°F.
16.0	2.6	2.9	3.3	....	.....	.....	.....	....	....	+10°F.
20.0	3.1	3.5	4.0	4.5	.....	.....	.....	....	....	0°F.
22.8	3.6	4.1	4.6	5.2	5.9	.....	.....	....	....	−10°F.
25.2	4.0	4.6	5.1	5.8	6.6	7.4	.....	....	....	−20°F.
27.2	4.6	5.2	5.8	6.6	7.5	8.4	9.6	....	....	−30°F.
29.0	5.1	5.8	6.6	7.4	8.4	9.5	10.8	12.3	14.1	−40°F.
Methanol ("International Critical Tables" and Landolt-Börnstein Tables, 5th ed.)										
10.0	2.6	3.2	....	....	.....	.....	.....	....	....	+20°F.
16.8	3.0	3.7	4.5	....	.....	.....	.....	....	....	+10°F.
22.0	3.4	4.2	5.0	6.1	.....	.....	.....	....	....	0°F.
26.2	3.6	4.4	5.3	6.5	7.9	.....	.....	....	....	−10°F.
30.4	3.7	4.5	5.5	6.8	8.3	10.3	.....	....	....	−20°F.
34.2	3.6	4.4	5.3	6.5	7.9	9.7	12.0	....	....	−30°F.
38.4	3.4	4.2	5.0	6.1	7.3	8.9	11.0	13.3	16.5	−40°F.
Ethanol ("International Critical Tables" and Landolt-Börnstein Tables, 5th ed.)										
15.0	4.1	5.2	....	....	.....	.....	.....	....	....	+20°F.
22.8	5.7	7.4	9.5	....	.....	.....	.....	....	....	+10°F.
29.0	6.9	9.0	11.8	15.5	.....	.....	.....	....	....	0°F.
35.0	7.2	9.5	12.4	16.4	22.0	.....	.....	....	....	−10°F.
41.2	7.0	8.9	11.6	15.0	19.7	26.0	.....	....	....	−20°F.
48.8	6.7	8.4	10.8	13.8	17.7	23.0	30.0	....	....	−30°F.
56.8	6.1	7.6	9.5	12.1	15.4	19.8	25.5	32.5	42.5	−40°F.
Glycerol [Green and Parke, *J. Soc. Chem. Ind.*, **58**, 319 (1939)]										
30	....	6.5	....	....	.....	.....	.....	....	....	+15°F.
40	....	10.3	14.4	....	.....	.....	.....	....	....	+ 4.3°F.
50	....	18.8	24.4	....	48.1	.....	.....	....	....	− 9.4°F.
60	....	41.6	59.1	....	108.0	.....	244.0	....	....	−30.5°F.

of the absolute viscosity against the logarithm of the absolute temperature gives a straight line [Genereaux, *Ind. Eng. Chem.*, **22**, 1382 (1930)]. Extrapolation of this plot has given the viscosities in Table 46. Thermal conductivities of brines are given in Table 47.

Small-scale Refrigeration. The construction of fractional-tonnage refrigeration machines equipped with condenser and cooling system has become an important industry. They fall normally into two classes:

1. **Commercial units** from ¼ to 5 tons for use in retail stores, butcher shops, clubs, etc. 2. **Household units,** with ice-melting capacities of from 50 to 500 lb. per day (1/40 to ¼ ton).

The commercial units are nearly all compression systems, usually operated with electric power and using a water-cooled condenser. Intermittent-absorption systems of this size were popular a few years ago, but a number of disastrous explosions, mainly due to inexperienced operation, have rendered them practically obsolete.

Table 47. Thermal Conductivities of Refrigerating Brines

Brines	Weight, %	Temp., °C.	Conductivity, ‡ cal. per cm. per sec. per °C.
*NaCl	12.5	32	0.001403
	25.0	32	.001141
*$CaCl_2$	15.0	32	.001383
	30.0	32	.001315
*$MgCl_2$	11.0	32	.001376
	14.5	32	.001329
	22.0	32	.001290
	29.0	32	.001238
†CH_3OH	0	19	.00141
	25	19	.00107
	50	19	.00078
	75	19	.00061
	100	19	.00050
†C_2H_5OH	0	11	.00149
	25	10	.00104
	50	11	.00079
	75	12	.00059
	100	10	.00047
†Glycerine	0	20	.00140
	25	20	.00119
	50	20	.00101
	75	20	.00081
	100	20	.00070
*Ethylene glycol	100	0	.00064

* Landolt-Börnstein Tabellen, 5th ed.
† Lees, *Phil. Trans. Roy. Soc.*, (A) **191**, 399 (1898).
‡ For conversion factors see pp. 106 and 948.

Household machines must be automatic in operation, which, with the compression system, necessitates electric drive. In these systems the condenser is usually air cooled, by either a fan or a combined condenser radiator on top of the cabinet. The refrigerants commonly used are sulfur dioxide, methyl chloride, ethyl chloride, and Freon (F-12). Dimethyl ether has been used in some German household units. Ammonia, however, is seldom used in machines with an air condenser. Among the household refrigerators, there is only one important unit employing the absorption system, namely the Electrolux (*cf.* Continuous Absorption Systems, p. 2540). This unit is noiseless since it has no moving mechanical parts and gives no interference with radio reception since it has no motor.

The automatic control of small machines is effected by a thermostat which turns off the power at any desired temperature and by an automatic pressure-reducing valve for maintaining a constant suction pressure. In addition to the automatic control many commercial units are usually equipped with brine tanks for storing up refrigeration. Alcohol may be used in place of brine to provide a non-freezing solution and has the advantage over brine of being non-corrosive.

In designing household machines it is imperative to prevent all leakage. This would not only be dangerous to health but would seriously interfere with the operation of the machine.

Small machines are usually less efficient than large ones (see Table 48), but their operating costs are not excessive (1 to 2 kw.-hr. per day for compression units or 50 cu. ft. gas per day for absorption units) and they have attained great popularity because of their convenience. They are apt to dehydrate the ice-box atmosphere excessively, but hydrators are now available for keeping materials which might spoil by excessively dry air. Temperatures required for food preservation are given in Tables 49 and 50. A typical diagram of a small household unit is shown in Fig. 31.

Natural Refrigeration Processes (Non-recirculating Systems). These processes, as distinguished from mechanical refrigeration processes, are those in which the heat flow follows its normal course from a higher to a lower temperature. They may also be distinguished from the conventional mechanical or absorption systems in that the refrigerant is not recovered. The most important natural refrigerant is ice, but within recent years solid CO_2 has become very popular and has replaced ice in several instances. As far back as 1910, the evaporation of liquid ammonia was used for cooling refrigerator cars on the Russian railways, the vapors being absorbed in water for recovery by redistillation at convenient points. Liquid butane and propane have recently been applied to truck refrigeration in the same way, the vapors being used as fuel in the motor (*Nat. Petroleum News*, Dec. 2, 1931, p. 33. "Butane-propane Gases Supplement," 2d ed., Western Business Papers, Los Angeles, p. 24, 1937). Chemical methods in which a solid is dissolved in a liquid or two solids melt and go into solution are other examples.

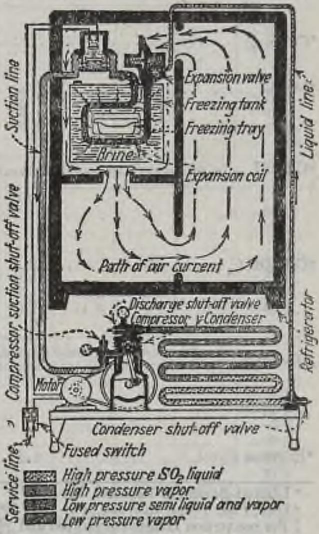


FIG. 31.—Household refrigeration unit (Kelvinator, dry system).

ICE MANUFACTURE

Natural ice was formerly the most important means of refrigeration, but its use is steadily decreasing because of the uncertainty of both the quality and quantity of the supply due to changing winter conditions. A large space is required for storage, considerable shrinkage takes place during storage, and it is difficult to cut up into cakes of convenient size and accurate weight for the retail trade. Nevertheless there are isolated communities in colder climates where it still can compete with the manufactured product.

Manufactured ice is made by two methods: (1) the **plate** system and (2) the **can** system. In the **plate system** a plate at 0°F. or lower is immersed in

Table 48. Tests on SO_2 Household Machine[*]

Data and results at constant room temperature, suction pressure of 15.22 lb.

	Number of test											
	1	2	3	4	5	6	7	8	9	10	11	12
Discharge pressure, lb. abs.	49.80	62.30	62.80	69.80	77.30	84.80	92.80	104.80	114.80	124.30	133.80	139.30
Suction pressure, lb. abs.	15.22	15.22	15.22	15.22	15.22	15.22	15.22	15.22	15.22	15.22	15.22	15.22
SO_2, lb. per hr	6.34	6.16	6.03	5.92	5.86	6.61	5.28	4.98	4.70	4.38	3.95	3.91
Sat. temp. and pressure, i''	184.20	184.20	184.20	184.20	184.20	184.20	184.20	184.20	184.20	184.20	184.20	184.20
Exp. valve liq. temp., °F	60.50	67.50	71.80	73.80	75.80	74.80	75.80	74.80	75.80	77.70	76.70	74.50
Capacity ice, lb. per 24 hr.	159.20	152.50	148.00	144.30	142.10	136.20	128.10	121.00	114.00	106.00	95.80	95.30
Compressor, r.p.m.	339.00	341.00	342.00	340.00	339.00	340.00	338.00	338.00	337.00	337.00	338.00	337.00
Suction temp., compressor, °F	61.50	71.30	73.80	77.30	78.80	78.80	83.70	83.70	85.50	88.00	91.00	87.40
Discharge temp., °F	127.50	146.80	149.70	157.00	162.00	163.20	172.50	173.00	174.10	175.50	182.00	184.50
Sat. SO_2, cu. ft. per hr.	31.90	31.00	30.40	29.80	29.50	28.30	26.60	25.10	23.70	22.10	19.90	19.70
Piston displ., cu. ft. per hr.	45.60	45.90	46.00	45.70	45.60	45.70	45.50	45.50	45.40	45.40	45.50	45.40
Apparent vol. eff., %	70.00	67.60	66.10	65.20	64.60	61.80	58.50	55.20	52.20	48.60	43.70	43.40
SO_2, cu. ft. per hr. at compressor	36.10	35.90	35.30	34.90	34.70	33.20	31.60	29.80	28.20	26.40	24.00	23.60
Actual vol. eff., %	79.20	78.30	76.70	76.30	76.10	72.80	69.50	65.50	62.10	58.20	52.80	52.00
Adiabatic power, watts	37.40	44.60	44.00	47.00	50.10	50.70	51.00	52.10	51.70	50.60	47.90	48.50
Motor power, watts	197.00	202.00	199.00	203.00	204.00	208.00	213.00	220.00	224.00	232.00	236.00	233.00
Motor eff., %	60.20	61.10	60.80	61.20	61.40	62.00	63.00	64.00	64.50	65.70	66.00	65.80
Compr. brake power, watts	118.00	123.00	121.00	124.00	125.00	129.00	134.00	141.00	144.00	152.00	156.00	153.00
No-load power, watts	60.00	60.00	60.00	60.00	60.00	60.00	60.00	60.00	60.00	60.00	60.00	60.00
Power to gas, watts	58.00	63.00	61.00	64.00	65.00	69.00	74.00	81.00	84.00	92.00	96.00	93.00
Compression eff., %	64.50	70.90	72.20	73.40	77.10	73.50	68.90	64.40	61.60	55.20	49.90	52.20
Mechanical eff., %	49.10	51.20	50.40	51.60	52.00	53.50	55.20	57.40	58.20	60.50	61.50	60.80
Compressor eff., %	31.70	36.30	36.40	37.90	40.10	39.30	38.10	37.00	35.90	33.30	30.70	31.70
Carnot eff., %	27.60	31.20	30.90	31.80	33.60	33.40	32.10	31.10	30.00	28.10	25.80	26.90
Performance factor, compressor	2.37	2.18	2.15	2.04	2.00	1.86	1.68	1.51	1.39	1.22	1.08	1.09
Performance factor, over all	1.42	1.33	1.30	1.25	1.22	1.15	1.06	0.97	0.89	0.80	0.71	0.72

---

[*] Test by L. A. Philipp and C. C. Spreen, *Refrigerating Eng.*, **13**, 75 (1927).

a tank of water. In a week's time a plate of ice about one foot thick is formed which is removed from the plate by permitting hot gas from the compressor to run through the hollow part of the plate. While the ice is of excellent quality, it is non-uniform in thickness and has not found much favor in the retail trade.

Table 49. Cold-storage Temperatures

Food articles	°F.	Food articles	°F.
Fruit		Liquids	
Apples	32–36	Beer, ale, porter, etc.	33
Bananas	34	Cider	30
Berries, fresh	36	Ginger ale	36
Cranberries	33–36	Wines	40–45
Cantaloupes	40	Flour and Meal	
Dates, figs, etc.	50–55	Buckwheat flour	36–40
Fruits, dried	35–40	Corn meal	36–40
Grapes	34–36	Oat meal	36–40
Lemons	33–36	Wheat flour	36–40
Oranges	34–36	Vegetables	
Peaches	34–36	Asparagus	34–35
Pears, watermelons	34–36	Cabbage	34–35
Meats		Carrots	34–35
Brined	38	Celery	34–35
Beef, fresh	33	Dried beans	32–40
Beef, dried	36–40	Dried corn	35
Calves	32–33	Dried peas	35–40
Hams, ribs, shoulders (not brined)	20	Onions	36
Hogs	29–32	Parsnips	34–36
Lard	38	Potatoes	36–40
Livers	20–30	Sauerkraut	35
Sheep, lambs	32	Miscellaneous	
Ox-tails	30	Cigars, tobacco	35
Sausage casings	20	Furs, woolens, etc.	35
Tenderloin, butts, etc.	33	Honey	45
Fish		Hops	40
Fresh fish	20	Maple syrup, sugar	40–45
Dried fish	36	Oils	35
Oysters in shell	30–35	Poultry, dressed, iced	28–30
Oysters in tubs	25	Poultry, dry picked	26–28
Canned Goods		Poultry, scalded	20
Sardines	35–40	Game, to freeze	15–18
Fruits	35–40	Game, after frozen	25–28
Meats	35–40	Poultry, to freeze	15–18
Butter, Eggs, Etc.		Poultry, after frozen	25–28
Butter	18–20	Nuts, in shell	35–40
Butterine	18–20	Chestnuts	33
Cheese	34		
Eggs	31		

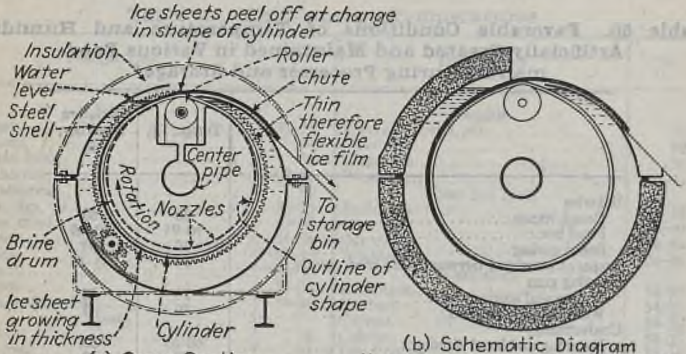
In the **can system,** a can containing 300 or 400 lb. water is immersed in brine at such a temperature that the water will freeze in about 44 hr. time. If distilled water is used the freezing is straightforward, but, if raw, hard water saturated with air is used, two effects must be overcome. Unless the water is agitated, air bubbles will be set free and freeze into the ice giving it a marblelike appearance and poor strength. Agitation with low- or high-pressure air is the usual method of overcoming this difficulty. The dissolved salts in the water will also precipitate on freezing and deposit on the ice surface to cause discoloration. Air agitation will also prevent this effect by keeping the particles in suspension, but at the end of the freezing period a core of turbid water remains. This is removed by a **core sucker** and replaced with fresh water. The air used for agitation must be dehumidified at a temperature below 32°F. to avoid freezing of the pipes. Air agitation is a rather expensive operation but is preferable to the use of distilled water such as was used in the older installations. The brine used for freezing is vigorously stirred to promote heat transfer. At the completion of freezing the cans are immersed in water at room temperature or above to loosen the cake, which is then removed from the can by dumping. The cake next goes to the **scoring machine** which cuts grooves at the points where it must be split up into the 25- and 50-lb. lots for the retail trade. Cakes which have been checked or cracked are used for crushed ice.

Table 50. Favorable Conditions of Temperature and Humidity Artificially Created and Maintained in Various Food-manufacturing Processes and Storage*

	Temp., °F.	Relative humidity, %
Bakeries		
Dough rooms	80	80
Proof box	90–95	80–90
Bread cooling	70	65
Prepared powdered beverages and crisp cereals	75	35–40
Chewing gum		
Rolling and scoring chicle	75	50
Wrapping and packing	70	45
Confectionery		
Enrobing and hand dipping	60–65	55
Hard-candy manufacturing	70	40
Starch room	75–85	50
Packing	65	50
Dairy products		
Butter manufacturing	60	60
Chill room	40	60
Fruits		
Apple storage	31–34	80–85
Avocado packing	40	50
Bananas		
Holding ripe fruit	56	70–75
Holding green fruit	58	70–75
Slow ripening	60–62	90
Normal ripening	64–68	90
Fast ripening	70–72	90
Danger of chilling	Below 49	
Meat products		
Butter substitutes		
Churn room	70	60
Print room	60	60
Chill room	30	60
Cooler	55	60
Bacon slicing	60	48
Egg candling	60	48
Sugar storage	80	35

* This table is offered only to demonstrate the wide variance of conditions which may be demanded even within a single plant. *Food Industries, May*, 1931.

Ice for refrigerator cars is manufactured in plants at strategic points along the railroad lines, with icing platforms sufficiently long to handle a whole train of cars at once. The time of icing a vegetable or fruit car with cake ice is 1 to 2½ min. A car in transit from the Rio Grande Valley to Chicago will use about 8 to 12 tons of ice. Many attempts have been made to refrigerate cars by mechanical means, but they have not been successful in this country though they are used to some extent abroad. The chief objection seems to be that no systems are wholly reliable without costly supervision.



(a) Cross Section

(b) Schematic Diagram How Frozen Water-Ribbons Peel

FIG. 32.—Flakice machine.

Flakice. For numerous applications ice cakes have to be cracked up before they can be used. Field has developed a machine for freezing water directly in small chips or flakes [*Refrig. Eng.*, **31**, 95, (1936)]. The machine consists of a flexible cylinder with several metal panels separated by rubber strips. This cylinder is cooled internally while revolving partly submerged in water. A thin layer of ice freezes upon the metal panels, which is discharged from the panels as the freezing edge emerges from the water.

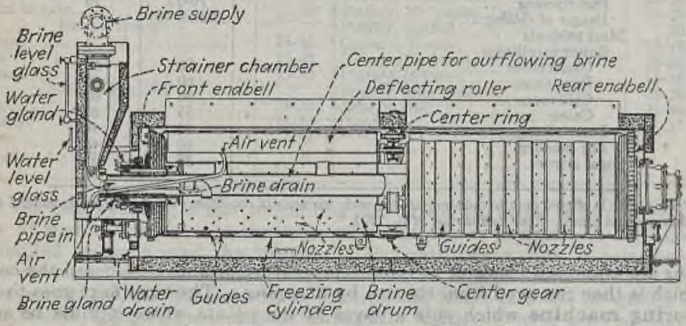


FIG. 33.—Flakice machine, longitudinal section.

The method of discharge is unique. The freezing cylinder is slightly flexible. Inside the cylinder at the point of discharge is a deflecting roller which distorts the circular form. The ice sheet being rigid leaves the cylinder at a tangent and rolls onto a chute where it cracks up and drops into a bin. The relation of thickness to time of freezing is shown in Fig. 34.

The operation of the machine is indicated by Figs. 32 and 33.

The width of the strips depends on the width of the panels, since no ice forms on the rubber separators because of their low thermal conductivity. The thickness is about ⅛ in. and the length not over 2 ft. In storage, about 20 to 30 per cent of the space is void. The chips are usually frozen to a temperature 10° to 12° below 32°F. and are therefore crisp and dry. Because of their large exposed surface, the chips cool water about six times as rapidly as does crushed ice.

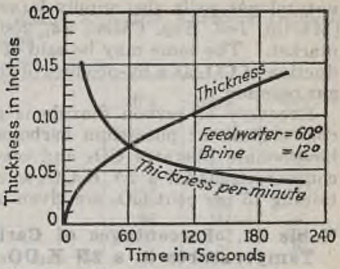


FIG. 34.—Freezing curve showing the rates of thickness to time of freezing.

Pak-Ice. Another recent development in the manufacture of ice is the Pak-Ice machine [Taylor, *Refrig. Eng.*, **22**, 307 (1931)].

> A corrugated liner is fitted into an outer casing and liquid ammonia under a 4-ft. head is fed into the space between the two. The inside of the liner is filled with circulating water (18 ft. per sec.) which freezes rapidly on the liner surface to a thickness not over 0.008 in. and is constantly removed by tool scrapers.

The ice scraped off the sides is driven toward the center and carried out of the machine in a stream of water and transferred to a bin when the water is drained off. The machine may also be used to produce frozen brines, a eutectic mixture of salt and ice (see Freezing Mixtures, p. 2626).

Both the Flakice and the Pak-Ice systems dispense with the items of air agitation, brine circulation, filling, and dumping cans, which, as noted above, add considerably to the cost of ice. They do not, however, produce the cakes of clear ice demanded by the retail trade.

Eutectic Salt-ice Mixtures. Where temperatures lower than the freezing point of ice are desired, it is possible to freeze a salt solution to the eutectic temperature −6°F. Such mixtures have been used in sealed cans and carried about for the servicing of ice-cream cabinets in isolated localities. When the brine melts, the can is returned for freezing. A Flakice machine can be used for freezing such mixtures [Field, *Refrig. Eng.*, **31**, 95 (1936)]. "Ice-cream bullets," another form of eutectic salt-ice mixture, have been manufactured by one of the larger metropolitan ice producers as a substitute for salt and cracked ice in making homemade ice cream.

The latent heat of fusion of 1 lb. of eutectic mixture is 101 B.t.u. Hence its reserve capacity for storing cold, although less than that of ice, 144 B.t.u., is considerably better than that of a cold liquid brine.

MANUFACTURE OF SOLID CARBON DIOXIDE

Solid carbon dioxide has become a very important source of cold, especially in the handling and transportation of foods. An exhaustive discussion of production and uses is given by Quinn and Jones ("Carbon Dioxide," Chaps. VI to VIII, Reinhold, 1936). Successful manufacture is dependent on a cheap supply of pure clean gas, as the product must be free from color, odor, taste, or any material injurious to health. The supply should be close to the market as there is considerable wastage in transportation.

The most important source of supply appears to be by product-fermentation gas [Jones, *Ind. Eng. Chem.*, **23**, 519, 798, 848 (1931)]. Other sources are flue gases from combustion and waste gases from lime burning. Certain natural gas wells also supply large quantities of gas at favorable pressures [Martin, *Ind. Eng. Chem.*, **23**, 256 (1931)] but are too far away from the market. The same may be said of other industrial sources, such as the production of CO_2 as a by-product of the manufacture of hydrogen by the water-gas reaction.

Recovery of carbon dioxide is generally accomplished by absorption in cold sodium or potassium carbonate solutions. On heating, the resulting bicarbonate liberates CO_2 and reverts to the carbonate. The equilibrium concentrations for a $2N$ K_2CO_3 solution (12 per cent) for a gas mixture containing 15 per cent CO_2 are given by Quinn and Jones (Table 51).

Table 51. Percentages of Carbonate and Bicarbonate at Varying Temperatures in a 2N K_2CO_3 Solution with 15 Per Cent CO_2 at Atmospheric Pressure

Temp., °C.	% K as K_2CO_3	% K as $KHCO_3$
20	9.5	90.5
40	14.2	85.8
60	22.2	77.8
80	34.0	66.0
100	47.9	52.1

Absorption is usually effected in coke towers and desorption in steam-heated lye boilers. Water vapor carried out with the gas is removed by condensers.

The CO_2 gas is next purified by chemical reagents such as permanganate or dichromate, or by adsorption on activated charcoal, silica, or aluminum gel. Traces of water vapor are removed by calcium chloride, by refrigeration, or by adsorption.

Following purification, the gas is compressed in three- or four-stage compressors and is condensed. Oil filters or separators are necessary to prevent oil contamination. The liquid carbon dioxide is the starting point for manufacture of the solid, but, if there is a demand for the liquid itself, it may be filled into cylinders.

Liquid carbon dioxide is transformed into solid carbon dioxide by the cooling effect of its own evaporation. At a temperature of 84°F. and 70 atm., the yield of solid on expansion to atmospheric pressure is 0.23 lb. per lb. of liquid, and 0.77 lb. of evaporated gas must be recompressed in a three- or four-stage compressor. Several cycles have been devised for carrying out this recompression most economically. These are: (*a*) *the simple cycle* in which the expanded gas is simply recompressed after expansion; (*b*) *the precooling cycle* in which the expanded gas is allowed to cool the liquid before it is expanded; (*c*) *the bleeder cycle* in which the liquid is expanded in three stages corresponding to the three stages of the compressor, the gas flashing in the first expansion being sucked into the third stage of the compressor, that from the second expansion into the second stage, and that from the last expansion into the first stage; (*d*) the *bleeder precooling cycle*, which is a combination of (*b*) and (*c*); (*e*) the *pressure snow-making* cycle in which the ice is frozen at the triple point rather than at atmospheric pressure; and finally (*f*) *the binary cycle* in which the liquid is cooled to some low temperature by an auxiliary ammonia refrigeration system.

The power requirements of the various cycles are shown in Table 52.

Table 52. Power Requirements for Solid CO_2 for Various Initial Temperatures*

Kw.-hr. per Ton

Cycle	40°F.	60°F.	80°F.
Simple	140	175	240
Precooling	135	160	200
Bleeder	115	140	155
Bleeder precooling	102	120	145
Pressure snow making	98	117	142
Binary	90	102	120
Ideal Carnot	65	75	90

* Stickney, *Refrig. Eng.*, **23**, 334 (1932).

The simple cycle obviously requires less capital investment than the more complicated cycles and may therefore be most satisfactory for small plants. The pressure snow-making cycle is used by most large plants. For further comparison see Rabe and Duevel, *Refrig. Eng.*, **22**, 18, 90, 260, 388 (1931).

The snow formed by any of the above cycles is in a light fluffy condition and is squeezed into solid blocks (10 by 10 by 10 in.) in a hydraulic press at 2000 lb. pressure. The transfer from the snow-making equipment to the press is effected by rabble arms.

Numerous other processes have been developed. Mention may be made of the Carba process in which the solid block is frozen directly without auxiliary pressure. Capital expenditures are low, but the regulation is delicate. Somewhat similar in principle are the Linde-Sürth and Agefko processes. The Maiuri process (*Cold Storage and Produce Rev.*, Sept. 21, 1931) uses an ammonia absorption system to produce temperatures of −76° to −94°F., which are used to cool an alcohol water bath surrounding freezing cans containing liquid carbon dioxide under 80 to 100 lb. pressure.

Solid carbon dioxide in 50-lb. blocks wrapped in paper are shipped in specially built refrigerator cars or trucks. When removed from the truck, it will not evaporate completely in 24 hours' time on exposure to the atmosphere, which is a decided advantage in handling as compared with water ice. Comparison of water ice and solid carbon dioxide is shown in Table 53.

Table 53. Physical Properties of Ice and Solid CO_2

Physical Property	Solid CO_2	Water ice
Specific gravity	1.53	0.90
Melting point, °F	−109.6	32
Latent heat of fusion, B.t.u. per lb	82.0	144
Latent heat of sublimation, B.t.u. per lb	240.6	
Sensible heat of gas to 32°F., B.t.u. per lb	34.4	
Net refrigerating effect, B.t.u. per lb	275	144

In many applications solid carbon dioxide may be used directly, but in some applications the vapors are detrimental, and the condensation of moisture may produce excessive dryness in the refrigerated space. The usual method of avoiding these effects is to use the solid indirectly through an intermediate low-freezing liquid, such as aqueous solutions of alcohol or methanol, the flow of which is regulated automatically to meet the refrigerating requirements. Calcium chloride brines cannot be used as their minimum freezing point is −60°F. (eutectic temperature), but 75 per cent solutions of ethanol or methanol in water have eutectic temperatures of −200°F. and are never in danger of freezing up by solid CO_2. Systems employing auxiliary circulating liquids are used extensively in truck refrigeration.

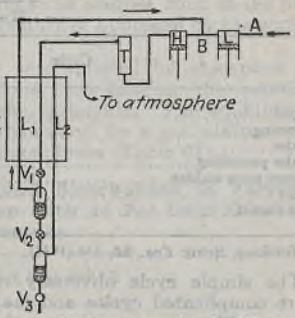


Fig. 35.—Linde process for liquid air.

A = air intake.
L = low-pressure compressor.
H = high-pressure compressor.
B = medium-pressure tee feeding high-pressure compressor.
T = drying tower.
HE = heat exchanger.
V_1 = throttling valve from high to medium pressure.
L_1 = return line for uncondensed medium-pressure air.
V_2 = reducing valve for liquid air at intermediate pressure to atmospheric.
L_2 = return line for liquid air evaporated in passage from intermediate to atmospheric pressure.
V_3 = draw-off valve for liquid air.

FREEZING MIXTURES

It is sometimes convenient to produce cold for temporary laboratory use by using mixtures of ice and various chemicals. Temperatures obtained by various mixtures of this type are shown in Table 54. In order to use such mixtures most effectively, both salt and ice should be finely divided and well stirred.

In freezing mixtures the source of cold is the latent heat of the ice and heat of solution of the salt. Dissolving a salt by itself will also produce cold (Table 55), but this process usually takes place so slowly that it is not a practical method with ordinary laboratory reagents. Some commercial preparations are occasionally announced which are claimed to be effective.

Where solid carbon dioxide is available, cooling mixtures may be prepared by dropping the cracked solid into acetone, methanol, or denatured alcohol. The solid alone is not very satisfactory for cooling because it cannot be brought in intimate contact with the vessel to be cooled.

LOW-TEMPERATURE PROCESSES

Gas Liquefaction and Separation. (See Wien-Harms, "Handbuch der Experimental-physik," vol. 9, pt. 1, pp. 45–186.) The liquefaction of gases is effected by cooling them below their critical temperature under sufficient pressure to cause condensation. Precooling by conventional refrigeration equipment is often used, but many refrigerants solidify above these low critical temperatures. Consequently the final stages of liquefaction require cooling by other means.

Two main processes are employed in attaining these low temperatures, respectively, the Linde and Claude processes. In the *Linde* process, the gas is expanded from a high to a low pressure and cools by the Joule-Thomson effect. A schematic diagram of the process is shown in Fig. 35.

The operation is as follows: Dry air free of CO_2 is compressed from a low pressure p_1 to an intermediate pressure p_2, where it is combined with partially

(Text continued on p. 2635)

Table 54. Temperatures of Freezing Mixtures[1]

Degrees Fahrenheit

Mixtures	Composition, parts by weight	Reduction in temp., from	to	Mixtures	Composition, parts by weight	Reduction in temp., from	to
Sodium phosphate	9	34.3°	21.2°	Sodium chloride	1	32.0°	−1.6°
Sal ammoniac	6			Snow	3		
Diluted nitric acid	4			Saltpeter	1	46.4°	−11.2°
Sodium sulfate	6	50.0°	−13.0°	Sal ammoniac	1		
Ammonium nitrate	5			Water	1		
Diluted nitric acid	4			Diluted nitric acid	1	57.2°	−31.0°
Sal ammoniac	5	50.0°	10.4°	Snow	1		
Saltpeter	5			Sodium sulfate	8	50.0°	−0.4°
Water	16			Hydrochloric acid	5		
Sodium carbonate	1	50.0°	6.8°	Sodium sulfate	8	50.0°	5.0°
Ammonium nitrate	1			Saltpeter	5		
Water	1			Sal ammoniac	5		
Sodium sulfate	3	50.0°	−2.2°	Water	16		
Diluted nitric acid	2			Sodium sulfate	5	50.0°	3.2°
Sodium sulfate	6	50.0°	−9.4°	Diluted nitric acid	4		
Sal ammoniac	4			Sodium chloride	1	32.0°	−0.4°
Saltpeter	2			Snow	1		
Nitric acid	4			Diluted sulfuric acid	1	23.0°	−41.8°
Ammonium nitrate	1	50.0°	3.2°	Snow	1		
Water	1			Calcium chloride	3	32.0°	−27.4°
Sodium phosphate	9	59.0°	15.8°	Snow	2		
Diluted nitric acid	4			Calcium chloride	2	32.0°	−43.6°
Potassium hydrate	4	32.0°	−38.3°	Snow	1		
Snow	3						
Sulfuric acid	1	−2.2°	−40.0°				
Nitric acid	1						
Snow	2						

---

[1] From "International Critical Tables."

Table 54. Temperatures of Freezing Mixtures—(*Continued*)

Substances	Composition of mixture (% anhydrous salt, unless otherwise stated). *E* = eutectic composition	Freezing point of solution	Initial condition of freezing mixture	Lowest attained temperature recorded	Heat absorbed at temperature of mixing, cal. per g. of mixture	Heat absorbed (at freezing or saturation point of solution) from objects to be cooled, cal. per g. of mixture. The * values are heats of fusion of the eutectic
$NaCl—H_2O$	22.4 (*E* for $NaCl.2H_2O$)	−21.2°				56.4*
	23.1 (*E* for NaCl)	−22.4°				
	24.8		Salt and ice, −1°	−21.3°		
			With ice	−21°		
$NaNO_3—H_2O$	33.3		Salt and ice, −1°	−17.75°		
	37.*E*	−18.5°				57.5*
	42.9		Water and salt, 13.2°	− 5.3°		
$Na_2CO_3.10H_2O—H_2O$	5.93*E*	− 2.1°				77.2*
	16.7		Salt and ice, −1°	− 2.0°		
$Na_2SO_4.10H_2O—H_2O$	3.8*E*	− 1.2°				80.1*
$Na_2S_2O_3.5H_2O—H_2O$	30.0*E*	−11°				
	52.4		Water and salt, 10.7°	− 8.0°		
$NaOOCCH_3.H_2O—H_2O$	45.9		Water and salt, 10.7°	− 4.7°		
$KCl—H_2O$	19.3	− 9.0°				71.2*
	23.1		Salt, −1° Ice, 0°	−10.9°		

KNO_3—H_2O	11.2*E*	− 3.0°					80.7*
	11.5			Salt and ice, −1°	− 2.85°		
K_2SO_4—H_2O	6.54*E*	− 1.55°					
	9.1			Salt and ice, −1°	− 1.9°		
KSCN—H_2O	60.0			Water and salt, 10.8°	−23.7°		
NH_4Cl—H_2O	18.7*E*	−15.8°					75.0*
	20.0			Salt and ice, −1°	−15.4°		
NH_4NO_3—H_2O	16.6	− 6°		Water and salt, 0°	−14.0°	12.2	2.6
				Ice and salt, 0°		78.8	73.6
	31.0			Ice and salt, −1°	−16.75°		
	31.2	−12°		Water and salt, 0°	−26.0°	19.7	6.8
				Ice and salt, 0°		74.6	65.6
	37.5			Water and salt, 13.6°	−13.6°		
	41.2	−17.4°					68.4*
	43.3*E*	−17.5°		Water and salt, 0°	−33.9°	24.3	8.2
				Ice and salt, 0°		69.5	57.1
	46.8	−12°	Salt separates	Water and salt, 0°	−36.4°	25.5	13.6
				Water and salt, 20°			3.1
				Ice and salt, 0°		68.1	59.8
	50.3	− 6		Water and salt, 0°	−39.3°	26.5	19.0
				Water and salt, 20°			8.9
				Ice and salt, 0°		66.2	62.1

Table 54. Temperatures of Freezing Mixtures—(*Continued*)

Substances	Composition of mixture (% anhydrous salt, unless otherwise stated). *E* = eutectic composition	Freezing point of solution		Initial condition of freezing mixture	Lowest attained temperature recorded	Heat absorbed at temperature of mixing, cal. per g. of mixture	Heat absorbed (at freezing or saturation point of solution) from objects to be cooled, cal. per g. of mixture. The * values are heats of fusion of the eutectic
NH_4NO_3—H_2O—(*Continued*)	54.1	0°	Salt separates	Water and salt, 0°	−42.2°	27.6	24.3
				Water and salt, 20°			14.5
				Ice and salt, 0°		64.4	64.4
	57.1	5°		Water and salt, 0°	−44.7°	28.4	28.4
				Water and salt, 20°			18.8
NH_4SCN—H_2O	57.1			Water and salt, 13.2°	−18.0°		
Ca_2Cl_26HO—H_2O	% hydrated salt 16.9	− 4.0°		Ice and salt, 0°		69.9	66.2
	26.8	− 8.1°		Ice and salt, 0°		63.8	57.3
	34.6	−12.4°		Ice and salt, 0°		59.3	50.2
	45.7	−22.7°		Ice and salt, 0°		53.0	38.4
	54.9	−39.9°		Ice and salt, 0°		48.0	26.0
	58.8*E*	−54.9°		Ice and salt, 0°		45.8	17.7
	63.7	−33.3°		Ice and salt, 0°		43.7	27.9
				Water and salt, 0°		14.4	None
	67.1	−19.7°		Ice and salt, 0°		41.9	33.2

			Salt separates	Water and salt, 0°		15.4	6.7
	69.0	−14.1°		Ice and salt, 0°		41.0	35.0
				Water and salt, 0°		16.0	10.1
				Water and salt, 20°		None	1.5
	74.1	0°		Ice and salt, 0°		38.7	38.7
				Water and salt, 0°		17.7	17.7
				Water and salt, 20°		None	10.2
	77.5	7.6°		Water and salt, 0°		19.0	21.6
				Water and salt, 20°		None	14.7
$MgSO_4.12H_2O—H_2O$	% anhyd. salt 19.0	− 3.9°				58.2	
$CuSO_4.5H_2O—H_2O$	11.9	− 1.6°				69.0	
$ZnSO_4.7H_2O—H_2O$	27.2	− 6.55°				50.9	
$FeSO_4.7H_2O—H_2O$	13.0	− 1.8°				67.2	
66.19 % $H_2SO_4—H_2O$	% of 66.19 % H_2SO_4 7.1			Ice and acid, 0°	−16°	− 2.1°†	68.6
	11.2			Ice and acid, 0°	−20°	− 3.1°†	62.0
	17.2			Ice and acid, 0°	−24°	− 5.5°†	52.9
	23.9			Ice and acid, 0°	−28°	− 9.5°†	43.0
	33.6			Ice and acid, 0°	−32°	−16.5°†	24.5
	44.2			Ice and acid, 0°	−36°	−30.2°†	7.5
	47.7			Ice and acid, 0°	−37°	−37°†	0
$HCl—H_2O$	% HCl 24.8*E*	−86°					

---

† Temperature when all ice is melted.

Table 54. Temperatures of Freezing Mixtures—(*Concluded*)

Substances	Composition of mixture (% anhydrous salt, unless otherwise stated). E = eutectic composition	Freezing point of solution	Initial condition of freezing mixture	Lowest attained temperature recorded	Heat absorbed at temperature of mixing, cal. per g. of mixture	Heat absorbed (at freezing or saturation point of solution) from objects to be cooled, cal. per g. of mixture. The * values are heats of fusion of the eutectic
$Na_2SO_4.10H_2O$—36.69% HCl	% $Na_2SO_4.10H_2O$ 21.05		0°		6.09	
	30.33		0°		9.17	
	36.59		0°		11.15	
	37.69		21.2°	− 8.1°		
	42.37		0°		13.15	
	50.22		21.6°	−12.2°		
	62.67		15°			21.2 at 0° 12.0 at −15°
	62.96		21.6°	−15.3°		
	63.88		0°		28.89	
	74.64		15°			30.6 at 0° 19.1 at −15°
	74.68		0°		30.85	
	75.30		21.5°	−14.8°		
	78.90		0°		27.43	
	86.63		15°			24.5 at 0° 13.4 at −15°

	86.72		0°		19.44	
	88.53		20.1°	−15.6°		
$Na_2SO_4.10H_2O$—30.13% HCl	% $Na_2SO_4.10H_2O$ 46.04		19.7°	−11.8°		
	49.74		19.7°	−11.8°		
	63.46		19.7°	−14.4°		
	65.23		20.4°	−15.6°		
	75.43		20.0°	−14.8°		
	82.54		19.9°	−17.2°		
	86.31		20.0°	−12.6°		
	89.88		20.4°	*ca.* 0°		
$Na_2SO_4.10H_2O$—24.47% HCl	% $Na_2SO_4.10H_2O$ 35.54		0°		12.67	
	38.16		19.9°	− 8.2°		
	50.42		19.8°	−10.0°		
	62.22		0°		26.84	
	63.86		20.5°	−12.0°		
	67.57		0°		27.18	
	71.46		0°		25.72	
	75.36		21.0°	−11.8°		
	78.40		0°		20.21	
$C_2H_5OH—H_2O$	% alcohol 50	−37°	Alc., 2°; ice, 0°	−24.2°		
			Alc., 1.5°; ice, −1°	−29.4°		
	51.3	−38°	Alc., 4°; ice, 0°	*ca.* −30°		
$CS_2—(CH_3)_2CO$	A temperature of −43.5° in a volume of 20 cc. was maintained by mixing 100 cc. carbon disulfide and 70 cc. acetone per hour, using a heat interchanger.					

Table 55. Cooling Mixtures

Salts	Temperature produced by mixing salts with water	Reduction of temperature produced by water with an equal weight of a salt or of a mixture of salts in equal parts	Salts	Temperature produced by mixing salts with water	Reduction of temperature produced by water with an equal weight of a salt or of a mixture of salts in equal parts
NH_4Cl	....	14°	KCl—NH_4NO_3	....	20°
$NaCl$	....	4°	NH_4NO_3—KNO_3	....	22°
KCl	....	12°	NH_4NO_3—$Na_2SO_4.10H_2O$	−19.5°	26°
NH_4NO_3	....	25°	Na_2NO_3—$NaSO_4.10H_2O$	....	10°
$NaNO_3$	....	9.5°	$NaNO_3$—$KCNS$	−37.4°	
KNO_3	....	10°	KNO_3—NH_4CNS	−28.2°	
NH_4SO_4	....	8°	NH_4Cl—NH_4NO_3—KNO_3	−22.6°	
$Na_2SO_4.10H_2O$	....	7.5°	NH_4Cl—NH_4NO_3—$NaNO_3$	−30.1°	
K_2SO_4	....	4.5°	NH_4Cl—$Na_2SO_4.10H_2O$—KNO_3	....	17°–23°
NH_4Cl—KNO_3	−18.2°	20°	NH_4Cl—$(NH_4)_2SO_4$—K_2SO_4	−15.2°	
NH_4Cl—$NaNO_3$	−31.5°	17°	NH_4Cl—$(NH_4)_2SO_4$—$Na_2SO_4.10H_2O$	−19.9°	
NH_4Cl—NH_4NO_3	....	22°	$NaCl.2H_2O$—$NaNO_3$—KNO_3	−24.6°	
NH_4Cl—$Na_2SO_4.10H_2O$	−17.6°	19°	KCl—KNO_3—K_2SO_4	−11.55°	
NH_4Cl—K_2SO_4	−18.0	.....	NH_4NO_3—KNO_3—$NaNO_3$	....	16°–27°
$NaCl$—KNO_3	....	10°	NH_4NO_3—KNO_3—$Na_2SO_4.10H_2O$	....	17°–26°
$NaCl.2H_2O$—KNO_3	−24.9°	.....	NH_4NO_3—$(NH_4)_2SO_4$—$Na_2SO_4.10H_2O$	−19.5°	
KCl—$NaNO_3$	....	11°			

precooled air at the same pressure, and further compressed to a high pressure p_3. The high-pressure air is dried and passed through a heat exchanger *HE* where it is further cooled. The throttling valve V_1 cools the air by the Joule-Thomson effect. Part of the air liquefies, and the rest returns to the high-pressure compressor. The liquid air from the intermediate pressure passes through a reducing valve to atmospheric pressure and boils off enough air to drop the temperature to the boiling point at atmospheric pressure where it may be collected in containers. In order to utilize heat exchange to the utmost, the tubes carrying the gases are gathered together in such a way that the low-pressure air is on the outside and the high-pressure air on the inside, as shown in Fig. 36.

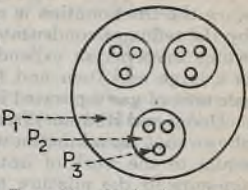


FIG. 36.—Linde heat exchanger.

When only small quantites of liquid air are needed, a Hampson spiral may be used. This consists of layers of one or more spirally wound copper tubes. High-pressure air is carried through these spirals and through a throttling valve, the uncondensed air returning through the free space around the spirals.

The Linde process can be applied only to gases which cool on expansion by throttling. The cooling effect is more pronounced the lower the temperature from which the expansion starts, and precooling by conventional refrigeration systems is therefore advantageous. Helium and hydrogen warm on expansion at normal temperatures and are therefore usually prepared in liquid form by auxiliary cooling with liquid nitrogen at low pressure. Preparation of liquid helium or hydrogen is, however, of little commercial interest.

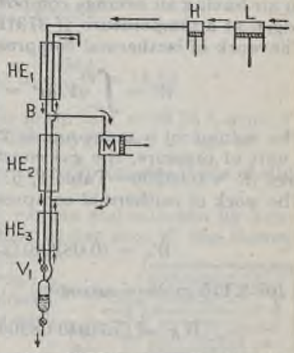


FIG. 37.—Claude process for liquid air.

L = low-pressure compressor.
H = high-pressure compressor.
HE_1, HE_2, HE_3 = heat exchangers.
M = expansion engine.
B = tee for by-passing part of air through expansion engine.
V_1 = throttling valve for final Joule-Thomson expansion to produce liquid air.

In the Claude (or Heylandt) process the compressed gas is made to perform work in an expansion engine in much the same way as is the air in a dense-air refrigerating machine. The work done by the expansion engine could theoretically be used for recompression but is so fluctuating as to be of little value and is usually dissipated in a brake or an electrical resistance. At very low temperatures lubrication is a serious problem. Liquid pentane is used in the more modern machines. The expansion engine is used not to drop the temperature to the point of condensation but only to precool the gas just above this point. The final liquefaction is by a Joule-Thomson expansion. A diagram of the Claude system is shown in Fig. 37.

Although the Claude process seems inherently more efficient than the Linde process, the power requirements are in practice about equal. For the

Linde system they are 0.6 to 1.6 hp.-hr. per lb. of oxygen and for the Claude system 0.77 hp.-hr. for oxygen of 88 per cent purity.

The separation of gases by liquefaction is essentially a process of fractional distillation and follows the same general principles (see Sec. 12). However, since the fractionation is carried out at low temperatures, the heat given off by the reflux or condensate must be raised from a low to a high temperature, which involves an expenditure of work. The cost of supplying this work is a large cost item and for that reason the horsepower-hour requirements per unit of gas separated is an important consideration.

Dodge and Housem [*Trans. Am. Inst. Chem. Engrs.*, **19**, 117–151 (1927)] have shown that the minimum work required to separate two gases by liquefaction is equal to the work of isothermal compression for each gas from its partial pressure in the mixture to the total pressure. In the specific case of air, the minimum work requirement is the isothermal compression of 21 volumes of oxygen from a partial pressure of 0.21 to 1.00 atm. and of 79 volumes of nitrogen from 0.79 to 1.00 atm. In making this calculation, it is important to use the correct conversion factor, and the detailed calculation follows based on the separation of 1 mol of pure oxygen and 3.76 mols of nitrogen from air having an average composition of 21 per cent oxygen and 79 per cent nitrogen at a temperature of 273°K. (°C.).

The work of isothermal compression is given by the formula

$$W = \int_{p_2}^{p_1} V\,dP = \int_{p_2}^{p_1} RT\,d\ln P = RT\ln\frac{p_1}{p_2}$$

The volume of a gram mol is 22.4 l. Since we are using atmospheres as the unit of pressure, the gas constant R should be expressed in liter-atmospheres ($R = 0.08206$—Table 1, p. 617).

The work of isothermal compression for 1 mol of oxygen is:

$$W_0 = (0.08206)(273)\ln\frac{1.00}{0.21} = 34.95 \text{ l.-atm.}$$

and for 3.176 mols of nitrogen:

$$W_N = (3.76)(0.08206)(273)\ln\frac{1.00}{0.79} = 20.25 \text{ l.-atm.}$$

The total work $= W_0 + W_N = 55.20$ l.-atm.

1 l.-atm. = 0.00003774 hp.-hr. (p. 104)
1 cu. ft. = 28.32 l.

Therefore, the work requirements per 1000 cu. ft. of oxygen and 3760 cu. ft. of nitrogen are:

$$(55.20)(0.00003774)\left(\frac{28.32}{22.4}\right)(1000) = 2.645 \text{ hp.-hr.}$$

This is the *minimum* work requirement. Dodge and Housem conclude that the efficiency is about 10 per cent of the ideal, and the actual requirements are therefore ten times larger.

If oxygen of 80 per cent purity were required, the work expenditure would be somewhat less, and the calculation may be carried out as before, though it is a little more involved.

Initial conditions: $P_{O_2} = 0.21$ atm.

and

$$P_{N_2} = 0.79 \text{ atm.}$$

Final conditions:

$$\text{Oxygen rich fraction: } P_{O_2} = 0.80 \text{ atm.}; \qquad P_{N_2} = 0.20 \text{ atm.}$$

$$\text{Nitrogen rich fraction: } P_{O_2} = \frac{0.20}{3.76} \text{ atm.}; \qquad P_{N_2} = \frac{3.56}{3.76} \text{ atm.}$$

There are now four work terms, two of which will be negative numerically, since the nitrogen in the oxygen fraction will be compressed from a high to a low pressure and similarly for the oxygen in the nitrogen fraction.

Oxygen fraction:

$$W_{O_2} = (0.8)(0.08206)(273) \ln \frac{0.80}{0.21} = 24.00$$

$$W_{N_2} = (0.2)(0.08206)(273) \ln \frac{0.20}{0.79} = -6.16$$

Nitrogen fraction:

$$W_{O_2} = (0.2)(0.08206)(273) \ln \frac{0.0532}{0.21} = -6.16$$

$$W_{N_2} = (3.56)(0.08206)(273) \ln \frac{0.948}{0.79} = 14.58$$

$$\text{Net work per gram mol} = 26.26 \text{ l.-atm.}$$

$$\text{Net work per 1000 cu. ft. of 80 per cent oxygen} = 1.26 \text{ hp.-hr.}$$

There is thus an appreciable saving in power requirements if gas of slightly inferior purity will serve the purpose.

For the separation of gases boiling below oxygen and nitrogen by liquefaction, the actual efficiency would be less than 10 per cent of the theoretical prevailing for the separation of air, because the heat of refluxing has to be elevated through a much wider temperature interval. Conversely, gases of higher boiling point can be separated with somewhat greater efficiency.

In processes for the separation of hydrogen from coke oven gas, or helium from natural gas, it is only necessary to condense the unwanted constituents. The hydrogen and helium do not require liquefaction.

The most important practical problem in gas liquefaction and separation is to reduce to a minimum the temperature difference at which heat is exchanged. All the heat transfer surfaces should be clean. Heat losses to the surroundings should be reduced by good insulation, and adequate heat transfer surface should be provided.

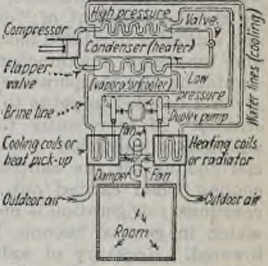


Fig. 38.—Heat-pump installation.

The Heat-pump Principle. In a mechanical refrigeration system, heat is abstracted in one part of the cycle at a low temperature and transferred in another part of the cycle to a higher temperature. While mechanical refrigeration is primarily designed to produce a temperature lower than that of the environment, it can also be used to produce a higher temperature. The first proposal of this sort was made by Lord Kelvin, who showed that a more efficient utilization of heat for warming could be obtained if it were applied to a steam engine operating between steam temperature and outside atmospheric temperature, driving a refrigerating machine operating between outside atmospheric temperature and inside room temperature. This proposal has acquired considerable interest in the last few years [Stevenson, Faust, and Roessler, *Refrig. Eng.*, **23**, 83 (1932)]. To illustrate the principle, a gas engine may be considered. This has a normal over-all efficiency of about 30 per cent in the conversion of **heat into work**. A refrigerating machine operating between an outside freezing temperature (32°F.) and normal room temperature (68°) has an over-all efficiency of about 900 per cent for the conversion of **work into heat**. Accordingly 3000 B.t.u. in the gas engine will produce 900 B.t.u. of work, which will raise 8100 B.t.u. of heat from 32°F. to 68°F. If the outside temperatures are at 0°F., the refrigeration efficiency will drop to about half. Moreover, if heat is distributed by the ordinary type of radiator, it will have to be raised to a higher temperature level than 68°F. Nevertheless, assuming electric power at 2 cts. per kw.-hr. for driving the refrigeration unit and gas to be used in a gas furnace at 75 cts. per 1000 cu. ft., the cost of heating by the heat pump is only 4 per cent greater in New Orleans, Charleston, S.C., and Los Angeles, while it becomes 30 per cent greater in Washington, D.C. (Stevenson *et al.*, *loc. cit.*). With lowered costs of electric power and the advantage that it can be used in summer for air conditioning, the Kelvin warming engine may find considerable development in the future. An installation which has been tried in practice is shown in Fig. 38 [Chawner, *Electric West*, **66**, 177 (1931)] abstracted in *Mech. Eng.*, **53**, 686 (1931).

A further application of the heat-pump principle is found in the vapor recompression systems of evaporation (see Sec. 8, p. 1071).

Uses of Refrigeration. Refrigeration is a comparatively new art, having received its first real impetus with the development of the ammonia compression machine by Linde in 1873. The first important applications were in the food industries, which have in recent years been further extended by the development of "quick freezing" methods. The brewing industry and a number of related biochemical industries, such as baking, candy making, and ice-cream manufacture, demand large tonnages. The necessity of regulating the temperature and humidity of air in chemical industries has given a decided impetus to air conditioning, a field closely related to refrigeration. More recently the strictly chemical industries have extensively adopted refrigeration processes, and most chemical plants employ artificial cooling for the control of delicate chemical operations which require cooling when carried out in the laboratory. Aside from the control of reactions, refrigeration is most commonly used for the separation of materials which in general become less and less compatible as the temperature is lowered. Recovery of sodium nitrate from caliche, wax separation in oil refining, recovery of condensable vapors (casing-head gasoline in natural gas, SO_2 in the sulfuric acid manufacture, ammonia in the Haber process, nitrogen oxides in the oxidation of ammonia to nitric acid), "salting out" in soap manufacture—all are examples of diverse processes which are based on this general principle. Most of the details of actual processes are kept more or less secret, but it is fairly evident that the art of refrigeration is an important tool for the chemical engineer.

SECTION 24

ELECTRICITY AND ELECTRICAL ENGINEERING

BY

R. G. Warner, Ph. B., E. E., Engineer, United Illuminating Co., New Haven Conn.; Member, American Institute of Electrical Engineers.

F. T. McNamara, Ph. B., E. E., Associate Professor of Electrical Engineering, Yale University; Associate Member, Institute of Radio Engineers; Associate Member, American Institute of Electrical Engineers.

CONTENTS

ELECTRICITY AND ELECTRICAL ENGINEERING

DEFINITIONS—UNITS*

The **electron** is the elementary charge or quantity of electricity.

Any ultimate carrier of electricity is called an **ion.** It may be an electron, or a charged atom, molecule or group of molecules.

Current (I, i) is the rate of flow of electricity. The unit, the **ampere** (amp.) is equal to a flow of 1 coulomb (6.285×10^{18} electrons) per second. The fundamental measurement of the ampere is determined by the deposition of silver from a silver nitrate solution (see p. 2723).

Potential difference (E, e, or V, v) or **electromotive force** (e.m.f.) is the force or pressure causing current to flow in a circuit. The unit, the **volt,** is based on a standard primary cell. The "normal" Weston cadmium cell has a potential of 1.0183 volts at 20°C. (see p. 2725). Electromagnetically, 1 volt is induced in a conductor cutting magnetic flux at the rate of 10^8 lines per second.

Resistance (R, r) is the property of a circuit (or substance) opposing the flow of current and causing heat when current flows. The unit, the **ohm** (Ω), is the resistance which limits the current, caused by 1 volt, to 1 amp. One ampere flowing through a 1-ohm resistance produces heat at the rate of 1 watt (10^7 ergs per sec.).

Resistivity (ρ) is the specific resistance of a substance usually expressed in ohms per centimeter cube, or ohms per circular-mil-foot. A circular-mil-foot is a conductor having a length of 1 ft. and a uniform cross section which is 0.001 in. in diam. The resistance of a cir.-mil-ft. of copper at 20°C. is 10.371 ohms, and at 60°C., 12.0 ohms.

Conductance (G, g) is the property of a circuit which permits the flow of electricity. The unit, the **mho,** is the reciprocal of the ohm.

Inductance (L) is a circuit property expressing the ratio of *flux linkages* (total flux times number of turns) to the current producing the flux. The unit, the **henry,** equals 10^8 flux linkages per ampere. *Self-inductance* is this ratio referring to flux linkages with the circuit producing the flux; *mutual inductance* is the ratio referring to flux linkages with one circuit caused by current in a neighboring circuit.

Capacitance (C) expresses the ability of a circuit to hold or store electric charges. The unit, the **farad,** is the capacitance which will store 1 coulomb for each volt. Commonly the **microfarad** is used, which is one-millionth of a farad.

Dielectric constant (ϵ) is the specific inductive capacity of a material. It is equal to the ratio of the capacitances of two condensers of identical size: one using the particular dielectric, the other using air or a vacuum as the dielectric.

Dielectric strength (S) is the rupturing strength of an insulating material when subjected to voltage stress. It is usually expressed in volts per mil or volts per millimeter or multiples thereof. Actually the breakdown varies with the shape of the electrodes and does not increase in proportion to the thickness of the material.

Frequency (f) is the number of cycles of alternating current per second.

---

*For more complete definitions see "Report on Proposed American Standard Definitions of Electrical Terms," American Institute of Electrical Engineers, August, 1932.

Reactance (X) is the property of alternating-current circuits opposing the flow of current due to inductance or capacitance. The unit is the **ohm.** The reactance is equal to $2\pi fL$ for *inductive circuits* and to $1/(2\pi fC)$ for *capacitive circuits.* When inductive reactance, X_L, and capacitive reactance, X_C, are both present, their effects oppose one another and, in a series circuit, the total $X = X_L - X_C$.

Impedance (Z) is the property of alternating-current circuits opposing the flow of current, combining the effects of resistance and reactance. The unit, the **ohm,** is the impedance requiring 1 volt to cause 1 amp. to flow: $Z = \sqrt{R^2 + X^2}$

Admittance (Y) is the reciprocal of impedance; the unit is the **mho.**

Magnetic flux (ϕ) is the magnetic flow or lines of magnetic force through a magnetic circuit. The unit, the **maxwell,** is one line.

Flux density (B) is the flux per unit area. The unit, the **gauss,** is 1 maxwell per square centimeter. Flux density is also expressed in lines per square inch.

Magnetomotive force (m.m.f.) is the force which causes flux in a magnetic circuit. The unit, the **gilbert,** is equal to 1.257 amp.-turns. Frequently magnetomotive force is expressed in ampere-turns.

Reluctance ($\mathcal{R}$) is the magnetic resistance to flux. The unit reluctance limits the flux to 1 maxwell with a m.m.f. of 1 gilbert.

Magnetic field intensity (H) is the m m.f. per unit length of path usually expressed in gilberts (or ampere-turns) per centimeter (or per inch).

Permeability (μ) is the magnetic conductivity of a material. The permeability of air and many other substances is unity.

Power (P, p) is the rate of supplying or developing energy. The unit, the **watt,** is the power delivered by 1 volt with 1 amp. flowing (unity power factor). One horsepower equals 746 watts; 1 kw. = 1.34 h.p.

Energy (W, w) is the capacity for doing work. The unit, the **watthour** is the energy supplied or developed by 1 watt in 1 hr. The **kilowatt-hour,** 1000 watthours, is the common unit for billing energy (occasionally but erroneously this is abbreviated to the form kilowatt).

Table 1. Electrical and Magnetic Units

Quantity	Symbol	Practical unit	m*	S†
Charge	q	Coulomb	10^{-1}	3×10^9
Current	I, i	Ampere	10^{-1}	3×10^9
Electromotive force	E, e	Volt	10^8	$\frac{1}{3} \times 10^{-2}$
Resistance	R, r	Ohm	10^9	$\frac{1}{9} \times 10^{-11}$
Inductance	L	Henry	10^9	$\frac{1}{9} \times 10^{-11}$
Capacitance	C	Farad	10^9	9×10^{11}
Magnetic flux	ϕ	Maxwell	1	$\frac{1}{3} \times 10^{-10}$
Flux density	B	Gauss	1	$\frac{1}{3} \times 10^{-10}$
Power	P, p	Watt	10^7	10^7
Energy	W, w	Watt-second	10^7	10^7
		Watthour	36×10^9	36×10^9

---

* For a given quantity, m is the ratio of the number of electromagnetic units to the number of practical units, or m is the ratio of the magnitude or size of a practical unit to the magnitude of an electromagnetic unit.

† For a given quantity, S is the ratio of the number of electrostatic units to the number of practical units, or S is the ratio of the magnitude or size of a practical unit to the magnitude of an electrostatic unit.

Volt-amperes (*Va*), frequently called the **apparent power**, is the product of volts and amperes in a circuit. At unity power factor, the volt-amperes equal the watts.

Power factor (P. F.) is the ratio of the power to the volt-amperes and hence must be equal to, or less than, 1.0. The volt-amperes in an alternating-current circuit are frequently greater than the watts due to the effect of inductance or capacity: $\text{P.F.} = W/Va = \cos\theta$, where θ is the angular phase difference between voltage and current.

Reactive volt-amperes (*vars*) is the component of the volt-amperes not producing power: $Vars = EI \sin\theta$ and the total $Va = \sqrt{w^2 + vars^2}$.

ELECTRICAL CIRCUITS

Each circuit to carry current must be completed from source through apparatus and back to source.

Direct current or continuous current is unidirectional and practically non-pulsating, although there may be some variation due to the commutators of generators.

Ohm's law for any part or parts of a circuit:

$$E = IR;\quad I = \frac{E}{R};\quad R = \frac{E}{I}$$

In a **series circuit** (Fig. 1):

$$I_1 = I_2 = I_3, \text{ etc.}$$
$$\Sigma E = E_1 + E_2 + E_3 + \cdots E_n$$
$$\Sigma R = R_1 + R_2 + R_3 + \cdots R_n$$

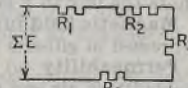


FIG. 1.

In a **parallel circuit** (Fig. 2):

$$E_1 = E_2 = E_3, \text{ etc.}$$
$$\Sigma I = I_1 + I_2 + I_3 + \cdots I_n$$
$$\sum\frac{1}{R} = \frac{1}{R_1} + \frac{1}{R_2} + \frac{1}{R_3} \cdots \frac{1}{R_n}$$

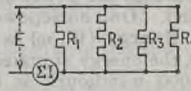


FIG. 2.

In **combination circuits**, compute equivalent values for each group and then combine group values.

Kirchhoff's laws: (1) The algebraic sum of the currents at any point must equal zero; or the currents coming into a point must equal the currents leaving. (2) The algebraic sum of the voltages about a complete loop in a circuit must equal zero; or between any two points the voltage must be the same as determined over each path.

Resistances:

$$R = \frac{\rho l}{a}$$

where l is length and a is area, in same terms as resistivity. ρ is nearly constant for any given material except for temperature changes. For different materials resistivity ranges from microhms to megohms per centimeter cube (Tables 2 and 3):

$$\rho_t = \rho_{20}[1 + \alpha(t - t_{20})]$$

or

$$\rho_t = \rho_0[1 + \alpha(t - t_0)]$$

where ρ_t is the resistivity at any temperature t; ρ_{20} is the resistivity at 20°C.; α is the temperature coefficient of resistance and is equal to 0.00393 for copper at 20°C. (Table 2); and ρ_0 is the resistivity at 0°C.

The **effective resistance** in alternating-current circuits increases at higher frequencies due to non-uniform distribution of currents and due to losses.

Table 2. Conductors and Resistances

Material	Resistivity, microhms per cm. cube, ρ_{20}	Temperature coefficient, α_{20}	Maximum working temperature, °C.
Aluminum	2.82	0.00421	
Carbon (filament)	4050	−0.00054	
Graphite	1365	−0.00028	
Copper	1.72	0.00393	
Iron	9.78	0.00634	
Lead	20.63	0.00367	
Mercury	95.9	0.00089	
Nickel	7.24	0.00491	500
Platinum	10.6	0.00369	
Silver	1.62	0.00361	
Tin	11.4	0.0040	
Tungsten	5.75	0.00454	
Zinc	5.92	0.00325	
Alloy:			
Advance	48–49	0.00001 to 0.00002	500
Alumel	33.3	0.0012	
Brass	5.7	0.0014 to 0.002	
Bronze	13–18	0.0005	
Calido	110	0.00012	1000
Cast Iron	57–114		
Chromel	70–110	0.00011 to 0.00054	
Climax	87	0.00067	600
Constantan	47–51	−0.00004 to +0.00001	
Duralumin	3.35		
German silver	17–41	0.00004 to 0.00038	
Ideal	49	0.000005	500
Invar	75	0.002	
Karma	103	0.0001	1150
Lucero	46	0.00076	500
Manganin	34–100	−0.00003 to −0.00002	100
Monel	42.5–45	0.00002 to 0.002	900
Nichrome	110	0.0004 to 0.00003	1000
Nichrome II	109–111	0.00015	
Nichrome III	90–97	0.00005 to 0.00019	1100
Nichrome IV	98–103	0.00018	1150
Phosphor bronze	2–12	0.003 to 0.004	
Swedish iron	20	0.0012	
Welding iron	18	0.006	

Note. The resistivity and temperature coefficients are based on data in the "International Critical Tables," vol. 6, p. 159, McGraw-Hill, and are subject to variation with the commercial composition of the materials and with the heat treatment.

Power in a direct-current circuit is equal to the product of volts and amperes:

$$W = EI = I^2R = \frac{E^2}{R}$$

Energy is the power for a given time or $I^2Rt = EIt$.

Table 3. Dielectric Properties

Resistivity ρ, dielectric strength S for samples of given thickness and the dielectric constant ϵ of materials used as insulators

Material	ρ, ohms per cm. cube	S, kv. per mm.	Thickness of sample, mm.	ϵ
Asbestos paper	2×10^6	4	1	2.7
Cellulose	10^9	....	....	3.9–7.5
Enamel	10^{14}	20–25	0.02	
Fiber, vulcanized	5 to 20×10^9	8–18	1	5–7.5
		3–6	12	
Fish paper	....	10–15	0.1–1.2	
Glass	10^{10} to 10^{14}	230	1.4	5–8.5
Pyrex	10^{14}	134	6.35	4.8
Maple, paraffined	3×10^{10}	4.5	15	4.1
Marble	10^9 to 10^{11}	2–4	25	8.3
Mica	10^{15} to 10^{17}	80–200	0.05	2.15–2.5
Oil, transformer	2 to 15×10^{13}	10–25	3.81	
Paraffin	10^{16} to 5×10^{18}	15–50	....	1.9–2.3
Petrolatum	2 to 10×10^{12}	20	2.5	2.2
Phenolic insulating materials, molded	10^{11} to 10^{12}	9–280	....	5.0–7.5
Porcelain	10^{12} to 10^{15}	12–28	5	4.4–6
Pressboard	10^9	12–5	0.2–3.0	
Rubber, hard	3×10^{13}	10–38	1–0.5	3
Rubber, soft vulcanized	1 to 15×10^{15}	....	....	2.7
Silica, fused	5×10^{17}	....	....	3.5
Slate	10^8	0.2–0.4	25	6.0–7.5
Soapstone	6×10^8	1.0	25	
Varnished cambric	....	60–45	0.1–0.4	3.5–5.5
Woods, hard, dry	10^{10} to 10^{13}	0.4–0.6	25	3.0

Note. These values are based on data in the "International Critical Tables," vol. 2, pp. 310–311, McGraw-Hill, and are subject to considerable variation with temperature and moisture conditions. The dielectric strength decreases with thicker pieces of material.

Alternating current varies periodically in magnitude and direction. Commercial alternating current approximates the **sine wave** and in this discussion sine waves will be assumed.

$$e = E_{max} \sin 2\pi ft$$

where e is the instantaneous value, f is frequency, t is time in seconds and $2\pi ft$ is the angle in radians.

The **effective value** of alternating current (or voltage) gives the same heating effect in a resistance circuit as the corresponding direct current. Inasmuch as the heating is proportional to the squared value, the effective value of alternating current is obtained by taking the square root of the average of the instantaneous squared values. This is referred to as the **root-mean-square value** and is designated as E or I. For sine waves, $E = 0.707E_{max}$ and $I = 0.707I_{max}$.

Average value of alternating current or voltage over a complete cycle is zero. The average value of a half cycle of a sine wave is $2/\pi$ times the maximum, hence $E_{av} = 0.636E_{max}$. This value has little importance. It is used with the effective value as a ratio to designate the form factor. Form factor is $E_{eff.}/E_{av.}$. For a sine wave the form factor is 1.11.

In non-inductive circuits the current is in phase with the voltage (Fig. 3), *i.e.*, it reaches a maximum value at the same time as the voltage and passes through zero with the voltage. When the circuit (or portion of the circuit) contains resistance only, the current is in phase with the voltage and the following relations hold:

$$e = iR; \quad E_{max} = I_{max}R; \quad E = IR$$

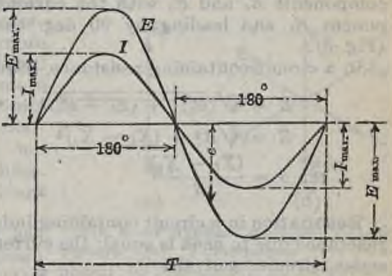


FIG. 3.—Sine waves; non-inductive circuit. (*"Standard Handbook for Electrical Engineers,"* *McGraw-Hill*, 1933.)

Inductive circuits require additional voltage; since the alternating current produces alternating flux and this change of flux induces a voltage, it is necessary to impress additional voltage (for a given current) to overcome the induced voltage. For a circuit containing inductance only, the current is out of phase with the voltage lagging it by a quarter cycle or 90 deg. If

$$i = I_{max} \sin 2\pi ft$$

then

$$e = 2\pi fLI_{max} \cos 2\pi ft = E_{max} \cos 2\pi ft$$

where $2\pi fL$ is the inductive reactance (X), and $XI_{max} = E_{max}$ and $XI = E$.

In a circuit containing resistance and inductance, the current and voltage are not in phase, the current lags the voltage by an angle θ, where $\theta = \tan^{-1} X/R$. The e.m.f. may be divided into components E_r and E_l, where the current is in phase with the resistance component E_r and lags by 90 deg. the inductance (reactive) component E_l (Fig. 4).

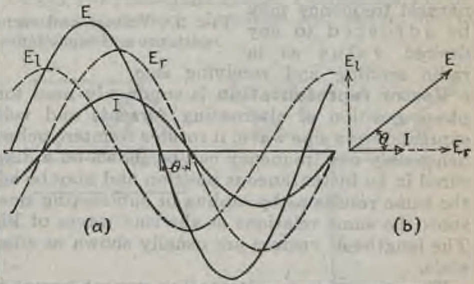


FIG. 4.—Voltage and current relations in series circuit of resistance and inductance. (*a*) Sine waves; (*b*) vectors.

Capacitive circuits also have a phase difference between the voltage and current. Inasmuch as capacitance absorbs coulombs in proportion to the voltage, the rate of absorbing (the current) is proportional to the rate of change of voltage. Hence the current is maximum when the voltage is changing fastest (at zero) and the current leads the voltage by 90 deg. for a circuit containing capacitance only. If

$$i = I_{max} \sin 2\pi ft$$

then

$$e = -\frac{1}{(2\pi fC)} I_{max} \cos 2\pi ft = -XI_{max} \cos 2\pi ft$$

where $1/(2\pi fC)$ is the capacity reactance and $XI_{max} = E_{max}$ and $XI = E$. In a circuit containing resistance and capacity reactance, the current leads the voltage by an angle θ where $\tan \theta = X/R$. The e.m.f. may be divided into components E_r and E_c with the current in phase with the resistance component E_r and leading by 90 deg. the capacity (reactive) component E_c (Fig. 5).

In a circuit containing resistance, inductance, and capacitance,

$$E = \sqrt{(E_r)^2 + (E_l - E_c)^2} = \sqrt{(IR)^2 + (IX_l - IX_c)^2} = IZ$$

$$Z = \sqrt{R^2 + (X_l - X_c)^2}$$

$$\tan \theta = \frac{(X_l - X_c)}{R} \qquad \cos \theta = \frac{R}{Z}$$

Resonance in a circuit containing inductance and capacity exists when the reactance due to each is equal; the current is limited only by resistance in the series circuit, and the voltages across the reactances are equal and opposite and may be very large. Under this condition $f = 1/(2\pi\sqrt{LC})$ and is the natural frequency of the circuit. This is the frequency at which such a circuit tends to oscillate. By changing the value of L and/or C, the natural frequency may be adjusted to any desired value as in radio sending and receiving sets.

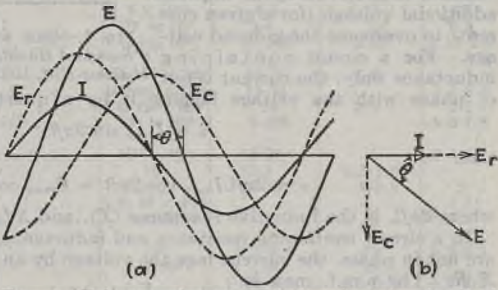


FIG. 5.—Voltage and current relations in series circuit of resistance and capacitance. (*a*) Sine waves; (*b*) vectors.

Vector representation is commonly used for indicating magnitude and phase position of alternating currents and voltages. Each vector is the expression of a sine wave; it rotates counterclockwise, one revolution per cycle, hence only one frequency can be shown on a diagram. The vectors are pictured in an instantaneous position and may be added or subtracted to obtain the same results as by adding or subtracting sine waves. Figures 4*b* and 5*b* show the same relations as the sine waves of Figs. 4*a* and 5*a*, respectively. The lengths of vectors are usually shown as effective values to an arbitrary scale.

The **energy** in an alternating-current circuit is the integrated product of the instantaneous voltage and current. The energy delivered to inductance or to capacitance is stored during a portion of the cycle and then returned; hence it does not affect the average energy. The energy for a period t is $EIt \cos \theta$. The angle θ is the phase displacement between E and I, and $\cos \theta$ is called the **power factor**.

Power in an alternating-current circuit is the instantaneous product of the voltage and current. Since e and i are sine waves, the product of the instantaneous values is a sine wave of double frequency, hence the power in a single-phase circuit is pulsating. The average value of power is

$$W = I^2R = \frac{EIR}{Z} = EI \cos \theta$$

Polyphase systems are commonly used because of better economy in generation and transmission, and permit more rugged motors with a uniform delivered torque. Such a system has two or more e.m.fs. displaced in phase by a definite amount with respect to one another. The common polyphase systems are three phase, two phase, and six phase. Figure 6 shows the displaced voltage vectors of these systems.

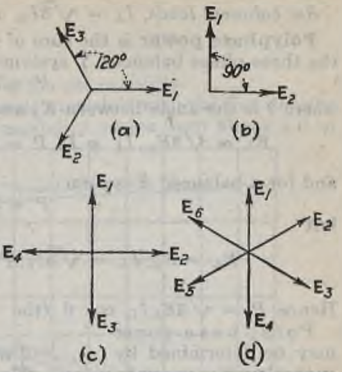


FIG. 6.—Polyphase systems—voltage vectors. (*a*) Three-phase; (*b*) two-phase; (*c*) four-phase; (*d*) six-phase.

A balanced polyphase system has equal loads in all the phases. Although in each phase the power is pulsating, the total power in a balanced polyphase system is constant.

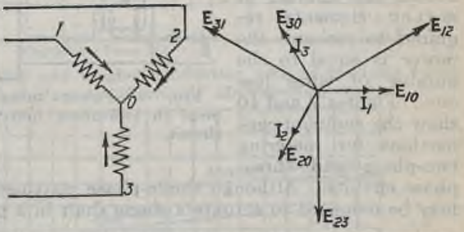


FIG. 7.—Three-phase system, Y-connection and corresponding vectors of voltage and current for 1.0 power factor.

Calculations for a polyphase system are normally made for each phase and then the results combined.

Three-phase Y system (Fig. 7). The line voltages are equal to the vector differences of the phase voltages. Using vector notation,

$$\dot{E}_{12} = \dot{E}_{10} - \dot{E}_{20}$$
$$\dot{E}_{23} = \dot{E}_{20} - \dot{E}_{30}$$
$$\dot{E}_{31} = \dot{E}_{30} - \dot{E}_{10}$$

For balanced phase voltages, $E_L = \sqrt{3}E_p, E_L =$ line voltage and $E_p =$ phase voltage. The line currents are the phase currents.

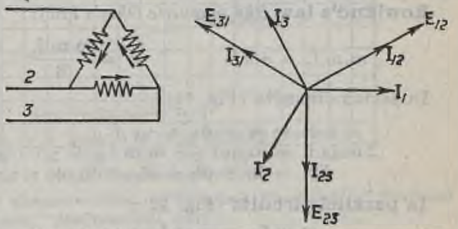


FIG. 8.—Three-phase system, delta-connection and corresponding vectors of voltage and current for 1.0 power factor.

Three-phase Delta System (Fig. 8). The line voltages are the phase voltages, $E_L = E_p$. The line currents equal the vector difference of the phase currents:

$$\dot{I}_1 = \dot{I}_{12} - \dot{I}_{31}$$
$$\dot{I}_2 = \dot{I}_{23} - \dot{I}_{12}$$
$$\dot{I}_3 = \dot{I}_{31} - \dot{I}_{23}$$

For balanced loads, $I_L = \sqrt{3}I_p$, I_L = line current and I_p = phase current.

Polyphase power is the sum of the power in the individual phases. For the three-phase balanced Y system

$$P = 3E_pI_p \cos\theta$$

where θ is the angle between E_p and I_p but

$$E_L = \sqrt{3}E_p,\ I_L = I_p,\ P = 3\left(\frac{E_L}{\sqrt{3}}\right)I_L \cos\theta = \sqrt{3}E_LI_L \cos\theta$$

and for a balanced Δ system

$$P = 3E_pI_p \cos\theta$$

but

$$E_L = E_p;\ I_L = \sqrt{3}I_p;\ P = 3E_L\left(\frac{I_L}{\sqrt{3}}\right) \cos\theta$$

Hence $P = \sqrt{3}E_LI_L \cos\theta$ (the same as in the Y system)

Polyphase power may be determined by **metering** the power in each phase. When the phases are interconnected the number of meter elements required to measure the power is equal to the number of wires less one. Figures 9 and 10 show the common connections for metering two-phase and three-phase circuits. Although single-phase wattmeters are shown, the elements may be mounted to actuate a single shaft in a polyphase wattmeter.

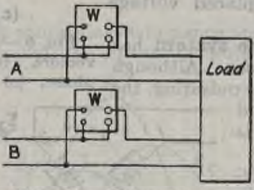


FIG. 9.—Power measurement in two-phase four-wire circuit.

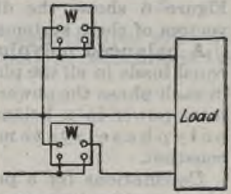


FIG. 10.—Power measurement in three-phase three-wire circuit or two-phase three-wire circuit.

MAGNETIC CIRCUITS

Flux lines are continuous throughout the magnetic circuit. Flux lines are represented as having direction from north poles to south poles externally. The poles are the areas where the flux lines leave or enter magnetic materials.

Rowland's law (*the magnetic Ohm's law*);

$$\text{m.m.f.} = \phi\mathcal{R}; \qquad \phi = \frac{\text{m.m.f.}}{\mathcal{R}}$$

In **series circuits** (Fig. 11):

$$\phi_1 = \phi_2 = \phi_3 = \cdots$$
$$\Sigma\text{m.m.f.} = \text{m.m.f.}_1 + \text{m.m.f.}_2 + \cdots$$
$$\Sigma\mathcal{R} = \mathcal{R}_1 + \mathcal{R}_2 + \mathcal{R}_3 + \mathcal{R}_4 + \cdots$$

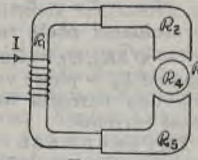


FIG. 11.

In **parallel circuits** (Fig. 12):

$$\text{m.m.f.}_1 = \text{m.m.f.}_2 = \cdots$$
$$\Sigma\phi = \phi_1 + \phi_2 + \cdots$$
$$\sum\frac{1}{\mathcal{R}} = \frac{1}{\mathcal{R}_1} + \frac{1}{\mathcal{R}_2} + \cdots$$

In **combination circuits**, compute equivalent values for each group and then combine group values.

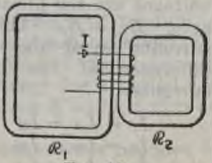


FIG. 12.

Reluctance.

$$\mathcal{R} = \frac{l}{\mu a}$$

where l is the length, a is the area, and μ is the permeability.

The permeability varies greatly in magnetic materials with the flux density (Fig. 13). In different materials the permeability varies from about 1.0 to 5000, or even to 10,000. In the case of special heat-treated alloys very high permeability (nearly 170,000) has been obtained [*Elec. J.*, **28**, 386 (1931); *J. Franklin Inst.*, **195**, 621 (1923), **206**, 503 (1928)], at relatively low field intensities. Commercial irons have a permeability of about 2000.

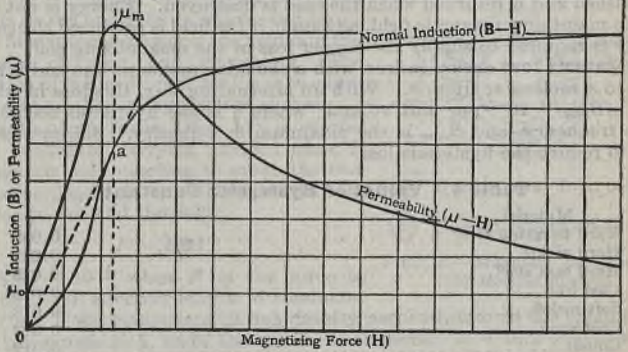


FIG. 13.—Variation of permeability and magnetic induction with magnetizing force. (*"Properties and Testing of Magnetic Materials,"* *McGraw-Hill*, 1927.)

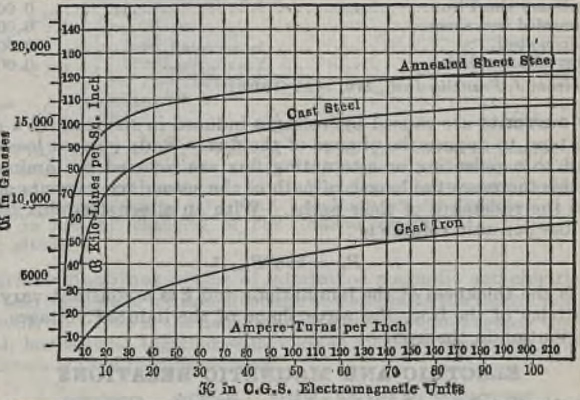


FIG. 14.—Normal magnetization characteristics. (*"Standard Handbook for Electrical Engineers,"* *McGraw-Hill*, 1922.)

With such a limited range of permeabilities the actual flux path is indefinite; there is a considerable leakage flux. This, coupled with the variation in permeability with flux density, makes it impossible to calculate magnetic circuits with the same precision as electric circuits. Most calculations involving magnetic materials use (B-H) magnetization curves directly rather than permeability (see Fig. 14). In electrical apparatus the iron is normally magnetized to a point near the knee (where the curve bends rapidly) of the magnetization curve.

Energy is stored *in the magnetic field;* it must be supplied when the field is established and is returned when the field is destroyed. Energy is not necessary to maintain a magnetic field, although, if the field is produced electrically, energy is required to supply the copper loss of the magnetizing coil.

Hysteresis loss occurs in iron with a changing magnetic flux and is attributed to a molecular friction. With an alternating flux, this loss in watts is $P_h = \eta f B_{max}^{1.6}\ 10^{-7}$ per unit volume, where η is the hysteresis coefficient, f is the frequency, and B_{max} is the maximum flux density. Silicon steels are used to reduce the hysteresis loss.

Table 4. Values of Hysteretic Constant*

Material	η
Hard tungsten steel	0.058
Hard nickel	0.039
Hard cast steel	0.025
Cast iron	0.013
Soft nickel	0.013
Cast steel	0.012
Cobalt	0.012
Electrolytic iron	0.009
Annealed cast steel	0.008
Ordinary sheet iron	0.004
Annealed iron sheet	0.002
Silicon steel	0.0009
Best silicon steel	0.0006

* Data from *J. Franklin Inst.*, **170,** 1–25 (1910).

Eddy currents are caused by voltages induced in the iron by a changing flux and tend to oppose the change of the flux. Eddy-current losses in iron subjected to a pulsating or alternating flux are reduced by laminating the circuit; this increases the length of path of the secondary currents and hence increases the resistance of their paths. With an alternating flux, the eddy-current loss per unit volume is

$$P_e = ks^2f^2B_{max}^2$$

where s is the thickness of the laminations and k is a constant varying with the resistivity of the iron, the wave shape of the induced voltages, and the flux distribution in the iron.

ELECTRIC AND MAGNETIC RELATIONS

Magnetic flux from electric current (Fig. 15):

$$\phi = \frac{\text{m.m.f.}}{\mathcal{R}} = \frac{1.257NI}{\mathcal{R}}$$

where NI = ampere-turns, $\mathcal{R}$ = reluctance, and ϕ = flux.
Direction of flux lines is reversed if direction of current is reversed (Fig. 15).

Electromotive force (Fig. 16) is induced in a conductor in a magnetic field (a) when there is relative physical motion of conductor with respect to the field, or (*b*) when there is change in magnitude of the field interlinking the conductor.

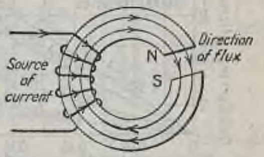
$$E = \frac{Z\phi \times 10^{-8}}{t}; \quad e = \left(\frac{d\phi}{dt}\right) N \times 10^{-8}$$

where Z is the number of conductors cutting the flux ϕ in t seconds; N is the number of turns. The direction of induced e.m.f. is reversed if in (*a*) the direction of motion is reversed or if the field is in the opposite direction, and in (*b*) if the field is increasing or if the field is in the opposite direction and decreasing. The direction of induced e.m.f. is such as to cause a current tending to oppose the change.

Mechanical force (Fig. 17) is exerted on a conductor carrying current when in a magnetic field, tending to move the conductor perpendicularly with respect to both the conductor and the field.

$$F = kBlI$$

$k = 8.83 \times 10^{-8}$ when F is the force in pounds, l the effective length of conductor in inches, B the component of flux density perpendicular to the conductor in lines per square inch, and I the current in amperes. $k = 10.2 \times 10^{-8}$ when F is in kilograms, l in centimeters, and B in gauss.



⊕ Current away from observer

⊙ Current toward observer

FIG. 15.—Magnetic flux produced by electric current.

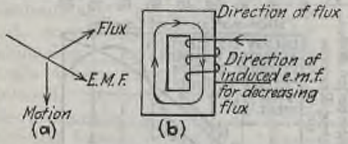


FIG. 16.—Direction of induced e.m.fs. (*a*) For maximum e.m.f., flux, motion and conductor are mutually perpendicular; (*b*) e.m.f. induced in coil by changing of flux linking with it.

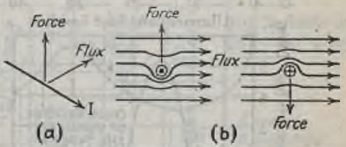


FIG. 17.—Force on conductor carrying current in magnetic field. (*a*) Direction of force, perpendicular to flux and current; (*b*) distortion of flux lines due to current.

Electrical machines consist of interlinked magnetic and electric circuits, usually fairly simple in nature, although in some cases the interactions become quite involved. *Iron* is the usual magnetic material and *copper* the electrical material, hence these together with various forms of *insulation* are the principal materials used in construction.

PHOTO-, PIEZO-, THERMOELECTRICITY

Photoelectric action is the liberation of electrons from matter under the influence of electromagnetic radiation. The *number* of electrons emitted is directly proportional to the quantity of light absorbed by the irradiated surface; their *velocity* varies directly with the frequency of the incident light. Since energy is required to release an electron, and since the quantum of energy in a light wave is a function of the frequency, it follows that only light above a given frequency will cause the emission of electrons from a given substance. This *threshold frequency* is a property of the emitting material and for most metals lies in or near the ultra-violet region of the spectrum; the alkali metals have a threshold frequency in or near the infra-red of the spectrum and are, therefore, good emitters when subjected to ordinary luminous radiation. In addition to the *normal action* of metals as described above, certain metals, including the alkalies, show a *selective action* whereby the emission has a sort of resonance maximum at a given frequency when the

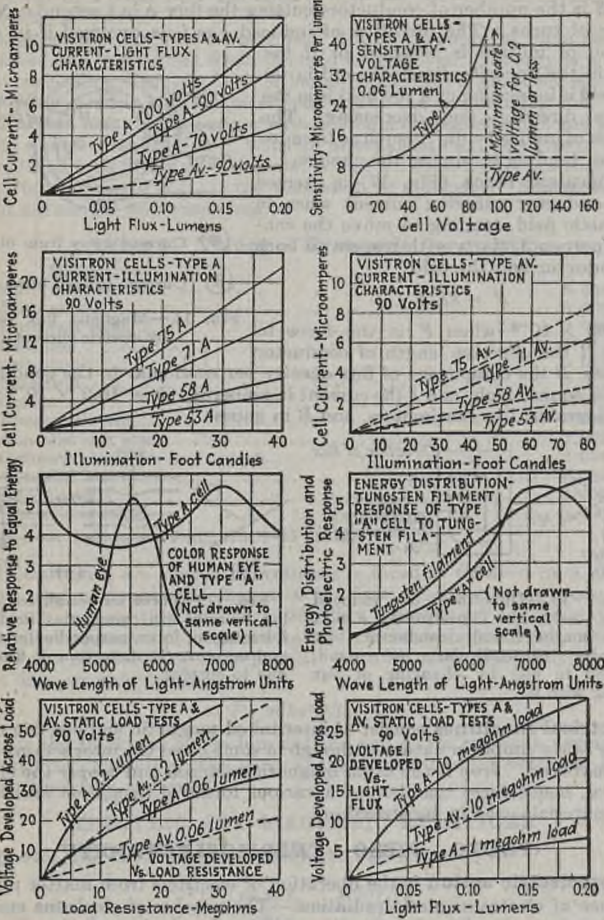


FIG. 18.—Typical characteristics of photoelectric tubes. (*Courtesy of G-M Laboratories.*)

electromagnetic radiation is *plane polarized* so that its electric vector has a component perpendicular to the emitting surface.

The **application** of photoelectric tubes is confined to the utilization of one of the following characteristics of light: (1) quantity; (2) color or wave length; (3) plane of polarization; (4) angle of incidence. Since the cathodes of commercial cells are not optically plane, the last two are not applicable with such cells. The current obtained from photoelectric cells is of the order of 10 microamp. per lumen; to utilize the photoelectric effect some form of *amplification* is necessary. The two methods in general use are: (1) that due to *ionization by collision* in a gas-filled cell (see Conduction of Electricity through Gases) and (2) *thermionic* amplification of direct currents (see Thermionics).

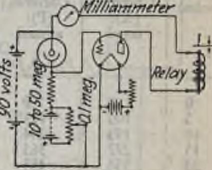


FIG. 19.—Photoelectric tube connected to one-stage amplifier and relay. (*Courtesy of G-M Laboratories.*)

Curves showing the electrical characteristics of typical photoelectric tubes are given in Fig. 18. Two typical thermionic amplifier circuits for use with photoelectric tubes are also shown, the first (Fig. 19) being adapted to static or low-frequency amplification and the second (Fig. 20) to use where the light source is modulated (Campbell and Ritchie, "Photoelectric Cells," Pitman, 1929).

Piezoelectricity. Certain asymmetrical crystals, of which quartz, tourmaline, and rochelle salt are typical, have the property of producing electrical charges on their surface when they are mechanically stressed and, conversely, of experiencing mechanical strain where electrically stressed. Both of these effects are called *piezoelectric;* the former is called the *direct effect* and the latter the *converse effect.* The direct effect has been used for measuring explosion pressures and the converse effect is now in very general use for the stabilization of thermionic oscillators [Cady, *Inst. Rad. Eng.*, **10,** 83 (1922)].

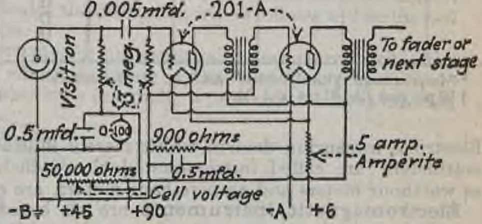


FIG. 20.—Amplifier for modulated light sources. (*Courtesy of G-M Laboratories.*)

Thermoelectricity.[*] If the junctions of two metals composing an electric circuit are held at different temperatures a thermal e.m.f. will exist around the circuit. To a close approximation and over a limited range this e.m.f. is directly proportional to the temperature difference between the two junctions. In practice it is customary either to hold one junction at some fixed temperature or else to compensate automatically for any variations which it undergoes. Thermoelectric instruments are in quite general use for the measurement of temperatures and of high-frequency currents (see Table 5 and p. 2062).

ELECTRICAL MEASUREMENTS[†]

The measurement of the magnitude of any phenomenon is usually based upon some effect produced by it. The effects utilized in electrical measuring devices are (1) *electromagnetic,* (2) *electrostatic,* (3) *thermal,* and (4) *chemical.*

[*] See p. 2062. [†] See p. 2082.

Table 5. Comparison of the More Common Thermocouples*

E (millivolts)	Temperature, °C.				E (millivolts)	Temperature, °C.		
	Chromel (X): copel	Chromel (P): alumel	Iron: constantan	Platinrhodium:† gold-palladium		Platinum: platinrhodium (Heraeus)	Platinum: platinrhodium (Johnston-Matthey)	Copper: constantan
0	0	0	0	0	0	0	0	0
5	105	121	96	131	1	147	146	25
10	195	244	186	237	2	265	260	49
15	277	365	277	335	3	374	364	72
20	353	483	367	429	4	478	461	94
25	425	600	457	513	5	578	553	115
30	495	719	546	607	6	675	641	136
35	...	843	632	694	7	769	725	156
40	...	970	713	779	8	861	806	176
45	...	1104	792	866	9	950	884	195
50	...	....	871	954	10	1037	959	213
55	...	....	950	1044	11	1122	1032	232
60	...	....	...	1136	12	1206	1103	250
					13	1289	1173	268
					14	1372	1242	285
					15	1455	1311	302
					16	1537	1379	320
					17	1620	1447	336
					18	1704	1515	353

* "International Critical Tables," vol. 1, p. 59, McGraw-Hill.

† 10 per cent Rh; 40 per cent Pd.

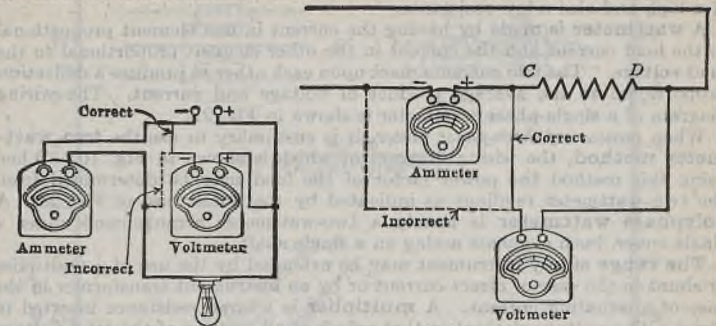
Electrical measuring devices which merely indicate, such as ammeters and voltmeters, are called *instruments;* devices which totalize with time, such as watthour meters and ampere-hour meters, are called *meters.*

Electromagnetic instruments are all based upon the fact that the current-carrying circuit has a force exerted upon it when located in a magnetic field. These instruments may be classified by types as follows: (1) The **moving-magnet** type in which a permanent magnet is caused to move under the influence of an electric current; (2) The **moving-coil** type in which the stationary and movable elements of (1) are interchanged; (3) the **moving-iron** type which differs from (1) in that the magnetization of the moving element is produced by the current in the fixed coil; (4) the **dynamometer** type in which two coils, one fixed and one movable, exert a force upon each other; (5) the **induction** type which operates on the same principle as the induction motor.

Electrostatic instruments are all based upon the general law that a charged body tends to move so as to increase the electrostatic field associated with it.

Thermal instruments are based upon one of two effects: (1) the expansion of a wire with heat, or (2) the generation of an e.m.f. when one junction of two dissimilar metals is raised to a higher temperature than the other. The second type is the more common; the voltage developed is applied to a direct-current instrument (see Thermoelectricity, p. 2653 and p. 2062).

Chemical instruments are based on Faraday's law of electrolysis. Electromagnetic instruments constitute the great bulk of commercial meters. Electrostatic instruments are used mainly on high voltages or as precise laboratory instruments. The chief importance of thermal instruments is for high-frequency measurements and chemical instruments are seldom used except as standards. Accordingly, the discussion which follows will be limited to instruments of the first class.



(*a*) **High Resistance** (*b*) **Low Resistance**

FIG. 21.—Connections for voltmeter and ammeter to obtain minimum effect of power taken by instruments. (*Dawes, "Electrical Engineering," vol. 1, McGraw-Hill,* 1927.)

All **electromagnetic instruments** are essentially ammeters in that the deflection of the needle is an effect produced by the flow of current. When used as a voltmeter, a large impedance (normally a pure resistance) is inserted in the current-carrying circuit. With constant impedance, the current is proportional to the voltage across the terminals and so the scale may be calibrated in volts.

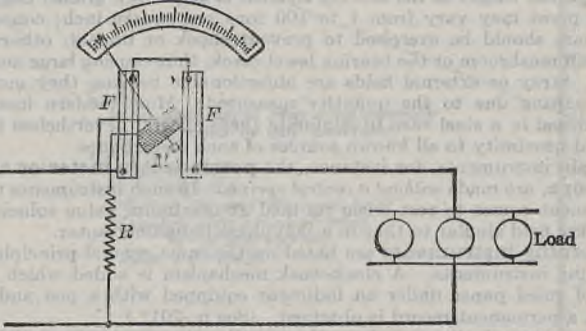


FIG. 22.—Internal and external connections of wattmeter. (*Dawes, "Electrical Engineering," vol.* 1, *McGraw-Hill,* 1927.)

When both an ammeter and a voltmeter are used simultaneously one of two *errors* is inevitable; the ammeter may read the current through the voltmeter as well as through the load, or the voltmeter may read the potential drop across the ammeter as well as across the load. The magnitude of the load resistance is the factor that decides which method shall be used. Figure 21 shows the correct and incorrect way of making a measurement in the case of a high and also a low resistance.

A **wattmeter** is made by having the current in one element proportional to the load current and the current in the other element proportional to the load voltage. The two currents react upon each other to produce a deflection proportional to the average product of voltage and current. The wiring diagram of a single-phase wattmeter is shown in Fig. 22.

When *measuring three-phase power*, it is customary to use the **two-wattmeter method,** the wiring diagram of which is shown in Fig. 10. When using this method the power factor of the load may be determined from the two wattmeter readings as indicated by the nomogram of Fig. 23. A **polyphase wattmeter** is merely a two-wattmeter arrangement under a single cover, both elements acting on a single shaft.

The range of any instrument may be extended by the use of a multiplier or shunt in the case of direct current or by an instrument transformer in the case of alternating current. A **multiplier** is a large resistance inserted in series with a voltage element so that only a small fraction of the total voltage will appear across the instrument terminals; similarly, a **shunt** is a low resistance used to by-pass a large portion of a given current around an ammeter. In self-contained instruments the multiplier or shunt is mounted inside the case. **Instrument transformers,** either potential or current, serve the same purposes in the case of alternating current; they have the additional advantage of insulating the measuring instruments from the main circuit.

The errors encountered in the operation of these instruments are usually due to one of two causes: (1) *improper handling* or (2) *external* or *stray fields*. Although the weight of the moving element is only a few grams, the pressure on the pivot may vary from 1 to 100 tons per square inch; consequently great care should be exercised to prevent shock or impact, otherwise the pivot will mushroom or the bearing jewel crack, thus causing large and erratic errors. Stray or external fields are objectionable because they modify the fields existing due to the quantity measured. Most modern instruments are enclosed in a steel case to eliminate these effects; nevertheless it is well to avoid proximity to all known sources of such disturbance.

Certain instruments, for instance, the **power-factor meter or the synchroscope,** are made *without a control spring*. In such instruments the moving element comes to rest when its field at maximum value coincides with a rotating field similar to that in a polyphase induction motor.

Recording instruments are based on the same general principles as the indicating instruments. A clock-work mechanism is added which draws a sheet of ruled paper under an indicator equipped with a pen and inkwell so that a permanent record is obtained. (See p. 2017.)

Integrating meters, such as the watthour meter, are essentially small motors of very slow speed so adjusted that the speed of rotation is proportional to the power absorbed. The moving element drives a gear train which integrates the number of revolutions made and so the energy utilized.

Besides these instruments and meters, however, there are several **comparison instruments** which enable two quantities of the same or related kinds to be compared. Such for example are: (1) the **potentiometer** for comparing e.m.fs. with the e.m.f. of a standard cell, and (2) the **Wheatstone's-**

bridge for comparing two resistances, or its prototype, the alternating-current bridge, for comparing impedances, inductances, capacitances, etc.

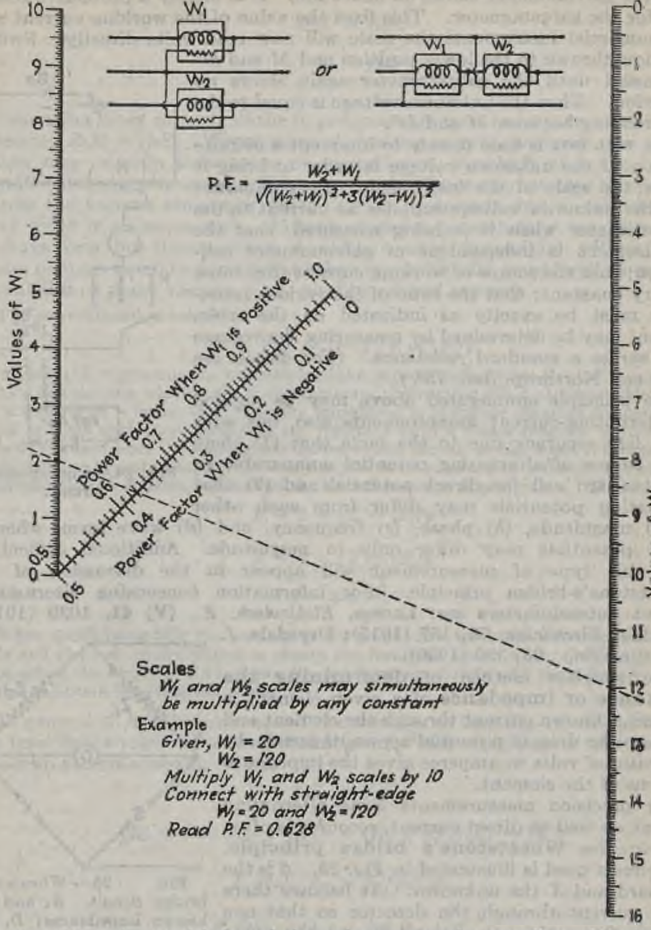


FIG. 23.—Power factor of balanced three-phase system from readings of single-phase wattmeters.

The **potentiometer** or **opposition principle** is that if two equal and opposing e.m.fs. are inserted in a circuit, no current flows in that circuit. The theory of the method may be readily obtained from a study of Fig. 24. The battery E_s is a Weston cadmium cell which has the property of giving a very constant e.m.f., the value of which is certified by the maker or the U. S.

Bureau of Standards. With switch K in the upper position and M and M' set to the scale reading corresponding to the standard cell voltage, r is adjusted until the galvanometer shows no deflection. R is merely a protective resistance for the galvanometer. This fixes the value of the working current and in commercial instruments the scale will now read volts directly. Switch K is now thrown to the lower position and M and M' readjusted until the galvanometer again shows no deflection. Then the unknown voltage is equal to the scale reading between M and M'.

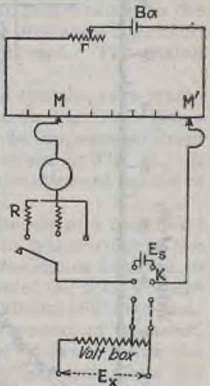


FIG. 24.—Potentiometer circuit.

The **volt box** is used merely to intercept a definite fraction of the unknown voltage in order to bring it within the scale of the instrument. It is apparent that the unknown voltage supplies no current to the potentiometer when it is being measured; that the measurement is independent of galvanometer calibration; that the source of working current (Ba) must be very constant; that the ratio of the various resistances must be exactly as indicated on the scale. Current may be determined by measuring the voltage drop across a standard resistance. (For details see Leeds and Northrup, *Bull.* 755.)

The principle enumerated above may be applied to alternating-current measurements also, but with much less accuracy due to the facts that (1) there is no source of alternating potential comparable to the standard cell for direct potential and (2) that alternating potentials may differ from each other in (*a*) magnitude, (*b*) phase, (*c*) frequency, and (*d*) wave form, whereas direct potentials may differ only in magnitude. Additional difficulties with this type of measurement will appear in the discussion of the Wheatstone's-bridge principle. [For information concerning alternating-current potentiometers see: Larsen, *Elektrotech. Z.*, (V) **41**, 1039 (1910); Drysdale, *Electrician*, **75**, 157 (1915); Drysdale, *J. Inst. Elec. Eng.*, **68**, 339 (1930).

The simplest method of **determining the resistance or impedance** of a given element is to send a known current through the element and measure the drop of potential across its terminals. The ratio of volts to amperes gives the impedance in ohms of the element.

For precision measurements using alternating current as well as direct current, recourse is often had to the **Wheatstone's bridge principle.** The circuit used is illustrated in Fig. 25. S is the standard and X the unknown. At balance there is no current through the detector so that one current flows through Z_1 and S and the other through Z_2 and X. Also, at balance the two terminals of the detector are at the same potential and, therefore,

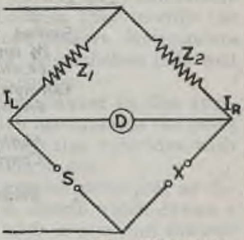


FIG. 25.—Wheatstone bridge circuit. Z_1 and Z_2, known impedances; D, detecting device; X, the unknown; S, a standard impedance.

$$I_L Z_1 = I_R Z_2$$

and

$$I_R X = I_L S$$

Multiplying the two equations term by term,

$$I_L I_R Z_1 X = I_L I_R Z_2 S$$

or

$$Z_1 X = Z_2 S \tag{1}$$

or

$$X = \frac{Z_2 S}{Z_1}$$

When using *direct current*, all the impedances are pure resistances, and Eq. (1) becomes $R_1X = R_2S$. When using *alternating current*, however, the impedances may contain inductance or capacitance, in which case the balance conditions become more complicated. In the direct-current case the voltages across the various arms may differ only in magnitude, but in this case they may differ in phase as well. If harmonics are present they may also differ in wave form, but this condition should be avoided if possible. When impedances contain reactive components they are ordinarily considered as complex quantities or plane vectors, *i.e.*, two-dimensional vectors.

The conventional notation is

$$Z = R \pm jX = |Z|\underline{/\theta}, \text{ or } |Z|\overline{/\theta}$$

In all cases $|Z|$ represents the magnitude of the impedance in ohms; $R \pm jX$ is composed of two components which have their maximum obstructing effect at times which differ by a quarter of a cycle; their effective impedance is therefore obtained by combining them at right angles and obtaining the resultant as in a vector diagram of forces; the plus sign is used when the reactance is inductive and the minus sign when it is capacitive. This notation is quite simple where only addition or subtraction is involved, but where multiplication or division is involved the second notation is preferable. Thus

$$Z = R + jX = |Z|\underline{/\theta}$$

or

$$Z = R - jX = |Z|\overline{/\theta}$$

where

$$\theta = \tan^{-1}\frac{X}{R}.$$

When multiplying take the product of the amplitudes to obtain the resultant amplitude and the sum of the angles to obtain the resultant angle; when dividing take the quotient of the amplitudes to obtain the resultant amplitude and the difference of the angles to obtain the resultant angle.

In general all four impedances will be complex so that Eq. (1) can be broken up into two equations, one containing only real components and one the quadrature components. Thus

$$Z_1 X = Z_2 S$$

or

$$(R_1 + jX_1)(R_x + jX_x) = (R_2 + jX_2)(R_s + jX_s)$$

or

$$Z_1\underline{/\theta_1} X\underline{/\theta_x} = Z_2\underline{/\theta_2} S\underline{/\theta_s}$$

or

$$Z_1 X\underline{/\theta_1 + \theta_x} = Z_2 S\underline{/\theta_2 + \theta_s}$$

Thus, in the general case, two conditions must be fulfilled for balance.

1. Amplitude balance

$$Z_1 X = Z_2 S$$

2. Angular balance

$$\underline{/\theta_1 + \theta_x} = \underline{/\theta_2 + \theta_s}$$

In the actual operation of the bridge, some systematized procedure must be adopted to obtain first one balance, then the other, then return to the first and check, and so on, since phase and amplitude balance are not independent.

The chief sources of **error in the alternating-current bridge** are: (1) imperfect standards; (2) stray fields or currents.

The effect of an **imperfect standard** may be readily obtained from the above equation. Thus if Z_1 and Z_2 are supposed to be pure resistances and of equal magnitude, θ_1 and θ_2 have no effect on the balance. If the standards are unequal, or if Z_2 and X are interchanged, the imperfections of the standard may have a large effect.

In the simple explanation of the theory of operation, it was assumed that all currents were confined to the indicated paths and that only the source produced any effective voltage. **Stray currents** or **stray fields** will vitiate these assumptions. The remedy, particularly at high frequencies, is suitably to shield the network. Care should be exercised as poor shielding is worse than none at all. For a good description of bridge shielding see Shackelton and Ferguson, *Am. Inst. Elec. Eng.*, **46**, 519 (1927).

Although the bridge is designed basically for the comparison of impedances, it has been applied successfully to a number of uses in which the quantity to be measured is a function of the impedance or *vice versa*. Such varied applications include the measurement of frequency and time, humidity, percentage of CO_2 in flue gas, temperature, and dielectric loss.

A bridge has its **maximum sensitivity**: where (1) the impedance of the total bridge network is equal to the impedance of the source; (2) the impedance of the detector is equal to the impedance measured external to the detector; (3) the branches all have the same impedance. To summarize, use a high-impedance detector with high-impedance arms and source and *vice versa*.

The *source* of alternating current is usually a thermionic oscillator. For direct-current work, a storage battery is used.

The *means of detection* is usually: (1) a vibration galvanometer for frequencies between 50 and 300; (2) a telephone for frequencies from 300 to 3000; (3) a thermionic detector or a heterodyne method for higher frequencies. For direct-current work, either a moving coil or a moving magnet galvanometer may be used.

CONDUCTION OF ELECTRICITY THROUGH GASES

A gas in its normal state is a very **poor conductor** of electricity due to absence of ions, or carriers of electricity. Any ultimate carrier of electric charge, be it molecule, atom, or electron, is called an **ion.** These carriers are present in small quantities under all conditions and a plentiful supply may be engendered in a variety of ways. The *source* of these carriers may be the gas itself or its container or the electrodes of the discharge; usually all of these sources contribute to the supply.

There are three major methods for releasing ions, usually electrons, **from solid bodies.** These are: *heat* (thermionics), *light* (photoelectrics), and *positive-ion bombardment*. The mechanism of the first two is discussed elsewhere; that of the third is somewhat obscure but seems to be similar in effect to a mechanical impact. The major cause of the production of **ions in gases** is *electron impact* with an atom or molecule.

If voltage be applied between two electrodes in a gas at a pressure, of 10 to 1000 mm. Hg, the resultant current-voltage curve is given by Fig. 26. At very low potentials the characteristic is practically linear (Ohm's law).

The carriers in this case are the few ions which seem to be omnipresent. Between B and C the current is saturated due to the fact that all of these available carriers are being drawn to the electrodes and no new carriers are being generated. At C these carriers are being so accelerated by the electric field that the electrons possess sufficient kinetic energy to release new electrons by impact with the gas molecules. The current consequently begins to increase. This increase is accumulative until at D the positive ions, which resulted from the release of free electrons from the gas molecules, acquire sufficient kinetic energy to ionize and the current increases very rapidly.

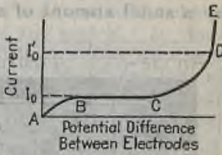


FIG. 26.—Gaseous-discharge characteristic.

The phenomena which occur in this region depend upon many factors, chief of which are the shape, size, and material of the electrodes and the voltage regulation of the source of energy. With sources having poor voltage regulation, the voltage will drop with increase of current as shown in Fig. 27. Hence a higher voltage is required to start the discharge than to maintain it. This type of response is made use of in gaseous glow tubes as voltage regulators. With sources having very poor regulation, the discharge may be instantaneous in the form of a spark. If the voltage is maintained so that the electrodes become heated due to ionic bombardment, electrons being emitted from the heated cathode, the discharge is self-sustaining and is called an *arc*.

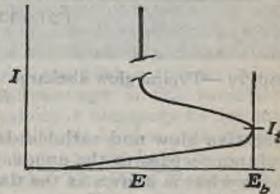


FIG. 27.—Gaseous-discharge characteristic. (*Moyer and Wostrel*, "*Radio Handbook*," *McGraw-Hill*, 1931.)

It is found that for a given distance d, between electrodes, there is a critical pressure P_0, at which the spark will most readily pass. This potential is called the **minimum sparking potential** for the gas and electrode material in question. It varies from 60 to 400 volts, the lower values applying when electrodes of the electropositive metals are used. For any given distance between plates it is always possible to find some pressure at which a spark passes at this minimum potential. At higher pressures the mean free path of the gas is so short that higher potentials are required to accelerate the electrons sufficiently to cause ionization; at lower pressures the probabilities of an electron striking a neutral gas molecule are reduced. It is found that P_0d is a constant for any gas and that above P_0, if the electric field is uniform, the sparking potential is proportional to Pd. In other words, the sparking potential is proportional to the mass of gas between the electrodes. This is called **Paschen's law** (Fig. 28). It should be noted that under the proper conditions a short length of gas makes a better insulator than a long one.

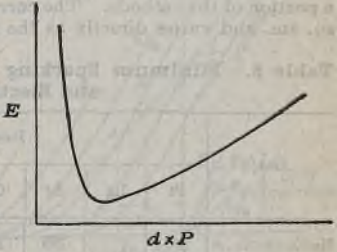


FIG. 28.—Minimum sparking potential related to gas pressure P and distance d between electrodes. (*Moyer and Wostrel*, "*Radio Handbook*," *McGraw-Hill*, 1931.)

As the *pressure is reduced*, the discharge gradually loses its spark characteristic and spreads out toward the walls of the tube and takes on the appearance of a **glow discharge.** At about 1 mm. Hg pressure we have the familiar Geissler tube effects (Fig. 29). Here the discharge has actually broken up and, going from anode to cathode, has a positive column (glow), a Faraday dark space, a negative glow, a cathode dark space, and a cathode glow. In some cases the positive column is *striated*, especially if the gas is contaminated by a small amount of another gas. With further reduction in pressure the negative glow and cathode dark space increase in length while the positive column recedes to the anode. Finally the whole tube becomes dark and we have what is known as the **dark discharge.**

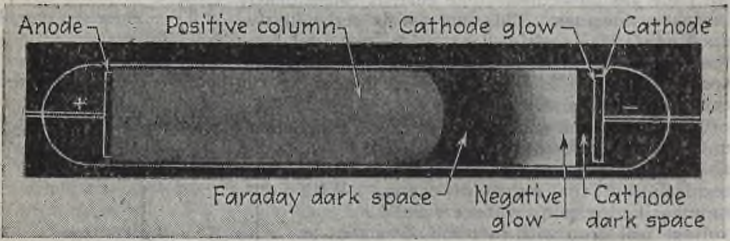


FIG. 29.—Typical glow discharge in a gas at about 1 mm. pressure. [*From Rev. Modern Phys.*, **3**, 192 (1931).]

At about 1 mm. Hg pressure the discharge is characterized by its **cathode fall of potential,** which is that sharp drop of potential which occurs within a few millimeters of the cathode. When the cathode fall is *normal* its value is equal to the minimum sparking potential, and the cathode glow covers only a portion of the cathode. The current density is then about 0.4 milliamp. per sq. cm. and varies directly as the pressure. When the current increases so that the glow extends over the entire cathode, the current density increases and the cathode fall of potential increases to *abnormal* values, sometimes reaching thousands of volts.

Table 6. Minimum Sparking Potentials for Various Atmospheres and Electrode Materials*

Gas	Potentials in volts for electrodes of									
	Pt	Hg	Ag	Cu	Fe	Zn	Al	Mg	Na	K
H_2	300	...	295	280	230	213	190	168	185	172
N_2	232	226	...	...	...	...	...	207	178	170
O_2	369	...	...	...	...	...	...	310		
He	160	142	162	177	161	143	141	125	80	69
Air	340									
A	167	...	...	...	...	...	100			

* Reprinted by permission from Cady and Dates, "Illuminating Engineering," Wiley.

At *very low pressures*, the discharge is not self-sustaining—the discharge does not itself produce ions to replace those drawn to the electrodes—and, therefore, some external source of ionization, as heat, light, or X-rays, is required to maintain the discharge. With modern X-ray tubes, hundreds of thousands or even millions of volts may be required to produce a current of a few milliamperes in the absence of such ionizing agents.

THERMIONIC TUBES

The applications of **thermionic tubes** which are of most use to the chemical engineer are probably in the field of *measurement* and *control*. In these fields they are useful to him in various ways: as **oscillators** they provide a source of current over a practically unlimited range of readily adjustable frequency; as **amplifiers** they increase the sensitivity, and hence precision, of measurement and control; moreover, they enable one to work with very small voltages and currents, a distinct advantage in measurements such as that of the conductance of electrolytes; as **detectors** they provide a very versatile electrical indicator and in the heterodyne arrangement enable one to apply audible methods to the range of supersonic frequencies.

The outstanding **advantages** of thermionic tubes as measurement or control instruments are: (1) the extremely *small influence* they exert upon the circuit which is being tested; (2) the *wide range* of frequency and amplitude over which observations may be made; and (3) the *high sensitivity* which they make possible. Their outstanding **disadvantage** is the *variability* of tube and associated apparatus with time, making it necessary to use substitution methods or frequent calibrations where high accuracy is desired.

The **fundamental phenomena** of thermionic tubes have been well described by Irving Langmuir in an article entitled The Pure Electron Discharge, *Gen. Elec. Rev.*, **18**, p. 327, 1915. A good textbook on thermionic tubes is Terman, "Radio Engineering," McGraw-Hill, New York, 1937. For our purposes it is sufficient to note that although a vacuum is ordinarily an almost perfect insulator, it may be made a fair conductor by releasing free electrons within the enclosure. In the thermionic tube this is accomplished by heating the filament or cathode; the mechanism of this emission is closely analogous to the evaporation of water upon the application of heat. Under working conditions not all the "evaporated" electrons are drawn to the plate or anode, which is held at a relatively high positive potential with respect to the cathode by means of an external B battery. In consequence a cloud of free electrons called the *space charge* gathers in the region between anode and cathode and exerts a repulsive or resistive effect on other electrons trying to reach the anode. In the triode or three-electrode tube a metallic grid or mesh, usually held at a small negative potential with respect to the cathode by means of a C battery, is inserted into this space-charge region and, by its tendency to neutralize or reinforce the space-charge effect as its potential is varied, gives a powerful and sensitive means of controlling the current between

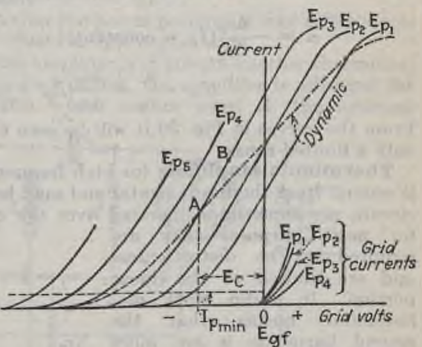


FIG. 30.—Thermionic-tube characteristics.

cathode and anode. It should be noted that the grid acts as a valve which may be turned or controlled by a very small amount of power, whereas the amount of power which it in turn controls, derived from the B battery, may be very great.

The **characteristics** of a thermionic tube are commonly presented in graphic form similar to the curves of Figs. 30, and 31. The three constants of greatest importance are: the **amplification constant** μ, which indicates the relative effect of small changes of grid and plate voltages on the flow of plate current; the **mutual conductance** G_M, which indicates the change in plate current produced by a small change in grid voltage (when used to define the same change with an impedance in the external plate circuit, it is commonly designated by G'_M), and the **internal** or **plate resistance** r_p, which is the ratio of a small change in plate voltage to the resultant change in plate current. Mathematically these become:

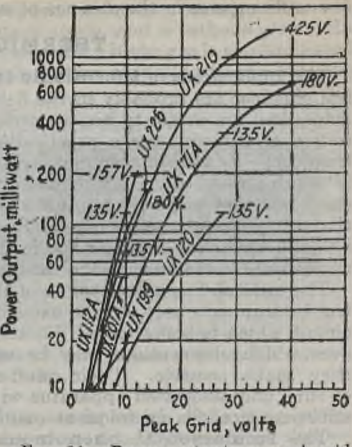


FIG. 31.—Power output *vs.* peak-grid voltage. (*Moyer and Wostrel, "Radio Handbook," McGraw-Hill*, 1931.)

$$\mu = -\frac{\Delta e_p}{\Delta e_g}(I_p = \text{constant});\ r_p = \frac{\Delta e_p}{\Delta i_p}(E_g = \text{constant});$$

$$G_m = \frac{\mu}{r_p} = \frac{\Delta i_p}{\Delta e_g}(E_p = \text{constant})$$

From the curves in Fig. 30 it will be seen that these are true constants over only a limited range.

Thermionic amplifiers for high frequencies, where the second harmonic is remote from the fundamental and may be eliminated by means of a tuned circuit, are sometimes operated over the complete tube characteristic, but for most purposes they are required to be distortionless and are limited to the linear portion. In radio work *distortionless* means that the second harmonic is not more than 5 per cent of the fundamental. In such applications the circuits would be adjusted to operate about some such point as *A* (Fig. 30). With a sinusoidal input voltage, the grid would just reach zero during the positive half of the cycle for the same value of input which would reduce the plate current to the minimum I_p on the negative half of the cycle. These represent the two limits for *Class A*, or *linear amplifiers*. If the current goes below the minimum I_p, the output is no longer a true reproduction of the input

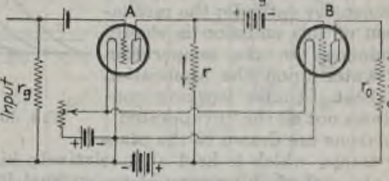


FIG. 32(*a*).—Resistance-coupled amplifier. (*Van der Bijl, "Thermionic Vacuum Tubes," McGraw-Hill*, 1920.)

due to the curvature of the characteristic, and, if the grid acquires a positive potential, it will draw a grid current, as indicated by the curves emanating from $E_o = 0$ (Fig. 30). This current flowing through a high impedance will distort the input voltage. Actually the *dynamic* or operating characteristic will differ from the *static* characteristic due to the impedance drop in the load, as indicated by the dotted curve through A, so that the actual potentials may differ somewhat from those indicated by the above simple explanation.

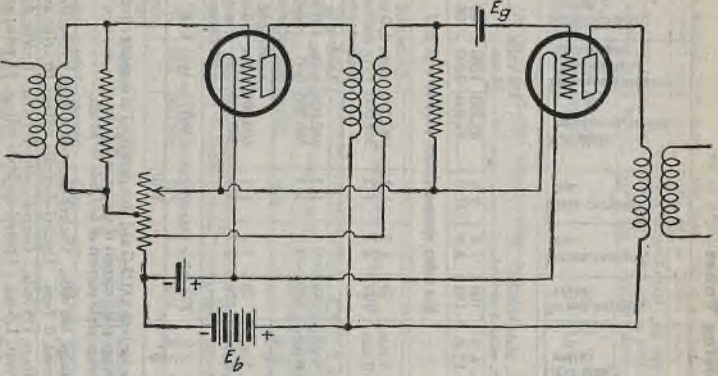


FIG. 32(*b*).—Transformer-coupled amplifier. (*Van der Bijl, "Thermionic Vacuum Tubes," McGraw-Hill*, 1920.)

If the **output** of such a thermionic amplifier is to govern another thermionic tube or any other potentially operated device, the amplifier is adjusted for **maximum voltage amplification** which occurs when Z, the external impedance, is as large as is feasible. Under this condition the *effective amplification* is $\mu Z/(r_p + Z)$. Note that this is always less than the nominal amplification, though it may be made greater by inserting a transformer. By using a transformer very *high amplifications* may be obtained over a limited range of frequency, but where a wide band is to be covered, as in the audio

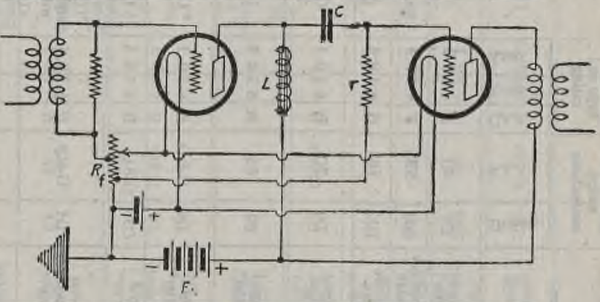


FIG. 32(*c*).—Choke or impedance-coupled amplifier. (*Van der Bijl, "Thermionic Vacuum Tubes," McGraw-Hill*, 1920.)

Table 7. Six-volt Receiving Tubes

Type	Name	Dimensions socket connections		Cathode type and rating			Use[t]	Plate supply, volts	Grid bias,[a] volts	Screen supply, volts	Screen current, ma.	Plate current, ma.	A.c. plate resistance, ohms	Transconductance (grid-plate), micromhos	Amplification factor	Load for stated power output, ohms	Power output, watts
		Dimen.	S.c.	C.t.	Volts	Amp.											
1-V	Half-wave rectifier	D5	4G	H	6.3	0.3		Maximum a.c. plate voltage.................... 350 volts, r.m.s. Maximum d.c. output current.................... 50 ma.									
6A4/LA	Power amplifier pentode	D12	5B	F	6.3	0.3	Class A amplifier	100 180	− 6.5 −12.0	100 180	1.6 3.9	9.0 22.0	83,250 45,500	1200 2200	100 100	11,000 8,000	0.31 1.40
6A6	Twin triode amplifier	D12	7B	H	6.3	0.8	Amplifier	For other characteristics, refer to type 6N7									
6A7	Pentagrid converter[b]	D9	7C	H	6.3	0.3	Converter	100 250	−1.5 min. −3.0 min.	50 100	1.3 2.7	1.1 3.5	600,000 360,000	Anode-grid (#2): 250[c] max. volts, 4.0-ma. oscillator-grid (#1) resistor.[d] Conversion conductance, 520 micromhos			
6A8	Pentagrid converter[b]	C1	8A	H	6.3	0.3	Converter	100 250	−1.5 min. −3.0 min.	50 100	1.5 3.2	1.2 3.3	600,000 360,000	Anode-grid (#2): 250 max. volts, 4.0-ma. oscillator-grid (#1) resistor.[d] Conversion conductance, 500 micromhos			
6A8-G	Pentagrid converter[b]	D8	G-8A[e]	H	6.3	0.3	Converter	100 250	−1.5 min. −3.0 min.	50 100	1.3 2.7	1.1 3.5	600,000 360,000	Anode-grid (#2): 250[c] max. volts, 4.0-ma. oscillator-grid (#1) resistor.[d] Conversion conductance, 550 micromhos			
6A8-GT	Pentagrid converter[b]	C3	G-8A[e]	H	6.3	0.3	Converter	For other characteristics, refer to type 6A8									
6AC5-G	High-mu power amplifier triode	D3	G-6Q[e]	H	6.3	0.4	Class B amplifier	250	0	...	....	5.0[f]	....	....	......	10,000	8.0[g]
							Dynamic-coupled amplifier with type 76 driver	250	Bias for both 6AC5-G and 76 is developed in coupling circuit Average plate current of driver = 5.5 ma. Average plate current of 6AC5-G = 32 ma.							7,000	3.7
6AF6-G	Electron-ray tube twin indicator type	B2	G-7AG	H	6.3	0.15	Visual indicator	Target voltage, 100 volts. Control-electrode voltage, 0 volts; shadow angle, 100°; target current, 0.9-ma. Control-electrode voltage, 60 volts; angle, 0° Target voltage, 135 volts. Control-electrode voltage, 0 volts; shadow angle, 100°; target current, 1.5-ma. Control-electrode voltage, 81 volts; angle, 0°									

6B6-G	Duplex-diode high-mu triode	D8	G-7V^{e}	H	6.3	0.3	Triode unit as class A amplifier	250	− 2.0	...	.....	1.0	91,000	1100	100		
6B7	Duplex-diode pentode	D9	7D	H	6.3	0.3	Pentode unit as amplifier	For other characteristics, refer to type 6B8-G									
6B8	Duplex-diode pentode	C1	8E	H	6.3	0.3	Pentode unit as r.f. amplifier	250	− 3.0	125	2.3	10.0	600,000	1325	800		
							Pentode unit as a.f. amplifier	90^{h} 300^{h}	Self-bias, 3500 ohms. Screen resistor = 1.1 meg. Self-bias, 1600 ohms. Screen resistor = 1.2 meg. grid resistor^{i} 0.5 megohm gain per stage = 55 gain per stage = 79								
6B8-G	Duplex-diode pentode	D8	G-8E^{e}	H	6.3	0.3	Pentode unit as r.f. amplifier	100 250	− 3.0 − 3.0	100 125	1.7 2.3	5.8 9.0	300,000 650,000	950 1125	285 730		
							Pentode unit as a.f. amplifier	90^{h} 300^{h}	Self-bias, 3500 ohms. Screen resistor = 1.1 meg. Self-bias, 1600 ohms. Screen resistor = 1.2 meg. grid resistor^{i} 0.5 megohm gain per stage = 55 gain per stage = 79								
6C5	Detector^{j} amplifier triode	B3	6Q	H	6.3	0.3	Class A amplifier	250	− 8.0	...	.....	8.0	10,000	2000	20		
								90^{r} 300^{r}	Self-bias, 6400 ohms Self-bias, 5300 ohms grid resistor,^{i} 0.25 megohm gain per stage = 11 gain per stage = 13								
							Bias detector	250	−17.0 approx. Plate current to be adjusted to 0.2 ma. with no signal								
6C5-G	Detector^{j} amplifier triode	D3	G-8Q^{k}	H	6.3	0.3	Amplifier detector	For other characteristics, refer to type 6C5									
6C6	Triple-grid detector amplifier	D13	6F	H	6.3	0.3	Amplifier detector	For other characteristics, refer to type 6J7									
6C8-G	Twin triode amplifier	D8	G-8G	H	6.3	0.3	Each unit as amplifier	250	− 4.5	...	.....	3.2	22,500	1600	36		
6D6	Triple-grid super-control amplifier	D13	6F	H	6.3	0.3	Amplifier mixer	For other characteristics, refer to type 6U7-G									
6D8-G	Pentagrid converter^{b}	D8	G-8A^{e}	H	6.3	0.15	Converter	135 250	− 3.0 min.	67.5 100	....	....	400,000 320,000	Anode-grid (#2): 250^{l} max. volts. Oscillator-grid (#1) resistor.^{d} Conversion conductance, 500 micromhos			
6E5	Electron-ray tube	D5	6R	H	6.3	0.3	Visual indicator	Plate and target supply = 100 volts. Triode plate resistor = 0.5 megohm. Target current = 1.0 ma. Grid bias, −3.3 volts; shadow angle, 0°. Bias. 0 volts; angle, 90°; plate current, 0.19 ma.									
								Plate and target supply = 250 volts. Triode plate resistor = 1.0 megohm. Target current = 4.0 ma. Grid bias, −8.0 volts; shadow angle, 0°. Bias. 0 volts; angle, 90°; plate current, 0.24 ma.									

Table 7. Six-volt Receiving Tubes—(*Continued*)

Type	Name	Dimensions socket connections		Cathode type and rating			Use[1]	Plate supply, volts	Grid bias,[a] volts	Screen supply, volts	Screen current, ma.	Plate current, ma.	A.c. plate resistance, ohms	Transconductance (grid-plate), micromhos	Amplification factor	Load for stated power output, ohms	Power output, watts
		Dimen.	S.c.	C.t.	Volts	Amp.											
6F5	High-mu triode	C1	5M	H	6.3	0.3	Amplifier	For other characteristics, refer to type 6SF5									
6F5-G	High-mu triode	D8	G-5M[k]	H	6.3	0.3	Amplifier	For other characteristics, refer to type 6SF5									
6F6	Power amplifier pentode	C2	7S	H	6.3	0.7	Pentode class A amplifier	250	−16.5	250	6.5	34.0	80,000	2500	200	7,000	3.0
								315	−22.0	315	8.0	42.0	75,000	2650	200	7,000	5.0
							Triode[m] class A amplifier	250	−20.0	...	.....	31.0	2,600	2600	6.8	4,000	0.85
							Pentode push-pull class A amplifier	315	Self-bias	315	16.0[f]	84.0[f]	Self-bias resistor, 220 ohms[f]			10,000	13.0[g]
								315	−22.0	315	16.0[f]	84.0[f]	....	....	.....	10,000	13.0[g]
							Pentode push-pull class AB$_2$ amplifier	375	Self-bias	250	8.0[f]	54.0[f]	Self-bias resistor, 340 ohms[f]			10,000	19.0[g]
								375	−26.0	250	5.0[f]	34.0[f]	....	....	.....	10,000	19.0[g]
							Triode push-pull[m] class AB$_2$ amplifier	350	Self-bias	...	.....	50.0[f]	Self-bias resistor, 730 ohms[f]			10,000	14.0[g]
								350	−38.0			45.0[f]	....	....	.....	6,000	18.0[g]
6F6-G	Power amplifier pentode	D10	G-7S[e]	H	6.3	0.7	Amplifier	For other characteristics, refer to Type 6F6									
							Triode unit as class A amplifier	100	− 3.0	...	.....	3.5	16,000	500	8		
6F7	Triode-pentode	D9	7E	H	6.3	0.3	Pentode unit as class A amplifier	100	− 3.0 min.	100	1.6	6.3	290,000	1050	300		
								250		100	1.5	6.5	850,000	1100	900		
							Pentode unit as mixer	250	−10.0	100	0.6	2.8	Oscillator peak volts = 7.0 Conversion conductance = 300 micromhos				
6F8-G	Twin triode amplifier	D8	G-8G	H	6.3	0.6	Each unit as amplifier	250	− 8.0	...	.....	9.0	7,700	2600	20		
6G6-G	Power amplifier pentode	D3	G-7S[e]	H	6.3	0.15	Class A amplifier	135	− 6.0	135	2.0	11.5	170,000	2100	360	12,000	0.6
								180	− 9.0	180	2.5	15.0	175,000	2300	400	10,000	1.1

6H6	Twin diode	A1	7Q	H	6.3	0.3	Detector rectifier	Maximum a.c. voltage per plate...................... 100 volts, r.m.s. Maximum d.c. output current...................... 4 ma.									
6H6-G	Twin diode	D3	G-7Q[k]	H	6.3	0.3	Detector rectifier	For other ratings, refer to type 6H6									
6J5	Detector amplifier triode	B3	6Q	H	6.3	0.3	Class A amplifier	250	− 8.0	...	.....	9.0	7700	2600	20		
6J5-G	Detector amplifier triode	D3	G-6Q[e]	H	6.3	0.3	Amplifier	For other characteristics, refer to type 6J5									
6J7	Triple-grid detector amplifier	C1	7R	H	6.3	0.3	Class A r.f. amplifier	100 250	− 3.0 − 3.0	100 100	0.5 0.5	2.0 2.0	1,000,000 1.5+[n]	1185 1225	1185 1500+		
							Class A a.f. amplifier	90[h] 300[h]	Self-bias, 2600 ohms. Self-bias, 1200 ohms.	Screen resistor = 1.2 meg. Screen resistor = 1.2 meg.	grid resistor,[f] 0.5 megohm gain per stage = 85 gain per stage = 140						
							Bias detector	250[h]	− 4.3	100	Cathode current 0.43 ma.		....	Plate resistor, 500,000 ohms Grid resistor,[f] 250,000 ohms			
6J7-G	Triple-grid detector amplifier	D8	G-7R[k]	H	6.3	0.3	Amplifier detector	For other characteristics, refer to type 6J7[i]									
6K5-G	High-mu triode	D8	G-5U	H	6.3	0.3	Class A amplifier	100 250	− 1.5 − 3.0	... ...	....	0.35 1.1	78,000 50,000	900 1400	70 70		
6K6-G	Power amplifier pentode	D3	G-7S[e]	H	6.3	0.4	Class A amplifier	100 250	− 7.0 −18.0	100 250	1.6 5.5	9.0 32.0	103,500 68,000	1450 2200	150 150	12,000 7,600	0.33 3.40
6K7	Triple-grid super-control amplifier	C1	7R	H	6.3	0.3	Class A amplifier	90 250	− 3.0 min.	90 125	1.3 2.6	5.4 10.5	315,000 600,000	1275 1650	400 990		
							Mixer in superheterodyne	250	−10.0	100	.....	.....	Oscillator peak volts = 7.0				
6K7-G	Triple-grid super-control amplifier	D8	G-7R[e]	H	6.3	0.3	Amplifier mixer	For other characteristics, refer to type 6K7									
6K7-GT	Triple-grid super-control amplifier	C3	G-7R[e]	H	6.3	0.3	Class A amplifier	100 250	− 3.0 min.	100 100	1.6 1.7	6.5 7.0	250,000 800,000	1325 1450	350 1160		
							Triode unit as oscillator	100	Triode-grid resistor[d]			3.8	Triode-grid and hexode-grid current, 0.15 ma.				
6K8	Triode-hexode converter	C1	8K	H	6.3	0.3	Hexode unit as mixer	100 250	− 3.0 min.	100 100	6.2 6.0	2.3 2.5	400,000 600,000	Conversion conductance, 325 micromhos Conversion conductance, 350 micromhos			

Table 7. Six-volt Receiving Tubes—*(Continued)*

Type	Name	Dimensions socket connections		Cathode type and rating			Use[1]	Plate supply, volts	Grid bias,[a] volts	Screen supply, volts	Screen current, ma.	Plate current, ma.	A.c. plate resistance, ohms	Transconductance (grid-plate), micromhos	Amplification factor	Load for stated power output, ohms	Power output, watts
		Dimen.	S.c.	C.t.	Volts	Amp.											
6L5-G	Detector amplifier triode	D3	G-6Q[e]	H	6.3	0.15	Class A amplifier	135 250	− 5.0 − 9.0	... ...	....	3.5 8.0	11,300 9,000	1500 1900	17 17		
6L6	Beam power amplifier	D7	7AC	H	6.3	0.9	Single-tube class A₁ amplifier	250	−14.0	250	5.0	72.0	....	....	.....	2,500	6.5
								250	Self-bias	250	5.4	75.0	Self-bias resistor, 170 ohms			2,500	6.5
							Push-pull class A₁ amplifier	250	−16.0	250	10.0[f]	120.0[f]	....	....	.....	5,000	14.5[g]
								250	Self-bias	250	10.0[f]	120.0[f]	Self-bias resistor, 125 ohms[f]			5,000	13.8[g]
							Push-pull class AB₁ amplifier	400	−25.0	300	6.0[f]	102.0[f]	....	....	.....	6,600	34.0[g]
								400	Self-bias	300	7.0[f]	112.0[f]	Self-bias resistor, 200 ohms[f]			6,600	32.0[g]
							Push-pull class AB₂ amplifier	400 400	−20.0 −25.0	250 300	4.0[f] 6.0[f]	88.0[f] 102.0[f]	....	....	....	6,000 3,800	40.0[g] 60.0[g]
6L6-G	Beam power amplifier	E2	G-7AC[e]	H	6.3	0.9	Amplifier	For other characteristics, refer to type 6L6									
6L7	Pentagrid mixer[o] amplifier	C1	7T	H	6.3	0.3	Mixer in superheterodyne	250	− 3.0	100	7.1	2.4	Oscillator-grid (#3) bias, −10 volts Grid 3 peak swing, 12 volts minimum Conversion conductance, 350 micromhos				
							Class A amplifier	250	− 3.0 min.[p]	100	6.5	5.3	800,000	1100	880		
6L7-G	Pentagrid mixer[o] amplifier	D8	G-7T[e]	H	6.3	0.3	Mixer amplifier	For other characteristics, refer to type 6L7									
6N5	Electron-ray tube	D5	6R	H	6.3	0.15	Visual indicator	Plate and target supply = 135 volts. Triode plate resistor = 0.25 meghom. Target current = 2.0 ma. Grid bias, −12.0 volts; shadow angle, 0°. Bias, 0 volts; angle 90°; plate current 0.5 ma.									

6N7	Twin triode amplifier	C2	8B	H	6.3	0.8	Class A amplifier (as driver)	250 294	− 5.0 − 6.0	... ...	....	6.0 7.0	11,300 11,000	3100 3200	35 35	20,000 or more	ex- ceeds 0.4
							Class B amplifier	250 300	0 0	... ...	....	Power output is for one tube at stated plate-to-plate load				8,000 10,000	8.0 10.0
6N7-G	Twin triode amplifier	D10	G-8B[e]	H	6.3	0.8	Amplifier	For other characteristics, refer to type 6N7									
6Q7	Duplex-diode high-mu triode	C1	7V	H	6.3	0.3	Triode unit as class A amplifier	100 250	− 1.5 − 3.0	... ...	....	0.35 1.1	87,500 58,000	800 1200	70 70		
								90[h] 300[h]	Self-bias, 7600 ohms Self-bias, 3000 ohms	grid resistor,[i] 0.5 megohm					gain per stage = 32 gain per stage = 45		
6Q7-G	Duplex-diode high-mu triode	D8	G-7V[e]	H	6.3	0.3	Triode unit as amplifier	For other characteristics, refer to type 6Q7									
6Q7-GT	Duplex-diode high-mu triode	C3	G-7V[e]	H	6.3	0.3	Triode unit as class A amplifier	100 250	0 − 3.0	... ...	....	2.3 1.1	43,000 58,000	1400 1200	60 70		
								250	− 9.0	...	.....	9.5	8,500	1900	16		
6R7	Duplex-diode triode	C1	7V	H	6.3	0.3	Triode unit as class A amplifier	90[h] 300[h]	Self-bias, 4400 ohms Self-bias, 3800 ohms	grid resistor,[i] 0.25 megohm					gain per stage = 10 gain per stage = 10		
6R7-G	Duplex-diode triode	D8	G-7V[e]	H	6.3	0.3	Triode unit as amplifier	For other characteristics, refer to type 6R7									
6S7	Triple-grid super-control amplifier	C1	7R	H	6.3	0.15	Class A amplifier	135 250	− 3.0 min.	67.5 100	0.9 2.0	3.7 8.5	 1,000,000	1250 1750	1750		
6S7-G	Triple-grid super-control amplifier	D8	G-7R[e]	H	6.3	0.15	Amplifier	For other characteristics, refer to type 6S7									
6SA7	Pentagrid converter[g]	B3	8R	H	6.3	0.3	Mixer	100 250	− 2.0 − 2.0	100 100	8.0 8.0	3.2 3.4	500,000 800,000	Grid 1 resistor, 20,000 ohms Conversion conductance, 450 micromhos			
6SC7	Twin triode amplifier	B3	8S	H	6.3	0.3	Each unit as amplifier	250	− 2.0	...	.....	2.0	53,000	1325	70		
6SF5	High-mu triode	B3	6AB	H	6.3	0.3	Class A amplifier	250	− 2.0	...	.....	0.9	66,000	1500	100		
								90[h] 300[h]	Self-bias, 8800 ohms Self-bias, 3200 ohms	grid resistor,[i] 0.5 megohm					gain per stage = 43 gain per stage = 63		
6SJ7	Triple-grid detector amplifier	B3	8N	H	6.3	0.3	Class A amplifier	100 250	− 3.0 − 3.0	100 100	0.9 0.8	2.9 3.0	700,000 1,500,000	1575 1650	1100 2500		
								90[h] 300[h]	Self-bias, 1700 ohms Self-bias, 860 ohms	grid resistor,[i] 0.5 megohm					gain per stage = 93 gain per stage = 167		

Table 7. Six-volt Receiving Tubes—(*Continued*)

Type	Name	Dimensions socket connections		Cathode type and rating			Use[1]	Plate supply, volts	Grid bias,[a] volts	Screen supply, volts	Screen current, ma.	Plate current, ma.	A.c. plate resistance, ohms	Transconductance (grid-plate), micromhos	Amplification factor	Load for stated power output, ohms	Power output, watts
		Dimen.	S.c.	C.t.	Volts	Amp.											
6SK7	Triple-grid super-control amplifier	B3	8N	H	6.3	0.3	Class A amplifier	100 250	− 3.0 min.	100 100	2.6 2.4	8.9 9.2	250,000 800,000	1900 2000	475 1600		
6SQ7	Duplex-diode high-mu triode	B3	8Q	H	6.3	0.3	Triode unit as class A amplifier	250	− 2.0	...	.....	0.8	91,000	1100	100		
								90[h] 300[h]	Self-bias, 11,000 ohms Self-bias, 3,900 ohms	grid resistor,[i] 0.5 megohm					gain per stage = 40 gain per stage = 53		
6T7-G	Duplex-diode high-mu triode	D8	G-7V[e]	H	6.3	0.15	Triode unit as class A amplifier	135 250	− 1.5 − 3.0	... ...	....	0.9 1.2	65,000 62,000	1000 1050	65 65		
6U5/6G5	Electron-ray tube	D4	6R	H	6.3	0.3	Visual indicator	Plate and target supply = 100 volts. Triode plate resistor = 0.5 megohm. Target current = 1.0 ma. Grid bias, −8 volts; shadow angle, 0°. Bias, 0 volts; angle, 90°; plate current, 0.19 ma.									
								Plate and target supply = 250 volts. Triode plate resistor = 1.0 megohm. Target current = 4.0 ma. Grid bias, −22 volts; shadow angle, 0°. Bias, 0 volts; angle, 90°; plate current, 0.24 ma.									
6U7-G	Triple-grid super-control amplifier	D8	G-7R[e]	H	6.3	0.3	Class A amplifier	100 250	− 3.0 min.	100 100	2.2 2.0	8.0 8.2	250,000 800,000	1500 1600	375 1280		
							Mixer in superheterodyne	100 250	−10.0 −10.0	100 100	.....	......	Oscillator peak volts = 7.0				
6V6	Beam power amplifier	C2	7AC	H	6.3	0.45	Single-tube class A₁ amplifier	180 250	− 8.5 −12.5	180 250	3.0 4.5	29.0 45.0	....	....	....	5,500 5,000	2.00 4.25
							Push-pull class AB₁ amplifier	250 300	−15.0 −20.0	250 300	5.0[f] 5.0[f]	70.0[f] 78.0[f]	....	....	....	10,000 8,000	8.5[g] 13.0[g]
6V6-G	Beam power amplifier	D10	G-7AC[e]	H	6.3	0.45	Amplifier	For other characteristics, refer to type 6V6									
6W7-G	Triple-grid detector amplifier	D8	G-7R[e]	H	6.3	0.15	Class A amplifier	250	− 3.0	100	0.5	2.0	1,500,000	1225	1850		

6X5	Full-wave rectifier	C2	6S	H	6.3	0.6	Maximum a.c. voltage per plate.............................. 350 volts, r.m.s. Maximum d.c. output current.............................. 75 ma.										
6X5-G	Full-wave rectifier	D3	G-6S[e]	H	6.3	0.6	For other ratings, refer to type 6X5										
6Y6-G	Beam power amplifier	D10	G-7AC[e]	H	6.3	1.25	Single-tube class A_1 amplifier	135 200	−13.5 −14.0	135 135	3.0 2.2	58.0 60.0	....	....	....	2,000 2,600	3.6 6.0
6Z7-G	Twin triode amplifier	D3	G-8B[e]	H	6.3	0.3	Class B amplifier	135 180	0 0	...	.....	Power output is for one tube at stated plate-to-plate load				9,000 12,000	2.5 4.2
6ZY5-G	Full-wave rectifier	D3	G-6S[e]	H	6.3	0.3	Maximum a.c. voltage per plate.............................. 350 volts, r.m.s. Maximum d.c. output current.............................. 35 ma.										
36	R.f. amplifier tetrode	D9	5E	H	6.3	0.3	Screen-grid r.f. amplifier	100 250	− 1.5 − 3.0	55 90	..... 1.7[t]	1.8 3.2	550,000 550,000	850 1080	470 595		
							Bias detector	100[u] 250[u]	− 5.0 − 8.0	55 90	....	Grid-bias values are approximate. Plate current to be adjusted to 0.1 ma. with no signal					
37	Detector[j] amplifier triode	D5	5A	H	6.3	0.3	Class A amplifier	90 250	− 6.0 −18.0	... ...	....	2.5 7.5	11,500 8,400	800 1100	9.2 9.2		
							Bias detector	90 250	−10.0 −28.0	... ...	....	Grid-bias values are approximate. Plate current to be adjusted to 0.2 ma. with no signal					
38	Power amplifier pentode	D9	5F	H	6.3	0.3	Class A amplifier	100 250	− 9.0 −25.0	100 250	1.2 3.8	7.0 22.0	140,000 100,000	875 1200	120 120	15,000 10,000	0.27 2.50
39/44	Super-control r.f. amplifier pentode	D9	5F	H	6.3	0.3	Class A amplifier	90 250	− 3.0 min.	90 90	1.6 1.4	5.6 5.8	375,000 1,000,000	960 1050	360 1050		
41	Power amplifier pentode	D5	6B	H	6.3	0.4	Amplifier	For other characteristics, refer to type 6K6-G									
42	Power amplifier pentode	D12	6B	H	6.3	0.7	Amplifier	For other characteristics, refer to type 6F6									
75	Duplex-diode high-mu triode	D9	6G	H	6.3	0.3	Amplifier	For other characteristics, refer to type 6SQ7									
76	Super-triode amplifier detector[j]	D5	5A	H	6.3	0.3	Class A amplifier	100 250	− 5.0 −13.5	... ...	....	2.5 5.0	12,000 9,500	1150 1450	13.8 13.8		
								90[r] 300[r]	Self-bias, 6500 ohms Self-bias, 6400 ohms	Grid resistor,[t] 0.25 megohm					gain per stage = 9 gain per stage = 10		
							Bias detector	250	−20.0 approx.	...	.....	Plate current to be adjusted to 0.2 ma. with no signal					

Table 7. Six-volt Receiving Tubes—(*Concluded*)

Type	Name	Dimensions socket connections		Cathode type and rating			Use^{1}	Plate supply, volts	Grid bias,^{a} volts	Screen supply, volts	Screen current, ma.	Plate current, ma.	A.c. plate resistance, ohms	Transconductance (grid-plate), micromhos	Amplification factor	Load for stated power output, ohms	Power output, watts
		Dimen.	S.c.	C.t.	Volts	Amp.											
77	Triple-grid detector amplifier	D9	6F	H	6.3	0.3	Class A amplifier	100 250	− 1.5 − 3.0	60 100	0.4 0.5	1.7 2.3	650,000 1,500,000	1100 1250	715 1500		
							Bias detector	250	− 1.95	50	Cathode current 0.65 ma.		....	Plate resistor, 250,000 ohms Grid resistor,^{i} 250,000 ohms			
78	Triple-grid super-control amplifier	D9	6F	H	6.3	0.3	Amplifier mixer	For other characteristics, refer to type 6K7									
79	Twin triode amplifier	D9	6H	H	6.3	0.6	Class B amplifier	180 250	0 0	... ...	....	Power output is for one tube at stated plate-to-plate load				7,000 14,000	5.5 8.0
84/6Z4	Full-wave rectifier	D5	5D	H	6.3	0.5	Maximum a.c. voltage per plate.......................... 350 volts, r.m.s. Maximum d.c. output current.......................... 60 ma.										
85	Duplex-diode triode	D9	6G	H	6.3	0.3	Triode unit as class A amplifier	135 250	−10.5 −20.0	... ...	....	3.7 8.0	11,000 7,500	750 1100	8.3 8.3	25,000 20,000	0.075 0.350
89	Triple-grid power amplifier	D9	6F	H	6.3	0.4	As triode^{v} class A amplifier	160 250	−20.0 −31.0	... ...	....	17.0 32.0	3,300 2,600	1425 1800	4.7 4.7	7,000 5,500	0.30 0.90
							As pentode^{w} class A amplifier	100 250	−10.0 −25.0	100 250	1.6 5.5	9.5 32.0	104,000 70,000	1200 1800	125 125	10,700 6,750	0.33 3.40
							As triode^{x} class B amplifier	180	0	...	.....	6.0^{f}	....	....	......	13,600 9,400	2.50^{y} 3.50^{y}
874	Voltage regulator	E4	4S	..	...	....	Minimum d.c. starting supply voltage 125 volts. D.c. operating voltage............ 90 volts.					D.c. operating current............ 10–50 ma. Maximum current (continuous)..... 50 ma.					
876	Current regulator	G1	......	F	...	....	Voltage range.............. 40 to 60 volts.					Operating current.......... 1.7 amp.					
886	Current regulator	G1	......	F	...	....	Voltage range.............. 40 to 60 volts.					Operating current.......... 2.05 amp.					

Note 1: Types with octal bases have *miniature metal cap;* all others have small metal cap.
Note 2: Subscript 1 on class of amplifier service (as AB_1) indicates that grid current does not flow during any part of input cycle. Subscript 2 on class of amplifier service (as AB_2) indicates that grid current flows during some part of the input cycle.

[1] Values to right give operating conditions and characteristics for indicated typical use.

[a] Either alternating current or direct current may be used on filament or heater, except as specifically noted. For use of d.c. on a.c. filament types, decrease stated grid volts by one-half (approx.) of filament voltage.

[b] Grids 3 and 5 are screen. Grid 4 is signal-input control grid.

[c] Supply voltage applied through 20,000-ohm voltage-dropping resistor.

[d] 50,000 ohms.

[e] This diagram is like the one having the same designation without the prefix G, except that pin 1 has no connection.

[f] For two tubes.

[g] Power output is for two tubes at stated plate-to-plate load.

[h] Applied through plate resistor of 250,000 ohms.

[i] For grid of following tube.

[j] For grid-leak detection—plate volts 45, grid return to + filament or to cathode.

[k] This diagram is like the one having the same designation without the prefix G, except that pin 1 is connected to internal shield.

[l] Requires different socket from small 7 pin.

[m] Grid 2 tied to plate.

[n] Megohms.

[q] Both grids connected together; likewise, both plates.

[r] Applied through plate resistor of 100,000 ohms.

[s] Grids 2 and 4 are screen. Grid 3 is signal-input control grid.

[t] Maximum.

[u] Applied through plate resistor of 250,000 ohms or 500-henry choke shunted by 0.25-megohm resistor.

[v] Grid 1 is control grid. Grids 2 and 3 tied to plate.

[w] Grid 1 is control grid. Grid 2 is screen. Grid 3 tied to cathode.

[x] Grids 1 and 2 connected together. Grid 3 tied to plate.

Key to Tube Dimensions

Symbol	Maximum over-all, in. length × diameter	Symbol	Maximum over-all, in. length × diameter	Symbol	Maximum over-all, in. length × diameter	Symbol	Maximum over-all, in. length × diameter	Symbol	Maximum over-all, in. length × diameter
A1	1 3/4 × 1 5/16	C3	3 5/16 × 1 5/16	D5	4 3/16 × 1 9/16	D11	4 11/16 × 1 7/16	E4	5 5/8 × 2 3/16
B1	2 5/8 × 1 1/16	C4	3 1/2 × 1 1/16	D6	4 5/16 × 1 3/16	D12	4 11/16 × 1 13/16	F1	6 1/4 × 2 7/16
B2	2 5/16 × 1 3/16	D1	4 × 1 3/16	D7	4 5/16 × 1 5/8	D13	4 15/16 × 1 9/16	G1	8 × 2 1/16
B3	2 5/8 × 1 5/16	D2	4 1/8 × 1 3/16	D8	4 15/32 × 1 9/16	E1	5 1/32 × 1 13/16		
C1	3 1/8 × 1 5/16	D3	4 1/8 × 1 9/16	D9	4 17/32 × 1 9/16	E2	5 5/16 × 2 1/16		
C2	3 1/4 × 1 5/16	D4	4 3/16 × 1 3/16	D10	4 5/8 × 1 13/16	E3	5 3/8 × 2 1/16		

Table 8. Index of Receiving Types by Use and Cathode Voltage*

Types having similar electrical characteristics (except capacitances) in each cathode-voltage series are grouped within parentheses; *e.g.*, (6SJ7, 6J7, 6J7-G, 6C6)

	Cathode volts											
	1.1	1.4	1.5	2.0	2.5	3.3	5.0	6.3	7.5	12.6	25	30	35, 50
Diode detectors and rectifiers:												
Diode detectors								(6H6, 6H6-G)					
Half-wave vacuum rectifiers								1-v	81	12Z3			35Z4-GT 35Z5-GT
Half-wave vacuum rectifier with power pentode											25A7-G		
Full-wave vacuum rectifiers							(5T4, 5U4-G, 5X4-G, 5Z3), (5W4, 5Y3-G, 5Z4, 5Y4-G, 80), (5V4-G, 83-v)	(6X5, 6X5-G, 84), 6ZY5-G					
Full-wave mercury-vapor rectifiers					82		83						
Full-wave gas rectifiers	Cold-cathode types: 0Z4, 0Z4-G												
Rectifier—doublers											(25Z6, 25Z6-G, 25Z6-GT, 25Z5)		
Duplex-diode detectors with voltage amplifiers:													
Medium-mu triode amplifiers				(1B5/25S, 1H6-G)	55			(6R7, 6R7-G, 85)					
High-mu triode amplifiers		(1H5-G† 1H5-GT†)			2A6			(6SQ7, 6Q7, 6Q7-G, 6Q7-GT, 6T7-G, 6B6-G, 75)		12SQ7, 12Q7-GT			

Pentode—amplifiers........				(1F7-GV, 1F6)	2B7			(6B8, 6B8-G, 6B7)		12C8			
Converters and mixers: Pentagrid converters.......		(1A7-G 1A7-6T)		(1C7-G, 1C6), (1D7-G, 1A6)	2A7			(6SA7, 6A8, 6A8-G, 6A8-GT, 6D8-G, 6A7)		12SA7, 12A8-GT			
Triode-hexode converter....								6K8					
Pentagrid mixers..........								(6L7, 6L7-G)					
Voltage amplifiers, detectors, oscillators: Triodes—medium mu, single unit	11, 12	1G4-G	26	(1H4-G, 30)	27, 56	99's	00-A, 01-A 40	(6C5, 6C5-G), (6J5, 6J5-G, 6J5-GT), 6L5-G, 76, 37					
Triodes—high mu, single unit								(6SF5, 6F5, 6F5-G, 6F5-GT), 6K5-G					
Triodes—medium mu, twin unit..................								(6C8-G, 6F8-G)					
Triodes—high mu, twin unit.								65C7		12SC7			
Tetrodes—sharp cut-off.....				32	24-A	22		36					
Tetrodes—remote cut-off....					35								
Pentodes—sharp cut-off.....		(1N5-G 1N5-GT)		(1E5-GP 1B4-P), 15	57			(6SJ7, 6J7, 6J7-G, 6J7-GT, 6C6), 77		(12SJ7, 12J7-GT)			
Pentodes—remote cut-off....				(1D5-GP, 1A4-P), 34	58			(6SK7, 6K7, 6K7-G, 6K7-GT, 78), (6S7, 6S7-G), (6U7-G, 6D6), 6W7-G, 39/44 6AB7/1853‡ 6AC7/1852‡ 1851		(12SK7, 12K7-GT)			
Pentode—remote cut-off, with triode..............								6F7					
Power amplifiers: Triodes—low mu, single unit				31	2A3, 45	20	71-A, 112-A		10, 50				

Table 8. Index of Receiving Types by Use and Cathode Voltage*—(*Concluded*)

Types having similar electrical characteristics (except capacitances) in each cathode-voltage series are grouped within parentheses; *e.g.*, (6SJ7, 6J7, 6J7-G, 6C6)

	Cathode volts												
	1.1	1.4	1.5	2.0	2.5	3.3	5.0	6.3	7.5	12.6	25	30	35, 50
Triodes—high mu, single unit				49	46			6AC5-G					
Triodes—high mu, twin unit		1G6-G		(1J6-G, 19)	53			(6N7, 6N7-G, 6A6), 6Z7-G, 79					
Tetrodes												48	
Beam power tubes		1Q5-GT						(6L6, 6L6-G), (6V6, 6V6-G), 6Y6-G, 6V6-GT, 6AG7‡			(25L6, 25L6-G 25L6-GT)		35L6-GT 50L6-GT
Pentodes		1A5-G 1C5-G		(1F5-G, 1F4), 1G5-G, 33, 1E7-G§	2A5, 47, 59			(6F6, 6F6-G, 42), (6K6-G, 6K6-GT, 41), 6G6-G, 38, 6A4, 89			(25A6, 25A6-G, 43), 25B6-G		
Diode-triode—power pentode		1D8-GT											
Direct-coupled amplifiers								(6B5, 6N6-G)					
Pentode with diode rectifier											25A7-G		
Electron-ray tubes:													
With sharp cut-off triode								6E5					
With remote cut-off triode								6N5, 6U5/6G5					
Twin-indicator type								6AF6-G					
Gas triode	Cold-cathode, starter-anode type: 0A4-G.												
Regulators:													
Voltage	874												
Current	876, 886												

---

[*] RCA Radiotron Division, RCA Manufacturing Company, Inc.

[†] Single diode.

[‡] For television applications.

[§] Two 1F4's in same bulb.

amplifier of a radio receiving set, transformers having a ratio of 3:1 or less should be used.

If the **maximum power output** for a given input voltage is desired, the external impedance is made just equal to the internal impedance of the tube. If the **maximum undistorted power output** which the tube can handle is desired, the external impedance is made equal to twice the tube impedance and the input voltage is raised to its limiting value. In no case should the input voltage be made so large that appreciable grid current flows. Where power output is desired, the reactance in the external circuit should be reduced to a minimum.

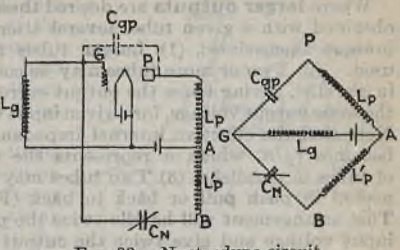


FIG. 33.—Neutrodyne circuit.

The **precautions** to be observed in operating such amplifiers are numerous and complex. Good apparatus and good electrical connections are of course essential. A common trouble, especially at high frequencies, is that of keeping the electrical quantities in the assigned paths and keeping all disturbances out. This involves the difficult and complex problem of **shielding.** A peculiar type of such disturbance occurs by means of the *feed back* of energy from the external plate circuit to the external grid circuit through the grid-to-plate capacitance of the tube itself. This may be eliminated by means of a four-element tube (screen grid) or it may be neutralized by means of an auxiliary balancing circuit as in the **neutrodyne** (Fig. 33). No high impedance should be common to two or more stages of an amplifier except that specifically inserted for coupling purposes. The output, voltage or power, is not uniform over a *range of frequencies* if the external impedance is a function of frequency; at high frequencies this condition always holds owing to the shunting effect of the plate-to-filament capacitance of the tube itself. The *grid current* cannot be neglected if the input circuit has a high impedance. With very low impedance input circuits, the tube may be operated about some such point as B (Fig. 30) and the grid allowed to swing to a positive potential.

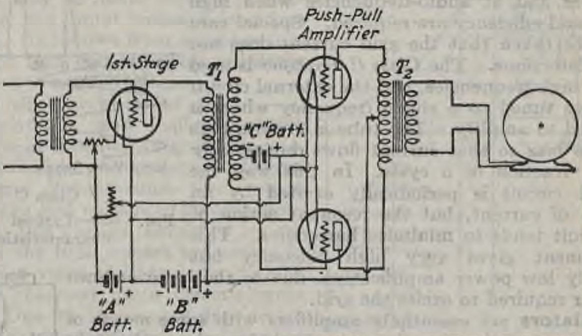
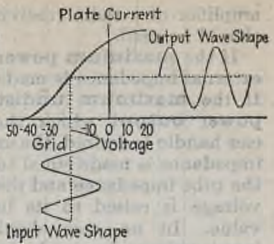


FIG. 34.—Push-pull amplifier circuit. (*Moyer and Wostrel, "Radio Handbook," McGraw-Hill*, 1931.)

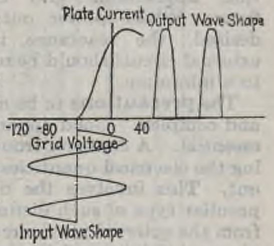
Such an arrangement will greatly increase the power output of the tube, but care must be exercised that the grid current has no deleterious effect.

Where **larger outputs** are desired than can be obtained with a given tube, several alternatives present themselves: (1) Larger tubes may be used. (2) Two or more tubes may be connected in parallel, giving twice the output current, and the same output voltage, for a given input voltage. Note that the effective internal impedance then becomes r_p/n, where n represents the number of tubes in parallel. (3) Two tubes may be connected in push pull or back to back (Fig. 34). This arrangement will handle twice the previous input voltage and give twice the output voltage with the same current. Owing to the fact that at any instant the distortion produced by one tube tends to neutralize that produced by the other, this arrangement is capable of giving somewhat more than twice the undistorted output of a single tube.

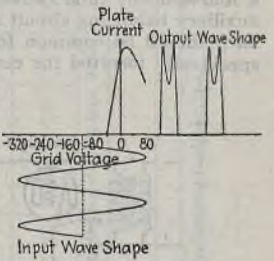
Two other classes of amplifier are worthy of mention. The *Class B amplifier* is normally a push-pull arrangement with the operating point fixed at or near cutoff (Fig. 35). Each tube carries current for one-half cycle and the output is essentially a true reproduction of the input. It is used at high frequencies to amplify modulated waves and at audio-frequencies when high output and efficiency are required. Special care should be taken that the grid current does not cause distortions. The *Class C amplifier* is used only at high frequencies, and the external circuit is always tuned to a single frequency which it is desired to amplify. The tube is given a high negative bias so that current flows during only a small fraction of a cycle. In this way the external circuit is periodically excited by an impulse of current, but the resonant action of the circuit tends to minimize harmonics. This arrangement gives very high efficiency but relatively low power amplification, due to the large amount of power required to excite the grid.



Class A



Class B



Class C

FIG. 35.—Typical amplifier characteristics.

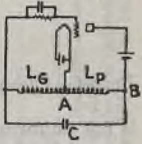


FIG. 36.—Hartley oscillator.

Oscillators are essentially amplifiers with some means of feeding part of the output energy back into the grid circuit for the purpose of maintaining the electrical vibrations. The oscillations are started by some sort of an electrical disturbance such as the closing of a switch; the *frequency* of the oscillations is given by $f = 1/(2\pi\sqrt{LC})$, where L and C are, respectively, the inductance and capacitance of the oscillating circuit. Figure 36 illustrates a so-called *Hartley oscillator*. The circuit between A and B represents the output impedance of the tube and L_g

furnishes the necessary feed back. $L = L_p + L_g + 2M$, where M is the mutual inductance between L_p and L_g. In order to have good *wave form*, the output impedance should be large and the feed back small. If possible the oscillator should be separated from the measuring circuit so that load variations will not affect the frequency or amplitude of the oscillator output. With this arrangement the oscillator may be worked at high efficiency, similar to Class C amplifiers and the wave form corrected in the intermediate circuits.

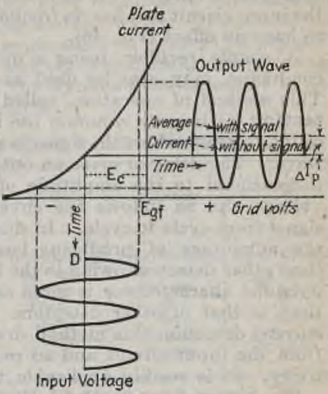


FIG. 37.—Detector action.

The operation of the amplifier is limited, for the most part, to the linear portion of the tube characteristic; the oscillator (and classes B and C amplifiers) swing over the entire tube characteristic; the operation of the **detector** is limited, for the most part, to the curvature of a tube characteristic. Either the $i_p - e_g$ or the $i_g - e_g$ characteristic may be used. From the diagram (Fig. 37), it is seen that a sinusoidal input voltage causes unequal increase and decrease of the resultant current. The increase (in this case) of the average current is a measure of the **rectification effect.** Mathematical analysis shows that it is proportional to the square of the input voltage for small inputs, though by proper arrangements it may be made directly proportional to the input voltage for large voltages. It follows from what has just been said that detectors are relatively insensitive to small voltages.

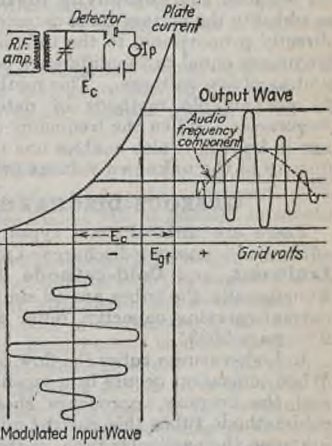


FIG. 38.—Grid-bias detection of a modulated wave.

When the curvature of the $i_g - e_g$ characteristic is used (called *grid-leak and condenser detection* (Fig. 39), or *grid-circuit detection*), the unequal changes in the grid current cause the average value of the grid voltage to become more negative; this change, amplified by the action of the tube, causes a decrease in the average value of plate current which may be observed on a direct-current meter. Due to the amplifying action of the tube this method *is more sensitive* for small voltages than that described below, but it has two compensating disadvantages: first, that grid current flows and therefore power is consumed in the input circuit; and second, that the effective input voltage varies with frequency owing to the effect of the condenser shunting the grid leak. With due care these disadvantages may be minimized for any given installation.

When the curvature of the $i_p - e_g$ characteristic is used (called *plate-circuit detection*) the C battery is adjusted so that operation is on the point of greatest curvature. This method will handle *larger signals* than the grid-leak and condenser method but is less sensitive. It consumes a minimum of power in the input circuit and has no frequency error until the input capacitance begins to have an effect (Fig. 38).

A simple rectifier, being a unidirectional conductor, may also be used as a detector. This method of operation, called **diode detection,** is in very common use in radio receiving circuits where the signal is a modulated wave. The detector gives an output voltage proportional to the amplitude of the signal voltage and so follows the envelope of the signal from cycle to cycle. In this field it has the advantage of producing less distortion than other detectors owing to the fact that its dynamic characteristic is more nearly linear than is that of other detectors. Like grid-current detection this method draws current from the input circuit and so reduces selectivity. It is readily applicable to large signals.

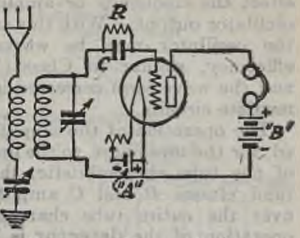


FIG. 39.—Non-oscillating detector with grid leak and condenser. (*Moyer and Wostrel, "Radio Handbook."*)

The **heterodyne method** (Fig. 40) of detection is used to detect very small inputs. In this method an auxiliary voltage of slightly different frequency (it is called the **homodyne method** if the two frequencies are the same) is added to the voltage to be detected in the input circuit. The output is then directly proportional to the unknown voltage and of a frequency equal to the difference between the unknown and auxiliary voltages. This method makes it possible to use audible methods of detection at *supersonic frequencies.* When the frequency of the auxiliary voltage is known, it also enables one to determine the frequency of the unknown voltage (see also Rectifiers).

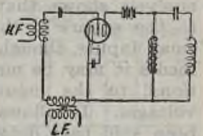


FIG. 40.—Connections for heterodyne detection. (*Van der Bijl, "Thermionic Vacuum Tubes."*)

GASEOUS-DISCHARGE TUBES

There are three distinct types of three-element, or controlled, gaseous-discharge tubes: **Thyratrons, Ignitrons,** and **Cold-cathode** discharge tubes. Functionally the tubes are all similar, but their characteristics, particularly current-carrying capacity, differ greatly in magnitude, as shown in Table 9 on page 2683.

In high-vacuum tubes the flow of current is restricted by the space charge. When ionization occurs in a gas-filled tube, the space charge is neutralized, and the current approaches short-circuit values. In the thyratron and cold-cathode tubes this current must be restricted to definite values, depending upon the particular tube, or the cathode will be destroyed by positive-ion bombardment; the ignitron has no such limitation.

All three tubes depend upon the initiation of ionization by a voltage applied to the control circuit. The current in the control or grid circuit of the thyratron and cold-cathode tube required to initiate ionization is of the order of a few microamperes; in the ignitron it is of the order of amperes. If desired, the control current of the ignitron may be supplied by the discharge current of a thyratron which is in turn controlled by a much smaller current.

Once ionization is started, the third element or grid loses all control. In order for it to regain control, it is necessary to interrupt the discharge for a period long enough for the tube to be deionized. (Similarly, to start the tube, the impulse or signal must be of sufficient duration, as well as magnitude, to initiate ionization.) The four generally used methods of extinguishing the discharge are (1) use of an alternating anode voltage, (2) opening the anode circuit by a switch or relay, (3) application of a surge of negative voltage to the anode through a capacitor, and (4) overshooting of the voltage due to the inductance of the circuit and the dynamic characteristic of the tube when a capacitor is discharged through it.

Table 9. Comparison of Gas-filled Control Tubes*†

Characteristics	Thyratron	Ignitron	Cold-cathode tube
Current capacity	Up to 100 amp.	5–10,000 amp.	Up to 100 ma.
Deionization time	10^{-4} sec.	10^{-4} sec.	10^{-2} sec.
Ionization time	10^{-6} sec.	10^{-6} sec.	10^{-4} sec.
Cathode heating time	Finite	0	0
Deterioration in standby service	Yes	No	No
Accuracy of characteristics	± 2 volts	Variable	± 10 volts
Sustaining voltage	15 volts	15 volts	75 volts

* Taken from article by Ingram in *Elec. Eng.*, July, 1939, p. 345.

† All values are approximate only.

The voltage required to initiate the discharge in a mercury-vapor tube depends upon the temperature, whereas in a tube filled with a noble gas, such as argon, it does not.

For refined control of the discharge the phase of the voltage applied to the grid may be shifted with respect to the voltage applied to the anode. In this way the discharge may be limited to any desired portion of one-half cycle.

RECTIFIERS

Rectifiers are devices for converting alternating to direct current. They are made in such varied and diverse types that it will be possible to mention only a few of the more important here.

For power work the **mercury vapor arc** is becoming an active competitor with the rotary converter. This arc is not self-starting (an auxiliary device being normally used for that purpose) but with load is self-sustaining. The source of electrons is the hot spot in the pool of mercury near the cathode, the required heat being produced by positive-ion bombardment. The positive ions serve also to reduce the "space-charge effect" of the electrons and so to hold the voltage drop across the tube to less than 15 volts thus permitting good voltage regulation. Since only one of the terminals or electrodes is hot, it alone will serve as a source of electrons so that when an alternating voltage is applied to the terminals, the flow of current will be unidirectional.

Where a polyphase source of energy supply is available, use is frequently made of **polyphase rectification**, which has the advantage of giving a more constant d.c. output with a cheaper filter. One simple way of doing this with a three-phase, Y-connected system is to attach the anode of each of three tubes to one terminal of each of the phase windings. The cathodes, or common point of the three filament circuits, are connected to the positive bus of the d.c. load, and the common point or neutral of the three-phase windings is connected to the negative bus of the d.c. load. As each of the anodes in turn is raised in potential above the neutral point (plus the potential of the d.c. load), that tube conveys a pulse of energy to the d.c. load. Thus each tube will convey energy to the load, always in the same direction, during approximately one-third of a cycle. Six-phase and twelve-phase rectifiers are also in common use.

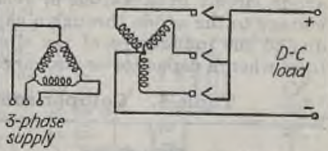


Fig. 40a.—Polyphase rectification.

Of the **separately maintained rectifiers**, *i.e.*, those using some auxiliary means to maintain the supply of free electrons, three in quite general use have externally heated cathodes as the sources of electrons. These three are the thermionic tubes and gaseous-discharge tubes of high-voltage and low-voltage types.

Thermionic tubes have unilateral conductivity and so may be used as rectifiers. Owing to the absence of gas they may be used on very high voltages but can pass only a very small current, limited by the saturation current of the cathode, and they have a high internal voltage drop.

Hot-cathode, gaseous-discharge tubes are of two different types. The *high-voltage type*, of which the 866 tubes or **Phanotron** is representative, contains only a small amount of mercury vapor so that the tube will still sustain relatively high voltages, 5000 to 7500 volts as a peak value, and in addition the internal voltage drop is less than 15 volts. The *current*, however, must be limited to the emission current of the cathode, since with greater currents the cathode fall of potential rises and the positive-ion bombardment is severe enough to strip the emitting surface from the cathode.

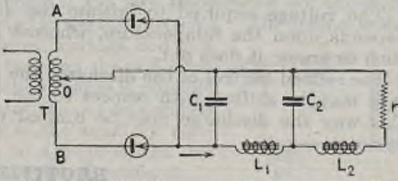


Fig. 41.—Rectifier and filter circuit. (*Van der Bijl, "Thermionic Vacuum Tubes."*)

The *low-voltage type*, of which the **Tungar** and **Rectigon** are typical contain argon at 3 to 8 mm. pressure and will pass 10 or 12 amp. at very low voltages. The maximum permissible voltage is only 100 to 200 volts. The current flow may be greater than the emission current from the cathode since the high gas pressure protects the cathode from excessive positive-ion bombardment.

Several other types of rectifiers are used occasionally but only for small currents and small voltages. Among these are *mechanical vibrators*, *copper oxide rectifiers*, and *point-to-plate rectifiers*.

The point-to-plate rectifier supplies small unidirectional currents without any accessory apparatus, such as heating transformers, etc. The pointed electrode has a high potential gradient around it and so will produce the ionization necessary for the maintenance of the discharge. When it is the anode, the fast-moving electrons are easily able to reach it; when it is the cathode, the electrons are impeded by the gas molecules from reaching the anode. Highly insulated transformers are required with such tubes to prevent breakdown due to surges.

Copper oxide rectifiers are, as their name implies, copper disks on which an oxide coating has been formed by sudden quenching. When used with another electrode, such as lead, they pass current unidirectionally provided the current density is low.

Filters should be used with all the above types of rectifiers if steady output voltage is desired. An electric filter consists essentially of electrical reservoirs (condensers) on the input and output ends to take up the fluctuations in source and load, and a choke coil to prevent these fluctuations from reaching from input to output and vice versa. Frequently the choke coil is split and a third condenser added at its mid-point. When the load resistance r does not vary, the circuit shown in Fig. 41 may be more desirable.

RHEOSTATS

Rheostats are adjustable resistances for controlling various circuits. Wire- or strip-wound resistance units with insulating supports arranged with suitable switches are commonly used as field rheostats, starting boxes, and for other purposes.

Carbon rheostats of the compression type are useful for fine adjustment. The resistance varies nearly inversely as the pressure, permitting a wide range of values. The resistance decreases with temperature increase, but this may not be objectionable for laboratory work.

Slide-wire rheostats are convenient for a variety of laboratory uses. These are obtainable in sizes from a fraction of an ohm to several thousand ohms, usually for use in circuits of less than 250 volts.

Water-cooled rheostats can be used to absorb large amounts of energy. Galvanized iron wire submerged in running water makes a suitable rheostat, where the various turns are kept apart. This is not satisfactory for continuous operation on direct current because of slime forming on the wire. The water should be kept below the boiling temperature.

Liquid rheostats are built in a variety of forms. A plate partially lowered into a watertank or barrel permits a variation in resistance. Pure water is not suitable for voltages below 1000; for lower voltages, salt or acid is used to lower the resistance. For moderate amounts of energy a small pipe may be lowered concentrically in a tank made of a larger pipe with suitable fittings. Table 10 gives data on such rheostats made from standard wrought-iron pipe.

Table 10. Data for Concentric-cylinder Water Rheostats*

Capacity, kilowatts	Volts	Amperes	Resistance, ohms	Specific resistance, ohms per in. cube	Nominal diameters of pipes, in.		Depth of immersion, in.
					Inner	Outer	
2	110	18.2	6.05	1000	¼	2	35
5	110	45.4	2.42	500	¾	3	35
10	110	91.0	1.21	500	½	1¼	33
2	110	18.2	6.05	2000	¾	2	36
5	110	45.4	2.42	1000	½	1¼	33
10	110	91.0	1.21	1000	2	3	34
5	110	45.5	2.42	2000	2	3	34
10	110	90.0	1.21	2000	3	4	37

* Marks, "Mechanical Engineers' Handbook," 3d ed., p. 2013, McGraw-Hill.

Iron pipe is occasionally used for busbars at high voltage. It may be arranged as a resistor at low voltages. For alternating current the skin effect increases the effective resistance over that for direct current several times. H. B. Dwight gives the approximate formula for the ratio of alternating-current resistance to direct-current resistance as $2\pi S\sqrt{\delta\mu f/P}$, where S is the area of conductor in square centimeters, P the perimeter in centimeters, δ the electric conductivity in 10^9 mhos per cm. cube at the operating temperature, μ the direct-current permeability, and f the frequency. Curves for 2.5-in. pipe show this ratio (checked by experiments) to be about 5 to 14 for 60 cycles [see *Elec. J.*, **23**, 295 (1926)].

MAGNETS

Magnets are used in lifting devices, in many and varied types of relays and in other types of apparatus.

Permanent magnets rely on residual magnetism for their usefulness The m.m.f. retained is approximately constant. The resultant constant field is particularly desirable for magnetos, instruments, and meters. The ability of steel to retain its magnetism depends on the kind of steel, its heat treatment, and the shape of the particular piece. Short magnets tend to become demagnetized unless made glass hard. Steels with tungsten, chromium, or cobalt are used for permanent magnets.

The steel to be used as a permanent magnet is usually magnetized electrically, the particular method varying with the shape of the magnet. In order that the field may be constant the magnet is usually overmagnetized and then artificially aged. Aging may be hastened by temperature, mechanical shock, vibration, or demagnetizing fields.

Alloys of iron, nickel, and aluminum have been developed which have very high coercive force and high residual induction. These qualities permit short magnets of small volume which are little affected by vibration, stray fields, or temperature up to 500°F. These alloys are difficult to machine and are usually cast so that a minimum of grinding to size is necessary. Comparative data of various materials used for permanent magnets is given by Williams [*Elec. Eng.*, **55**, 19 (1936)]; Webb [*Inst. Elec. Eng.*, **82**, 303 (1938)]; and Adams [*Gen. Elec. Rev.*, **41**, 518 (1938)].

Electromagnets depend on electric currents in coils for their m.m.f. and are usually made of soft iron so that the flux will vary with the magnitude of the current in the coil.

The pull of a magnet, where the air gap between the stop and the armature is small, may be expressed by the equation

$$F = kB^2$$

where F is in dynes per square centimeter, $k = 0.0398$, and B in gauss. For B in lines per square inch and F in pounds per square inch,

$$k = 1.39 \times 10^{-8}$$

With direct-current magnets the current in the solenoid is limited by the resistance of the winding, giving a constant m.m.f. regardless of position of the armature. As the air gap is reduced, the flux density increases greatly increasing the pull. In order to increase the pull and reduce the time of moving an armature, the coil is frequently wound for a lower voltage and is provided with a resistance which is automatically inserted in the coil circuit as the armature comes to its final position.

Solenoids are a very common form of electromagnet, used where a rod or plunger is to be moved through some distance. The coil is wound on a tube into which the plunger is drawn. The pull increases as the plunger approaches the center of the coil (Fig. 42), at which time the pull F in pounds is

$$F = \frac{CANI}{l}$$

where l is the length of the solenoid in inches; C is the unit pull per ampere-turn per inch of the coil and depends on the proportions of the coil and the properties and the length of the plunger; A is the cross section of the plunger in square inches; and NI is the ampere-turns of the coil (see Table 11). The pull may be increased at the end of the stroke by adding a magnetic stop (Fig. 42).

Solenoids must be designed with sufficient radiating surface so that the temperature does not become excessive. The design must be sufficiently liberal so that with the increased resistance of the coil at its maximum temperature the required current will flow. A solenoid may be rewound for another voltage by changing the size of wire. If a solenoid operates satisfactorily with E volts when wound with a wire of bare diameter D, it may be rewound with wire of bare diameter d to operate on voltage e, where $D^2E = d^2e$. The new winding will occupy the same space as the old except for the space occupied by the insulation [see Brooks, *Industrial Engineering*, **87**, 349 (1929)].

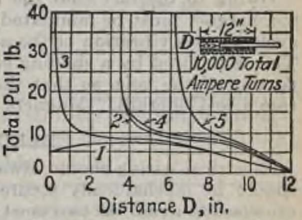


FIG. 42.—Pull of solenoid on plunger. 1, Coil and plunger; 2, coil and plunger with stop; 3, iron-clad coil and plunger; 4 and 5 same as 3 with different length stop. (*Marks*, "*Mechanical Engineers' Handbook.*")

Magnets are built into plates to form *chucks* for holding steel while it is being ground. *Magnetic separators* of many forms are built for separating magnetic material from other materials (see p. 1730).

Table 11. Maximum Pull per Square Inch of Core for Solenoids with Open Magnetic Circuit*

Length of coil, in.	Length of plunger, in.	Area of core, sq. in.	Total amp. turns	Max. pull, lb. per sq. in.	1000 × C
6	Long	1	15,000	22.4	9.0
9	Long	1	11,330	11.5	9.1
9	Long	1	14,200	14.6	9.2
10	10	2.76	40,000	40.2	10.0
10	10	2.76	60,000	61.6	10.3
10	10	2.76	80,000	80.8	10.1
12	Long	1	11,200	8.75	9.4
12	Long	1	20,500	16.75	9.8
18	36	1	18,200	9.8	9.7
18	36	1	41,000	22.5	9.8
18	18	1	18,200	9.8	9.7
18	18	1	41,000	22.5	9.8

* "Standard Handbook for Electrical Engineers," McGraw-Hill, 1933.

Lifting magnets are particularly useful in handling magnetic materials such as scrap iron, billets, pigs, and castings.

Alternating-current magnets inherently have considerable reactance, which limits the current and the flux. The relations are essentially the same as in a transformer thus

$$\phi_{max} = \frac{E \times 10^8}{(4.44fN)}$$

where E is the impressed voltage, f is the frequency, and N is the number of turns. This indicates that the flux and consequently the pull are nearly the same regardless of the armature position instead of increasing rapidly with decreased air gaps as in the d.c. magnet.

Owing to the fact that the flux is continuously changing the iron of the a.c. magnet must be laminated. The pull, being reduced to zero twice each cycle, causes vibration of the armature and consequently considerable noise. This is reduced by a shading coil consisting of a short-circuited coil about a portion of the pole, so that there is some pull on the armature at all times (see also Underhill, "Magnets," McGraw-Hill, 1924).

ELECTRICAL WIRING

Electrical wiring should always be installed with care and neatness; joints should be mechanically secure, well soldered, and taped. Care should be exercised to avoid the two most common faults, (*a*) incomplete circuits and (*b*) short circuits. All installations should comply with the "National Electrical Code" and with the "National Electrical Safety Code."

The **minimum size of wire** for a particular installation must be large enough (*a*) to provide *carrying capacity* as indicated in the Electrical Code, (*b*) to have *sufficient strength* on spans as indicated in Safety Code, and (*c*) so that the resistance and reactance drop permit *good regulation*. **Regulation** is the rise in voltage from full load to no load expressed as a percentage of the full-load voltage. Good regulation varies with the particular application and while 3 per cent may be a maximum variation for lighting, 10 per cent may be reasonable for motors.

The regulation for a **direct-current circuit** may be easily calculated, as the voltage change from full load to no load is the IR drop of the circuit. The resistance of the circuit (two wires) may be obtained from Table 12; the current is determined from the load.

Example. To find the regulation of a line suppling a 50-kw. d.c. load at 220 volts. The line is 200 ft. long and consists of two 0000 wires.

$$\frac{50{,}000 \text{ watts}}{220 \text{ volts}} = 227.3 \text{ amp.}$$

Resistance of 1000 ft. 0000 wire = 0.049 ohm
Resistance of 400 ft. 0000 wire = 0.0196 ohm
The drop = 227.3 amp. × 0.0196 ohm = 4.46 volts

$$\text{Regulation} = \frac{(4.46 \times 100)}{220} = 2 \text{ per cent}$$

In an **alternating-current circuit,** the power factor of the load and reactance of the circuit must be taken into account.

Example. To find the regulation of a three-phase line supplying 440 volts to a 90-kva. load of 0.8 power factor. The line is 2000 ft. long and consists of three 0 wires with 24-in. spacing of wires.

From Table 12 resistance of 0 wire is 0.0983 per 1000 ft.
From Table 12, reactance of 0 wire is 0.121 per 1000 ft. at the given spacing
Resistance of 2000 ft. = 0.197
Reactance of 2000 ft. = 0.242
Calculations should be per phase. Assume Y connection.
Kilovolt-amperes per phase = 30

Volts per phase, $440/\sqrt{3} = 254$
Amperes per phase, $30{,}000/254 = 118$
IR drop per phase, $0.196 \times 118 = 23.2$
IX drop per phase, $0.240 \times 118 = 28.3$

Referring to vector diagram $E \cos \theta = 203.2$ and $E \sin \theta = 152.4$.

$203.2 + 23.2 = 226.4$
$152.4 + 28.3 = 180.7$

Sending voltage per phase $= \sqrt{(226.4)^2 + (180.7)^2} = 289.5$

Sending voltage $= 289.5\sqrt{3} = 501.4$

$$\text{Regulation} = \frac{(501.4 - 440)100}{440} = 14 \text{ per cent}$$

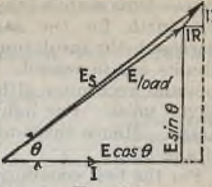


FIG. 43.

The **most economical size of wire** is that which gives the lowest annual cost. The annual cost consists of the fixed or capital costs (interest on investment, depreciation, and taxes) and the cost of energy (copper losses) in the wires. In general, the *minimum annual cost* will be found for that size of wire for which the fixed cost is equal to the cost of energy loss in the wire. This wire size may be larger than the permissible minimum. It is desirable when installing electrical wiring to anticipate future increase of loads.

Table 12. Wire Table

Size, American wire gage	Area, cir. mils	Diameter, in.	Weight bare, lb. per 1000 ft.	Resistance, ohms per 1000 ft. at 20°C.	Reactance per wire per 1000 ft. at 60 cycles—distance between centers						Allowable carrying capacity, in amperes, with rubber insulation
					1 in.	6 in.	1 ft.	2 ft.	4 ft.	8 ft.	
	1,000,000	1.152	3,090	0.0108	.....	0.063	0.079	0.095	0.111	0.127	377
	800,000	1.031	2,470	0.0135	.....	0.066	0.082	0.098	0.113	0.129	340
	600,000	0.893	1,850	0.0180	0.027	0.069	0.085	0.101	0.117	0.133	293
	500,000	0.815	1,540	0.0216	0.030	0.071	0.087	0.103	0.119	0.135	265
	400,000	0.728	1,240	0.0270	0.032	0.073	0.089	0.105	0.121	0.137	233
	300,000	0.630	926	0.0361	0.035	0.076	0.092	0.108	0.124	0.140	198
	250,000	0.575	772	0.0433	0.038	0.079	0.095	0.111	0.127	0.143	177
0000	211,600	0.460	653	0.0490	0.040	0.081	0.097	0.113	0.129	0.145	160
000	167,800	0.410	518	0.0618	0.042	0.083	0.099	0.115	0.131	0.147	138
00	133,100	0.365	411	0.0779	0.045	0.086	0.102	0.118	0.134	0.150	120
0	105,500	0.325	326	0.0983	0.047	0.089	0.105	0.121	0.137	0.153	105
1	83,690	0.289	258	0.124	0.050	0.091	0.107	0.123	0.139	0.155	91
2	66,370	0.258	205	0.156	0.053	0.094	0.110	0.126	0.142	0.158	80
4	41,740	0.204	126	0.248	0.058	0.099	0.115	0.131	0.147	0.163	60
6	26,250	0.162	79.5	0.395	0.063	0.104	0.120	0.136	0.152	0.168	45
8	16,510	0.128	50.0	0.628	0.069	0.110	0.126	0.142	0.158	0.174	35
10	10,380	0.102	31.4	0.999	.....	.....	.....	.....	.....	.....	25
12	6,530	0.081	19.8	1.59	.....	.....	.....	.....	.....	.....	20
14	4,107	0.064	12.4	2.52	.....	.....	.....	.....	.....	.....	15
16	2,583	0.051	7.8	4.02	.....	.....	.....	.....	.....	.....	7
18	1,624	0.040	4.9	6.38	.....	.....	.....	.....	.....	.....	5

Note. Values for wire sizes larger than 0000 are for stranded wire. Allowable current-carrying capacity is for code rubber insulation without more than three conductors in raceway, as shown in the 1940 National Electrical Code. Greater current is allowed for other insulations and for conductors in open air.

The *higher the voltage* the less the copper required for a given load. Since the load is equal to EI, if the voltage is doubled the current is halved and the copper cross section may be halved for the same current density or reduced to one-fourth for the same percentage voltage drop. When voltages are increased, the insulation problems and hazards to employees working on live circuits are increased. Hence, for industrial motors, 440 (460) volts is the common maximum, although larger motors are operated at 2300, 6600, and at 13,200 volts. For lighting, 115-volt lamps are more rugged than 230-volt lamps. Hence three-wire circuits are frequently used for lighting (see Figs. 54 and 55).

For the best economy and regulation, the loads on the two sides of a three-wire circuit should be approximately **balanced.** Likewise in connecting single-phase loads to a polyphase circuit, the phase loads should at all times be approximately balanced. Polyphase motors normally draw balanced currents tending to decrease the effect of other loads which may not be balanced. The maximum permissible unbalance depends on the particular installation and would, in general, be about 10 to 15 per cent of the full-load value.

Approximate Values of Wire. No. 10 wire has an approximate diameter of 0.1 in., a cross section of 10,000 circular mils, and a resistance of about 1 ohm per 1000 ft. The cross section doubles for every third size of wire (smaller size number), and the resistance is reduced to one-half. Again, the cross section increases tenfold for ten sizes of wire and the resistance is one-tenth.

COST OF POWER

The **rates** for energy from a utility to an industrial consumer must include items to cover the *fixed charges* and *operating costs* of the utility. These costs are similar to those of a privately owned plant.

The **demand charge** in the rate is to cover a portion of the fixed charges. This demand charge is usually based on the maximum 15-min. load, inasmuch as the utility must have available sufficient generating and distributing equipment to supply that load. Since the peak load of all industries will not occur during the same period, the demand charge is less than the fixed charges for an equivalent private plant. Since the apparatus to supply a given amount of power is determined by the voltage and the current, the required rating is greater if the power factor is low, and, therefore, the demand charge is frequently based on kilovolt-amperes rather than kilowatts.

The **energy charge** in the rate must cover the operating costs including coal, labor, distribution losses, etc., and frequently includes some of the fixed charges. This charge, per kilowatt-hour, may be decreased with increase of kilowatt-hours used.

Owing to better efficiency and lower first cost of large units, and frequently better operating conditions and diversity of loads, power from a utility costs less than that from an industrial power plant, except in the case of unusual local conditions, such as large steam requirements for process work.

The true cost of power in an industry should be valued in terms of increased efficiency of labor, better product, etc. In most industries the cost of power is less than 4 per cent of the value added by the manufacture of the product. It may, therefore, be economical to add a large percentage to the cost of apparatus or power, provided it will increase the value of the product by a small amount.

ILLUMINATION

Correct lighting has proved its ability to *increase production* economically, to give *greater comfort* to workmen, and to *reduce accidents.* Because of the proved value of increased illumination, lighting standards have been gradually raised. **Good illumination** requires light of *sufficient intensity* in the *correct place, without glare,* and usually with only *soft shadows.*

Table 13. Recommended Levels of Illumination for Industrial Interiors*

	Foot-candles recommended
Aisles, stairways, passageways	3– 2
Assembling:	
Rough	8– 5
Medium	12– 8
Fine	20–12
Extra fine	100–25
Chemical Works:	
Hand furnaces, boiling tanks, stationary driers, stationary or gravity crystallizing	5– 3
Mechanical furnaces, generators and stills, mechanical driers, evaporators, filtration, mechanical crystallizing, bleaching	6– 4
Tanks for cooking, extractors, percolators, nitrators, electrolytic cells	10– 6
Clay Products and Cements:	
Grinding, filter presses, kiln rooms	5– 3
Molding, pressing, cleaning and trimming	8– 5
Enameling	10– 6
Color and glazing	15–10
Electric Manufacturing:	
Storage battery, molding of grids, charging room	10– 6
Coil and armature winding, mica working, insulating processes	20–12
Elevator—freight and passenger	8– 5
Engraving	100–25
Forge shops and welding	10– 6
Foundries:	
Charging floor, tumbling, cleaning, pouring and shaking out	8– 5
Rough molding and core making	10– 6
Fine molding and core making	15–10
Garage—Automobiles:	
Storage—dead	3– 2
live	8– 5
Repair department and washing	15–10
Glass Works:	
Mix and furnace rooms, pressing and lehr, glass-blowing machines	10– 6
Grinding, cutting glass to size, silvering	12– 8
Fine grinding, polishing, beveling, inspection, etching and decorating	15–10
Glass cutting (cut glass), inspecting fine	50–15
Offices:	
Private and general	
Offices (*Continued*):	
Close work	15–10
No close work	10– 8
Distribution of mail in postoffices	15–10
Drafting room	25–15
Packing:	
Crating	6– 4
Boxing	10– 6
Paint manufacturing	10– 6
Paint Shops:	
Dipping, spraying, firing	8– 5
Rubbing, ordinary hand painting and finishing	12– 8
Fine hand painting and finishing	15–10
Store and Stock Rooms:	
Rough bulky material	3– 2
Medium or fine material requiring care	8– 5
Structural steel fabrication	10– 6
Sugar grading	25–15
Testing:	
Rough	8– 5
Fine	15–10
Extra fine instruments, scales, etc.	100–25
Textile Mills:	
Cotton	
Opening and lapping, carding, drawing-frame, roving, dyeing	8– 5
Spooling, spinning, drawing-in, warping, weaving, quilling, inspecting, knitting, slashing (over beam end)	12– 8
Silk	
Winding, throwing, dyeing	12– 8
Quilling, warping, weaving and finishing—	
Light goods	15–10
Dark goods	20–15
Woolen	
Carding, picking, washing and combing	6– 4
Twisting and dyeing	10– 6
Drawing-in, warping—	
Light goods	10– 6
Dark goods	15–10
Weaving	
Light goods	12– 8
Dark goods	20–12
Knitting machines	15–10
Toilet and wash rooms	6– 4
Upholstering:	
Automobile, coach, and furniture	15–10
Warehouse	3– 2

---

* By permission of the Illuminating Engineering Society; from "Code of Lighting" (American Standard, 1930).

The **intensities,** expressed in **foot-candles,** found by experience to be suitable for various applications are given in Table 13. These values should

Table 14. Glare Classification[*]

Grade	Standard
A	10-watt tungsten-filament lamp in 6-in. frosted ball globe
B	15-watt tungsten-filament lamp in 6-in. frosted ball globe
C	25-watt tungsten-filament lamp in 6-in. frosted ball globe
D	40-watt tungsten-filament lamp in 6-in. frosted ball globe
E	50-watt tungsten-filament lamp in 6-in. frosted ball globe
F	60-watt tungsten-filament lamp in 6-in. frosted ball globe
G	100-watt tungsten-filament lamp in 6-in. frosted ball globe
H	150-watt tungsten-filament lamp in 6-in. frosted ball globe
I	300-watt frosted lamp
J	500-watt frosted lamp
K	1000-watt frosted lamp

[*] By permission of the Illuminating Engineering Society.

Table 15. Grades of Light-source Glare Which Should Not Be Exceeded for Good Conditions of Vision[*]

Height of light source above floor, ft.	Space or work to be lighted[†]			
	Roadways and yard thoroughfares	Storage spaces	Ordinary manufacturing operations	Offices and drafting work and certain manufacturing operations[‡]
6.5 or less	..	*D*	*C*	*A*
6.5–7.5	..	*D*	*C*	*A*
7.5–9	*F*	*E*	*D*	*C*
9–11	*G*	*G*	*E*	*D*
11–13	*H*	*G*	*G*	*E*
13–16	*H*	*H*	*H*	*F*
16–20	*I*	*I*	*I*	*G*
20 and up	*J*	*J*	*J*	*H*

[*] By permission of the Illuminating Engineering Society, from "Code of Lighting," 1930.

[†] Where backgrounds are very dark in tone, a light source one grade softer than above is recommended for interiors.

[‡] Those operations in which workers are seated facing in one direction for long periods of time.

Table 16. Mounting Heights of Lighting Units[*]

Direct lighting units				Semi-indirect and indirect lighting	
Actual spacing between units, ft.	Distance of units from floor, ft., not less than ft.	Desirable mounting height in industrial interiors	Desirable mounting height in commercial interiors	Actual spacing between units, units, ft.	Recommended suspension length (top of bowl to ceiling), ft.
7	8	12 ft. above floor if possible—to avoid glare, and still be within reach from stepladder for cleaning	The actual hanging height should be governed largely by general appearance, but particularly in offices and drafting rooms, the minimum values shown should not be violated	7	1–3
8	8½			8	1–3
9	9			9	1–3
10	10			10	1½–3
11	10½			11	2–3
12	11			12	2–3
14	12½	Where units are to be mounted much more than 12 ft. it is usually desirable to mount the units at ceiling or on roof trusses		14	2½–4
16	14			16	3–4
18	15			18	3–4
20	16			20	4–5
22	18			22	4–5
24	20			24	4–6
26	21			26	4–6
28	22			28	5–7
30	24			30	5–7

Note. Based upon the assumption that the plane of work is 30 in. above the floor.

[*] Tables 15, 16, and 17 reprinted by permission from *Westinghouse Lamp Co., Bull.* E-108.

Table 17. Room Index for Narrow and Average Rooms

For indirect lighting, use ceiling height		Feet						
		9 and 9½	10 to 11½	12 to 13½	14 to 16½	17 to 20	21 to 24	25 to 30
For direct lighting, use mounting height		Feet						
		7 and 7½	8 and 8½	9 and 9½	10 to 11½	12 to 13½	14 to 16½	17 to 20
Room width, ft.	Room length, ft.	Room index						
9 (8½–9½)	8–10	1.0	0.8	0.6	0.6			
	10–14	1.0	0.8	0.8	0.6			
	14–20	1.2	1.0	0.8	0.6	0.6		
	20–30	1.2	1.2	1.0	0.8	0.6	0.6	
	30–42	1.5	1.2	1.0	0.8	0.6	0.6	0.6
	42–up	2.0	1.5	1.2	1.0	0.8	0.6	0.6
10 (9½–10½)	10–14	1.2	1.0	0.8	0.6	0.6		
	14–20	1.2	1.0	0.8	0.6	0.6	0.6	
	20–30	1.5	1.2	1.0	0.8	0.6	0.6	
	30–42	1.5	1.2	1.2	1.0	0.8	0.6	0.6
	42–60	2.0	1.5	1.2	1.0	0.8	0.6	0.6
	60–up	2.0	1.5	1.5	1.0	1.0	0.8	0.6
12 (11–12½)	10–14	1.2	1.0	0.8	0.8	0.6	0.6	
	14–20	1.5	1.2	1.0	0.8	0.6	0.6	
	20–30	1.5	1.2	1.2	1.0	0.8	0.6	0.6
	30–42	2.0	1.5	1.2	1.0	0.8	0.6	0.6
	42–60	2.0	1.5	1.5	1.2	1.0	0.8	0.6
	60–up	2.0	2.0	1.5	1.2	1.0	0.8	0.6
14 (13–15½)	14–20	1.5	1.2	1.0	1.0	0.8	0.6	0.6
	20–30	2.0	1.5	1.2	1.0	0.8	0.6	0.6
	30–42	2.0	1.5	1.5	1.2	1.0	0.8	0.6
	42–60	2.0	2.0	1.5	1.5	1.0	0.8	0.6
	60–90	2.5	2.0	2.0	1.5	1.2	1.0	0.6
	90–up	2.5	2.0	2.0	1.5	1.5	1.2	0.8
17 (16–18½)	14–20	2.0	1.5	1.2	1.0	0.8	0.6	0.6
	20–30	2.0	1.5	1.5	1.2	1.0	0.8	0.6
	30–42	2.5	2.0	1.5	1.2	1.0	1.0	0.6
	42–60	2.5	2.0	2.0	1.5	1.2	1.2	0.8
	60–110	2.5	2.0	2.0	1.5	1.2	1.2	0.8
	100–up	3.0	2.5	2.0	2.0	1.5	1.2	1.0
20 (19–21½)	20–30	2.5	2.0	1.5	1.2	1.0	0.8	0.6
	30–42	2.5	2.0	2.0	1.5	1.2	1.0	0.8
	42–60	2.5	2.5	2.0	2.0	1.5	1.2	0.8
	60–90	3.0	2.5	2.0	2.0	1.5	1.2	1.0
	90–140	3.0	2.5	2.5	2.0	1.5	1.5	1.0
	140–up	3.0	2.5	2.5	2.0	1.5	1.5	1.0
24 (22–26)	20–30	2.5	2.0	2.0	1.5	1.2	1.0	0.8
	30–42	3.0	2.5	2.0	1.5	1.2	1.2	0.8
	42–60	3.0	2.5	2.5	2.0	1.5	1.2	1.0
	60–90	3.0	2.5	2.5	2.0	1.5	1.5	1.0
	90–140	3.0	3.0	2.5	2.0	2.0	1.5	1.2
	140–up	3.0	3.0	2.5	2.0	2.0	1.5	1.2
30 (27–33)	30–42	3.0	2.5	2.5	2.0	1.5	1.2	1.0
	42–60	3.0	3.0	2.5	2.5	1.5	1.5	1.0
	60–90	4.0	3.0	3.0	2.5	2.0	1.5	1.2
	90–140	4.0	3.0	3.0	2.5	2.0	2.0	1.5
	140–180	4.0	3.0	3.0	2.5	2.0	2.0	1.5
	180–up	4.0	3.0	3.0	2.5	2.0	2.0	1.5
36 (34–39)	30–42	4.0	3.0	2.5	2.0	1.5	1.5	1.0
	42–60	4.0	3.0	3.0	2.5	2.0	1.5	1.2
	60–90	5.0	3.0	3.0	3.0	2.0	2.0	1.5
	90–140	5.0	4.0	3.0	3.0	2.5	2.0	1.5
	140–200	5.0	4.0	3.0	3.0	2.5	2.0	1.5
	200–up	5.0	4.0	3.0	3.0	2.5	2.0	1.5
40 or more	42–60	5.0	4.0	3.0				
	60–90	5.0	4.0	4.0				
	90–140	5.0	4.0	4.0				
	140–200	5.0	5.0	4.0				
	200–up	5.0	5.0	4.0				

Table 18. A Guide to the Selection of Reflect

Lighting unit		Efficiency based upon		Appearance of lighted room	Direct glare	Reflected glare	Shadows	Maintenance
		Illumination on horizontal	Illumination on vertical					
Direct lighting—general industrial reflectors								
1 RLM Dome White bowl lamp 90°–180°, 0% 0°–90°, 66%		*A* Excellent	*B* Good	*B* Good	*B*+ Very good	*B* Good	*B*+ Very good	*A*− Very good
2 Glassteel diffuser Clear lamp 90°–180°, 7% 0°–90°, 60%		*A*− Very good	*B* Good	*A*− Very good	*A*− Very good	*B*+ Very good	*A* Excellent	*B* Good
Store and general								
3 White glass enclosing globe 90°–180°, 35% 0°–90°, 45%		*B*+ Very good	*B*+ Very good	*A* Excellent	*B* Good	*B*+ Very good	*A*− Very good	*B*+ Very good
4 Enclosed semi-indirect cased-glass bottom etched top 90°–180°, 51% 0°–90°, 21%		*C*+ Very fair	*C*+ Very fair	*A* Excellent	*A* Excellent	*A* Excellent	*A* Excellent	*B*− Very fair
5 Open indirect 90°–180°, 80% 0°–90°, 0%		*C*+ Very fair	*C* Fair	*B*+ Very good	*A*+ Excellent	*A*+ Excellent	*A*+ Excellent	*C* Fair

ing Equipment and Coefficients of Utilization

Depreciation factor (D)			Ceiling	Very light (70%)			Fairly light (50%)			Fairly dark (30%)	
			Walls	Fairly light (50%)	Fairly dark (30%)	Very dark (10%)	Fairly light (50%)	Fairly dark (30%)	Very dark (10%)	Fairly dark (30%)	Very dark (10%)
Clean conditions	Average conditions	Dirty conditions	Room index	Coefficients of utilization (U)							
Calculation data—general units											
0.80	0.75	0.65	0.6	0.32	0.28	0.25	0.32	0.28	0.25	0.27	0.25
			0.8	0.40	0.36	0.34	0.39	0.35	0.33	0.35	0.33
			1.0	0.43	0.39	0.37	0.42	0.39	0.37	0.39	0.37
			1.2	0.46	0.43	0.41	0.45	0.43	0.41	0.43	0.41
			1.5	0.48	0.45	0.43	0.47	0.45	0.43	0.45	0.43
			2.0	0.52	0.50	0.48	0.51	0.49	0.47	0.49	0.47
			2.5	0.56	0.54	0.52	0.55	0.53	0.51	0.53	0.51
			3.0	0.57	0.55	0.53	0.56	0.54	0.52	0.54	0.52
			4.0	0.60	0.58	0.56	0.59	0.57	0.55	0.57	0.55
			5.0	0.61	0.59	0.57	0.60	0.58	0.57	0.58	0.56
0.75	0.70	0.60	0.6	0.29	0.25	0.21	0.28	0.24	0.21	0.23	0.21
			0.8	0.36	0.32	0.29	0.35	0.31	0.28	0.31	0.28
			1.0	0.39	0.36	0.33	0.38	0.35	0.33	0.34	0.32
			1.2	0.42	0.39	0.36	0.41	0.38	0.36	0.37	0.35
			1.5	0.45	0.42	0.39	0.43	0.40	0.38	0.39	0.38
			2.0	0.49	0.46	0.43	0.48	0.45	0.43	0.44	0.42
			2.5	0.53	0.50	0.47	0.51	0.49	0.47	0.47	0.46
			3.0	0.54	0.52	0.49	0.52	0.50	0.49	0.49	0.47
			4.0	0.57	0.55	0.53	0.55	0.53	0.51	0.51	0.50
			5.0	0.58	0.56	0.54	0.56	0.54	0.53	0.52	0.51
utility units											
0.80	0.75	0.65	0.6	0.22	0.17	0.14	0.20	0.16	0.13	0.14	0.12
			0.8	0.27	0.22	0.19	0.25	0.21	0.18	0.19	0.17
			1.0	0.31	0.26	0.23	0.28	0.24	0.21	0.22	0.19
			1.2	0.35	0.30	0.26	0.31	0.27	0.24	0.25	0.22
			1.5	0.38	0.33	0.29	0.34	0.30	0.27	0.27	0.24
			2.0	0.42	0.38	0.33	0.38	0.34	0.31	0.31	0.28
			2.5	0.46	0.41	0.37	0.41	0.37	0.34	0.34	0.31
			3.0	0.49	0.45	0.40	0.43	0.39	0.36	0.36	0.33
			4.0	0.53	0.48	0.44	0.47	0.43	0.40	0.38	0.36
			5.0	0.55	0.51	0.47	0.49	0.45	0.42	0.40	0.38
0.75	0.65	....	0.6	0.16	0.12	0.10	0.13	0.10	0.08	0.08	0.07
			0.8	0.20	0.16	0.14	0.17	0.14	0.11	0.11	0.09
			1.0	0.23	0.19	0.17	0.19	0.16	0.14	0.13	0.11
			1.2	0.26	0.22	0.19	0.22	0.18	0.16	0.14	0.13
			1.5	0.29	0.25	0.21	0.24	0.20	0.19	0.16	0.14
			2.0	0.32	0.28	0.25	0.27	0.23	0.21	0.18	0.17
			2.5	0.35	0.31	0.28	0.29	0.26	0.24	0.20	0.19
			3.0	0.38	0.34	0.31	0.31	0.28	0.26	0.22	0.21
			4.0	0.41	0.38	0.35	0.34	0.31	0.29	0.24	0.23
			5.0	0.43	0.39	0.37	0.36	0.33	0.31	0.26	0.24
0.70	0.60	....	0.6	0.15	0.12	0.10	0.11	0.09	0.07	0.05	0.04
			0.8	0.18	0.15	0.13	0.13	0.11	0.09	0.07	0.06
			1.0	0.22	0.19	0.16	0.15	0.13	0.11	0.08	0.07
			1.2	0.25	0.22	0.19	0.18	0.15	0.13	0.09	0.08
			1.5	0.27	0.24	0.21	0.20	0.17	0.15	0.10	0.09
			2.0	0.30	0.27	0.25	0.22	0.19	0.17	0.11	0.10
			2.5	0.34	0.31	0.28	0.24	0.22	0.20	0.13	0.12
			3.0	0.36	0.33	0.30	0.26	0.24	0.22	0.14	0.13
			4.0	0.40	0.37	0.34	0.28	0.26	0.24	0.15	0.14
			5.0	0.42	0.39	0.37	0.30	0.28	0.26	0.17	0.15

be considered minimum, and, for seeing under difficult conditions, higher values should be used. **Glare** is present when the light intensity in the line of vision is extremely high, particularly in comparison with the light intensity on near-by objects. Reflectors and lighting glassware are used to *diffuse the light.* Degrees of light-source glare which should not be exceeded are given in Table 15, in conjunction with the grading given in Table 14. The quantity of light is expressed in lumens. A **lumen** is the amount of light which will give a light intensity of 1 ft.-candle on 1 sq. ft. Hence to obtain an intensity of 10 ft.-candles on an area of 100 sq. ft. would require 1000 lumens projected on the surface.

Artificial illumination is usually obtained from incandescent lamps, arc lamps, or gaseous-discharge lamps. For lighting of highways and large areas, lighting units combining two types of lamps are sometimes used.

The **incandescent lamp** produces light from a heated filament of tungsten (early filaments were of carbon and tantalum) enclosed in a vacuum or an inert gas. Through continuous research the efficiency has been gradually increased from about 1.5 lumens per watt in 1881, to 10 lumens per watt for 25-watt lamps and 20 lumens per watt for 1000-watt lamps (see Table 19).

The **arc light** is the oldest type of electric light; it derives the light from an arc maintained between two electrodes. The electrodes may be of a variety of materials although carbon electrodes have been most common. The d.c. magnetite arc produces a brilliant white light with an efficiency of about 18 lumens per watt between a cathode of magnetic oxide of iron and titanium and an anode of copper. Many arc lights have been superseded by series incandescent lamps; these are as efficient on alternating current as on direct, and the costs of replacing electrodes are eliminated.

Electric-discharge lamps produce luminescence from an arc or discharge in a vacuum or a rarified gas. There are several types of these lights and they are unstable except when provided with a special transformer, or reactor, or resistor to limit the current. Only the mercury-vapor lamp, the neon-type lamp, and the fluorescent lamps will be briefly discussed here.

The **mercury-vapor lamp** is used in many industrial plants because of its high efficiency (14 to 18 lumens per watt) and its freedom from sharp shadows. The high actinic value of this lamp makes it useful for photographic work. In the early mercury-vapor lamps the arc was struck by tilting the tube. The usual method is to strike the arc by means of a high voltage obtained by breaking an inductive circuit.

The **neon lamps,** used extensively in signs, require relatively high voltage varying with the length of tubing. Because of its color it is not desirable for ordinary illumination.

The **fluorescent lamps** obtain a high lumen output by using salts deposited on the inside of the tube, called phosphors, which glow when activated by ultra-violet rays (see Table 19). Various colors are obtained by using different combinations of phosphors. The arc or discharge is struck between two heated tungsten filaments, one at each end of the tube, which have been treated to produce high electronic emission. The auxiliary control equipment limits the current to a predetermined value and is not interchangeable with that for lamps of a different size. The losses in the auxiliaries are 20 to 30 per cent of the nominal watts of the lamp with which they are designed to operate. The power factor of many of these lamps with auxiliaries is about 50 to 60 per cent; however, this can be improved with condensers. Newer control devices are reported to give an over-all power factor of nearly unity. The lower temperature, higher efficiency, and available color qualities promise a large field of application. A useful life of nearly 2000 hr. may be expected. For further discussion see Inman and Thayer, *Trans. Am. Inst. Elec. Engrs.*, **57**, pp. 723–726 (1938).

In planning industrial lighting the illumination sources should be uniformly spaced, and the distance between units should approximate the height of the ceiling. A convenient way to *locate outlets* is to divide the room into equal squares or rectangles locating the outlets at the center of each space. The distance between outlet and wall is thus one-half the distance between outlets. The appearance of the resulting fixtures and the location of machines must be considered also in locating outlets. The recommended mounting height of units is given in Table 16. For high lighting intensity and for contrast, local lighting units supplementing the general illumination are necessary.

Table 19. Approximate Lumen Output of Multiple Mazda Lamps

Incandescent Lamps*

110–120 volt standard		110–120 volt daylight		220–240 volt standard	
Size, watts	Lumen output	Size, watts	Lumen output	Size, watts	Lumen output
15	140	100	990	100	1,100
25	260	200	2,210	200	2,920
40	440	300	3,590	300	4,560
60	760	500	6,370	500	8,350
75	1,065	...	......	750	13,125
100	1,530	...	......	1000	19,000
150	2,535	...	......	....	......
200	3,400				
300	5,520				
500	9,800				
750	14,550				
1000	20,700				
1500	33,000				

Fluorescent Lamps†

	Nominal watts (add auxiliary watts for total)			
	15	20	30	40
	Lumen output			
Daylight	450	660	1110	1600
White	525	760	1320	1800
Blue	315	460	780	
Green	900	1300	2250	
Pink	300	440	750	
Gold	375	540	930	
Red	45	60	120	

* From *Gen. Elec. Bull.* LD-6A, October, 1936.

† From *Gen. Elec. Bull.* LD-1 (Sup.), June, 1939.

The **size of lamp** for each unit is selected to give the necessary light, expressed in lumens. *Additional lumens* at the lamp sources are required, because all of the light falling on the walls and ceiling is not reflected; with dark surfaces a considerable amount of light is absorbed. This is taken into account by a **coefficient of utilization** based on a room index (Table 17) for the dimensions of the room, the type of unit, and the color of the walls. The lumens from a unit are decreased by the light absorbed by the lighting unit itself and by dust accumulating on the unit. These factors are considered in calculating the lumens required as follows:

$$L = \frac{AF}{(NUD)} = \frac{aF}{UD}$$

where L is the lumens per outlet, A is the floor area in square feet, a is the floor area per outlets in square feet, N is the number of outlets, F is the desired foot-candles based on Table 13, U is the coefficient of utilization from Table 17, and D is the expected average efficiency of the lighting unit from Table 17. The size of lamp is selected according to the required lumens from Table 19.

Wiring should be installed to provide not only for modern lighting but for possible future uses of the structure. It is recommended that the wiring provide for a capacity of about 4 watts per square foot of floor area for offices, drafting rooms, and factories, 3 watts per square foot for stores and school rooms, 2 watts per square foot for neighborhood stores and storage areas in factories, and 1 watt per square foot for storage areas in garages and unimportant basements. The wire size selected for each branch should allow not more than 2 volts drop to the farthest outlet. The "Code of Lighting" (American Standard, 1930) recommends for branch circuits a minimum wire size of No. 12; and for runs from panel board to first outlet of from 50 to 100 ft., No. 10 wire is the smallest that should be used; runs exceeding 100 ft. to first outlet should be avoided.

Additional data on illumination standards and calculations may be obtained from "Recommended Practice of Industrial Lighting" prepared by *Illuminating Engineering Society*, 1939 and *Gen. Elec. Bull.* LD-6A.

ELECTRIC HEATING

Heating is produced (*a*) when current is forced through the resistance of a conductor and (*b*) in some cases by an arc. The first is by far the more common, giving light in incandescent lamps and heat in a variety of devices. The induction-type heaters are a modification of (*a*), utilizing the material to be heated as the resistance.

The **advantages of electric heat** are the convenience of the control, the facility in producing the heat where desired, and the elimination of combustion and its products. This permits heating under unusual conditions as with hydrogen surrounding the object being heated.

The **rate** at which **energy** is converted into heat in a resistance is equal to I^2R. The **ultimate temperature** of a device is that which will cause the heat energy to be dissipated at the same rate as it is produced. Heating devices utilize conductors of much higher specific resistance than copper and which will not deteriorate rapidly at the required temperature. Many of these resistance materials are alloys which have a small temperature coefficient of resistance (Table 2).

The **efficiency** of a heating device may be expressed as the ratio of the energy absorbed by the material being heated to the total energy consumed. This efficiency is relatively high in electric heaters, as ventilation to remove products of combustion is unnecessary. Further, the heater element can usually be placed in close proximity to the material being heated, and occasionally it is possible to use either the material itself or its container as the heater element. For a discussion of the latter method see Carleton, *Ind. Eng. Chem.*, **21**, 525 (1929).

Insulation for electric conductors is usually heat insulation also. Consequently, solid insulation about the conductors of heating ovens is used sparingly and in many cases is replaced by air.

Resistance furnaces and ovens are constructed in a variety of forms to be most useful and efficient for the work to be performed. The more common forms include (*a*) the box type, with single-door opening and with modifications, such as the car type where the material is carried on trucks or cars with and without heaters in the cars, the pusher type where one piece (or container) is used to push the next, and the conveyor type for continuous process; (*b*) the rotary hearth furnace used for continuous heating; and (*c*) the crucible or pot furnaces with resistors surrounding the pot or with the heating elements submerged in the bath itself.

The **induction-type furnace** utilizes the material to be heated as the secondary of a transformer; the primary coil is connected to the supply. Certain furnaces provide a magnetic structure for the flux path interlinking the primary and secondary. Other furnaces apply a high-frequency alternating current to the primary and use a non-magnetic flux path (see Electric Furnaces under Electrochemistry, Sec. 25, p. 2801).

Box-type ovens for temperatures up to 600° and even 700°F. are listed by electrical manufacturers for connected loads from about 1.5 up to nearly 100 kw. These ovens are suitable for drying, baking, evaporating, japanning, and low-temperature heat treatment. **Box-type furnaces** for temperatures up to 1850°F. are listed with connected loads approximately the same as for the ovens. These furnaces are suitable for hardening, annealing, tempering, and carburizing.

Commercial **pot-type furnaces** suitable for melting babbitt and other soft metals have connected loads from 4.5 to 22 kw. holding from 150 to 1500 lb. of metal. To expedite the handling of the metal the furnace may be equipped for bottom pouring or for tilting. Other furnaces of the same general type use a lead, salt, or cyanide bath for hardening or tempering tools or machine parts. In large sizes as for continuous hardening or drawing of wire or strip, or for heating large parts, standard furnaces have capacities up to 15,000 cu. in. and have a connected load of 75 kw. Such a furnace with lead bath will average in a 9-hr. day a production of 1200 lb. steel per hour raised to 1200°F. with 800 kw.-hr. per day (see Catalogue 250, Westinghouse Electric and Mfg. Co.). Certain pot-type furnaces for heat treatment use a conducting salt: one electrode is immersed in the salt; the containing vessel is used for the other. An arc is produced to start the melting of the salt which is non-conducting in its solid state [see Hall, *Trans. Am. Soc. for Steel Treating*, p. 399 (1929)].

25

Many resistor-type furnaces utilize **heating units** which are easily replaced; these are connected by steel busbars as copper oxidizes rapidly at furnace temperature. In designing an oven or other heating device, such units may be applied. Certain units use nickel-chromium ribbon wound on porcelain insulators. Another form is a flat heater element protected by a steel casing; still others are designed for immersion in the liquid to be heated. The cartridge-type heater is used for local heating in a portion of a machine, the cartridge fitting closely in a hole drilled in the machine, so that the heat may be conducted away.

Table 20. Steel-sheathed Strip Heaters*

(Dimensions $\frac{3}{8}$ in. thick by $1\frac{1}{2}$ in. wide)

Watts	Volts	Maximum length, in.
500	115	24
350	115	18
250	115	12
150	115	7

Note. Heaters for 230 volts have the same dimensions and price. These may be connected in series for 440-volt circuits.

* Data from General Electric Supply Catalogue.

In the **design**† of a **heating element,** materials must be selected which will withstand the operating temperature of the element without deteriora-

---

† By Hood Worthington.

tion. Nickel-chromium alloys are capable of withstanding an operating temperature of 1150°C., and certain less expensive alloys operate satisfactorily at lower temperatures. Figure 44 gives the allowable heat flux at the surface of nichrome IV wire or ribbon as a function of the temperature of the material being heated and the method of applying the heating element. The empirical design curve for exposed radiating strips or rods is given by Fleischmann [*Trans. Am. Inst. Elec. Engrs.*, **48,** 1196 (1929)] as useful because it gives long-lived elements even under conditions of close spacing and double banking.

Table 21. Electric Heater Units*

Kw.	Volts	Length, in.	Width, in.	Height, in.
1.75	110	21½	6¼	10
2.5	110	21½	6¼	10
3.5	220	36½	6¼	10
5.0	220	36½	6¼	10

* Westinghouse catalogue 247.

Table 22. Electric Cartridge Heaters*

Watts	Volts	Outside diam., in.	Length, in.	Terminal connection
75	110	0.496	3²³⁄₃₂	Monel wire
100	110	.496	4⁷⁄₃₂	Monel wire
150	110	.621	2⅝	Monel wire
250	110	.621	4⅛	Monel wire
200	110	.746	4⁹⁄₃₂	Monel wire
150	110	.933	3⅝	Brass terminal
250	110	.933	5⅛	Brass terminal
400	110	.933	5⅝	Brass terminal
300	110	1.306	4¹³⁄₁₆	Brass terminal
600	110	1.306	5⁵⁄₁₆	Brass terminal

* Westinghouse catalogue 247.

Woodson's curve for confined radiant coils [*J. Am. Inst. Elec. Engrs.*, **44,** 1354 (1925)] checks closely with Driver-Harris Company's recommendation for range units and radiant heaters made of coiled wire.

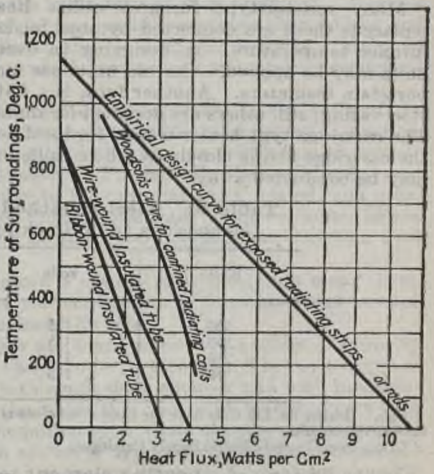


FIG. 44.—Permissible heat flux from nichrome IV.

The two curves marked **wire-wound** and **ribbon-wound insulated tube** give the allowable heat flux at the surface of a wire wound with the turns one diameter apart and of a ribbon wound in such a way that the surface of the ribbon is twice the surface on which it is wound, respectively. Both windings are assumed to be placed over ⅛-in. asbestos insulation and to be delivering heat to a metal tube having the temperature given on the ordinate. If the ribbon or wire is differently spaced, or the thickness or thermal conductivity of the insulation is changed, the allowable heat flux will be altered. The factor of safety enters the last two curves only in that the maximum temperature which the winding can stand is taken as 900°C. instead of 1150°C.

From these curves it will be apparent that the method of embedding a winding in an insulating medium should be avoided if it is possible to use the method of supporting the winding so that it is free to give up its heat by radiation. However, the use of a winding over an insulated tube has the advantage of simplicity of construction.

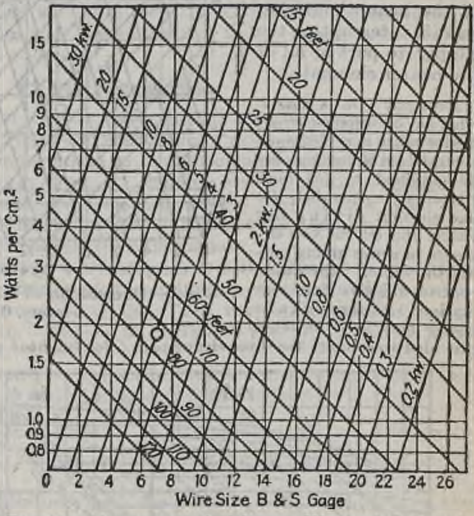


Fig. 45.—Dimensions of nichrome IV wire for 110-volt heaters, 0.2 to 30 kw.

The use of Figs. 45 and 46 will be made plain by an example. The position of the circle in Fig. 45 indicates that if the winding is such that it permits a heat flux of 1.9 watts per square centimeter, a 5-kw. heater operating on 110 volts may be constructed from 73 ft. of No. 7 B. & S. gage nichrome IV **wire**. For a 6-kw. heater on 220 volts (Fig. 46), 120 ft. of No. 10 wire should be used. (Note the slightly higher heat flux used to get an even wire size.)

When **ribbon** is used there are more possibilities. Suppose that the allowable heat flux in a 20-kw. heater to be used on a 220-volt line is 3.7 watts per square centimeter. Moving horizontally at 3.7 watts per square centimeter across the lower left-hand field of Fig. 47 to the line marked 20 kw. at 220 volts, and then vertically to the upper left-hand field, one finds that there are four sizes of ribbon that may be used. Then moving horizontally to intersect the 3.7 watts per square centimeter line in the right-hand field and reading from the lengths at 220 volts, one finds the following solutions: 37 ft. of 1 in. × 32 gage; 51 ft. of ¾ in. × 26 gage; 70 ft. of ½ in. × 20 gage; or 86 ft. of ⅜ in. × 16 gage.

Because of their superior resistance to oxidation, the ½ in. × 20 gage or the ⅜ in. × 16 gage are preferable. If, however, the ribbon is too thick it loses its advantage in ease of winding and uniform covering of the winding area, and wire may be used with equal satisfaction.

For certain heaters the **control** may be manual; for others the temperatures are controlled automatically by thermostats or pyrometers. The

energy delivered to a heating device may be varied by (1) adjusting the applied voltage as from transformer taps, (2) changing the connections so that the voltage applied to the heating elements is varied, (3) rheostatic control in series with the element, (4) resistance change of the heater due to the temperature coefficient of the heater material, and (5) interrupting the supply to part or all of the heater elements.

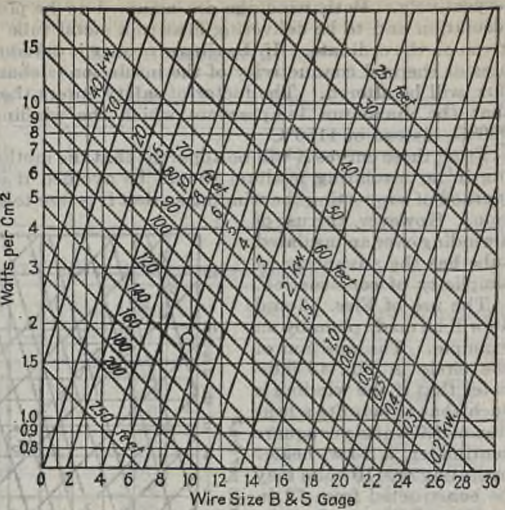


FIG. 46.—Dimensions of nichrome IV wire for 220-volt heaters, 0.2 to 40 kw.

Example. On a single-phase or direct-current heater with two elements, three values of heating may be obtained by (*a*) both elements in parallel, (*b*) one element giving half value, and (*c*) the two elements in series giving one-fourth value. Rheostatic control will give finer regulation. A three-phase heater may have switches provided to connect the three elements in Δ for full heat or in Y for one-third value.

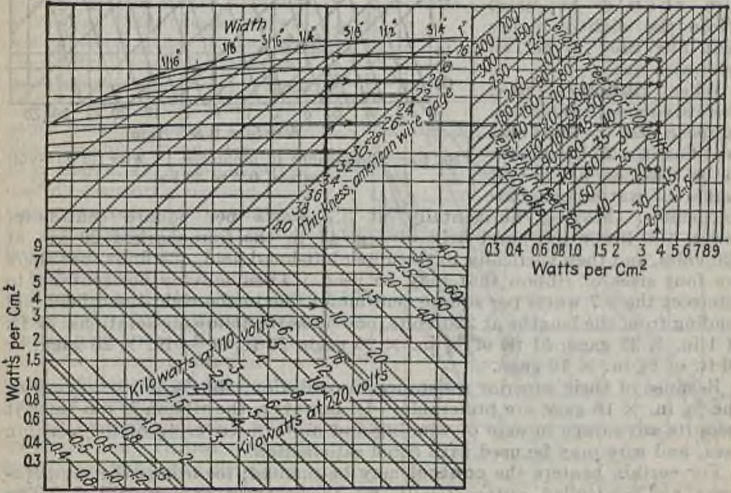


FIG. 47.—Dimensions of nichrome IV ribbon for 110- and 220-volt heaters, 0.4 to 60 kw.

Thermostats indicate (or record) the furnace temperature and, by means of two adjustable contacts for the "high" and "low" temperature limits, close the control circuits at high for shutting off the supply to the heaters and at low for turning on the power to the heating elements. The thermostat is actuated by heating of a bulb containing mercury, a liquid, or a gas, causing a pressure change, which is transmitted through a tube to the indicating and controlling mechanism. The maximum temperature to be controlled by thermostats is 1000°F.

For temperature above 1000°F., **pyrometers** are used. The pyrometer is actuated by the voltage from a thermocouple placed in the furnace or oven. The pyrometer is a sensitive instrument and should be mounted in a location free from vibration. The calibration is for a particular type of thermocouple and may vary with length of wires between the couple and the instrument. By means of two adjustable contacts the power to the heater is controlled similarly to the thermostat control.

TRANSFORMERS

A transformer consists of two or more *electric circuits* interlinked by a *magnetic circuit.* The electric circuits are usually insulated from each other but may be electrically connected as in the autotransformer. This device permits the transfer of energy from a circuit of one alternating voltage to that of another voltage and serves to insulate the two circuits from one another. The primary winding is the one connected to the supply; the winding connected to the receiver circuit is the secondary.

The **magnetizing current** of a transformer is just sufficient to produce a flux, such that the rate of change of flux induces a voltage in the primary practically equal to the impressed voltage. The vector difference is due to the IR drop and is negligible. This small current is the current which flows when the secondary is open.

The **induced e.m.f.** equation is similar to that in a generator, though this is due to stationary windings and to changing magnitude of flux.

$$E = 4.44 f \phi N \times 10^{-8}$$

where f is the frequency, ϕ is the maximum instantaneous value of flux, and N is the number of turns.

Since the magnetic flux interlinks both the primary and the secondary, the *voltage* induced in each winding *is proportional to the number of turns.* Due to a slight leakage, *i.e.*, flux which does not interlink both windings, this relation may not be exact for all loads.

As load is connected to the secondary and current flows, it tends to modify the interlinking flux, hence the primary current increases to maintain the flux.

The *ampere-turns in the primary equal the ampere-turns in the secondary* except for the magnetizing current:

$$\frac{E_p}{E_s} = \frac{N_p}{N_s}; \qquad I_pN_p = I_sN_s; \qquad \frac{I_s}{I_p} = \frac{N_p}{N_s}$$

therefore,

$$E_pI_p = E_sI_s$$

These relations of input and output hold as to volts, amperes, power, and power factor, except as modified slightly by the losses and by the magnetizing current. Subscripts refer to primary or secondary.

The **losses** in a transformer consist of *iron losses* (hysteresis and eddy-current losses) and *copper losses* (primary and secondary). These losses are small so that the full-load efficiency is about 98 per cent and is higher for large transformers. The iron losses of transformers which are continuously on the supply line will reduce the daily average efficiency.

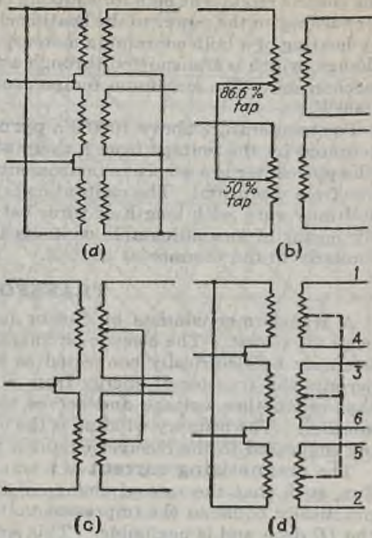


Fig. 48.—Transformer connections in polyphase circuits. (*a*) three-phase delta to three-phase Y; (*b*) three-phase to two-phase (Scott Connection); (*c*) two-phase three-wire to two-phase five-wire; (*d*) three-phase delta to six-phase diametrical.

In **polyphase circuits,** transformers are used to change from one system to another as well as to change the voltage. Any polyphase system may be secured from any other polyphase system with transformers provided with suitable windings. A few of the more common *transformer connections* are shown diagrammatically in Fig. 48. The desired voltage for both systems may be obtained from transformers with the proper voltage ratio. A single-phase load cannot be balanced on a polyphase supply by any combination of transformers; neither can a polyphase load be taken from a single-phase supply without energy-storage devices.

Autotransformers (Fig. 49) have a portion of the winding in common for the primary and secondary. In this portion of the winding, the current is the difference between the primary and secondary currents resulting in a higher utilization of materials, particularly where the transformer ratio is near unity. For example, with a ratio 3: 4, the capacity as an autotransformer would be four times the capacity used as an ordinary transformer. Since the primary and secondary are interconnected the autotransformer does not insulate one section of the circuit from the other.

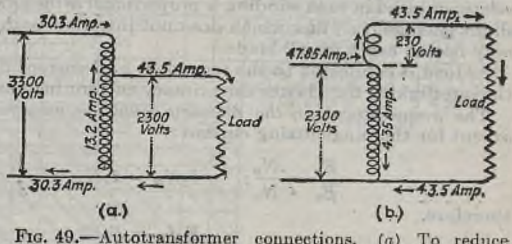


Fig. 49.—Autotransformer connections. (*a*) To reduce voltage at load; (*b*) to increase voltage at load. (*Marks, "Mechanical Engineers' Handbook."*)

The **induction regulator** is an autotransformer or booster transformer with a variable ratio. In the *single-phase regulator*, the flux common to the primary and secondary is varied by turning the primary with respect to the secondary. In the *three-phase regulator*, a constant voltage is added vectorially; the angle is varied by the position of the rotor so that the resultant voltage is regulated to the value desired. Regulators are commonly used to control distribution voltages; by means of the line-drop compensators the point of regulation may be at a distance from the regulator.

The **constant current transformer** utilizes the repulsion between the primary and secondary coils to separate the coils when the current starts to increase. The greater separation increases the leakage flux, reducing the voltage until the current is the same. If the current starts to decrease, the coils come closer together so that the voltage increases and the current also. By properly counterbalancing the moving coil, the current can be maintained practically constant.

Instrument transformers are used to reduce voltage (potential transformers) and current (current transformers) for applying to instruments and relays. This permits arrangement of main leads independent of the switchboard location by running small wires to the instruments; it permits standardizing on 5-amp. 110-volt instruments with proper scales according to the ratio of the transformers, and it permits insulating the instruments from high-potential circuits.

Table 23. Approximate Cost of Small Transformers—2300/115–230 Volts

Kva.	$ per kva., single phase	$ per kva., three phase
5	12.6	19.6
10	10.0	14.4
50	6.3	8.5

GENERATORS

Direct-current Generators. Direct-current generators consist of a stationary field and yoke and a rotating armature with commutator. The **armature** is laminated to reduce the eddy-current losses and has slots for the armature windings to reduce the reluctance of the magnetic path. Inasmuch as the conductors pass alternately under north and south poles the e.m.f. induced is alternating. The armature conductors are connected through a switching device, the commutator and brushes, to the external circuit so that the e.m.f. delivered is unidirectional.

The armature winding is continuous in order that the commutator may function, hence there are two or more paths for current through the armature:

$$E = \left(\frac{Z}{\text{paths}}\right)\phi p\left(\frac{\text{r.p.m.}}{60}\right) \times 10^{-8}$$

where Z is the total number of conductors on armature; paths refer to number of parallel paths through armature; ϕ is total flux per pole actually entering the armature (due to leakage this is less than the total flux per pole); p is the number of poles; and r.p.m. is the speed of the armature in revolutions per minute. For a given machine this may be written $E = k'\phi$ r.p.m.

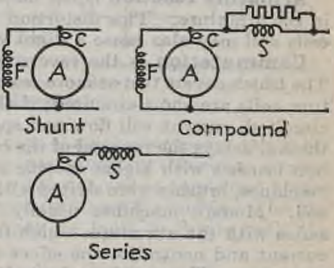


FIG. 50.—Direct-current generator connections. A, armature; F, shunt field; C, commutating field; S, series field. Note that diverter resistance is shown in parallel with series field of compound-wound generator.

The speed of the generator is usually fixed by the driving engine or motor; and the number of conductors, paths, and poles by the designer. Hence any *adjustment of e.m.f.* must be by field control; this may be by a hand-controlled rheostat or by automatic devices.

Types of generators are designated by the field connections. When the field circuit is connected across the armature, it is a **shunt** machine; if connected in series with the armature, it is a **series** machine; if there are both shunt and series field coils, it becomes a **compound** machine; the field may be connected to some other source making it a **separately excited** machine.

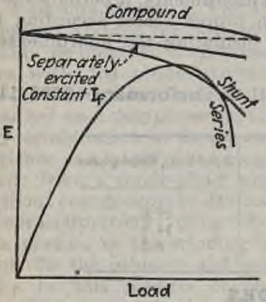


Fig. 51.—Voltage characteristics of direct-current generators.

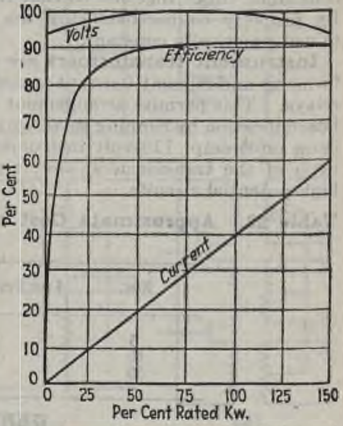


Fig. 52.—Typical characteristics of compound-wound generator.

Armature reaction is the distortion of the main field due to the current in the armature. This distortion tends to induce e.m.fs. in the commutated coils and may also cause a slight weakening of the field.

Commutation is the reversing of current in the coil under the brush. The brush covers two or more commutator segments, hence one or more armature coils are short-circuited; if there is an e.m.f. induced while it is short-circuited, current will flow and sparking will result. The self-inductance of the coil delays the reversal of the current and this also causes sparking. Carbon brushes with higher specific resistance decrease the sparking. In older machines, brushes were shifted with load to change the flux in the commutated coil. Modern machines usually have **commutating poles** connected in series with the armature, which furnish sufficient flux to aid the reversal of current and neutralize the effect of armature reaction at the point of commutation. The commutating pole makes it unnecessary to shift brushes with load, but the proper position for the brushes is critical and the manufacturer's setting should not be disturbed.

Regulation of direct-current generators, *i.e.*, change of voltage with load, depends on the type of generator (Fig. 51). The voltage of the separately excited generator decreases with load due to *IR* drop in the armature circuit and the effect of armature reaction. The shunt machine has a further reduction in voltage with load because the lower voltage is impressed on the field. In the compound-wound machine, the series-field coil increases the m.m.f. with load. This may be adjusted by the number of turns on the series field so that the full-load voltage is just equal to the no-load voltage (flat compounded) or so that it is more or less as desired. On commercial generators, excess turns are usually put in the series field and then diverters (resistances) are placed in parallel so that enough of the current is diverted to give the desired characteristic. The series generator is seldom used.

Paralleling of generators is frequently desirable in order that more than one generator may be used to supply a load. **Shunt machines** may be paralleled with little difficulty, care being taken that the same terminal voltages exist and the same polarities are connected together. For proper distribution of load, all generators operating in parallel should have the same regulation. **Compound-wound generators** must be provided with an equalizer connected in order that two or more machines be stable when paralleled. The *equalizer* is a low-resistance conductor connecting the series fields in parallel. With the equalizer bus compound machines may be adjusted to divide the loads properly.

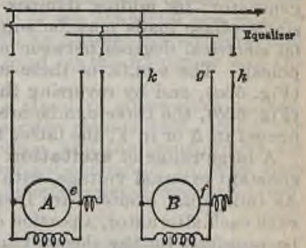


Fig. 53.—Connections of compound-wound generators for parallel operation. (*"Standard Handbook for Electrical Engineers."*)

Three-wire generators are used to obtain the advantage of double the load voltage for distribution. Common voltages are 115–230 volts. Motor loads are usually connected at the higher voltage. The equivalent of the three-wire generator may be obtained by means of a balancer motor-generator set on a system of the higher voltage. The most common three-wire generator is of the Dobrowolsky type, in which a balance coil is connected across the armature; the mid-point of the balance coil furnishes the mid-point for the direct-current system. The balance coil may be built to rotate with the armature or may be external.

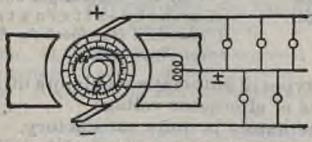


Fig. 54.—Three-wire direct-current generator. (*Marks, "Mechanical Engineers' Handbook."*)

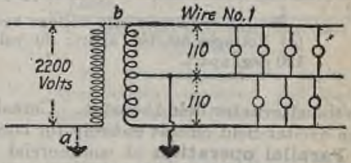


Fig. 55.—Three-wire single-phase from transformer. (*Marks, "Mechanical Engineers' Handbook."*)

Homopolar generators are commutatorless direct-current generators so arranged that the conductors always cut the flux in the same direction. The connection to the conductor is by slip rings, two for each conductor. The machine is not standard, is of low voltage, heavy, and expensive.

Alternating-current Generators. Alternating-current generators are built with rotating fields and stationary armatures, permitting a very rugged construction; the only moving contacts are for the field current at low voltage. Commercial alternators generate at 2300 volts in small sizes, 13,500 for medium and large sizes, and a few very large units are built to generate 22,000 volts. On account of the large capacity and high voltage the dissipation of the heat due to the losses is a major problem requiring enormous amounts of air for cooling.

Three-phase alternators are supplied for the usual installation. Two phase is frequently obtained by means of transformers from a three-phase generator; by adding damper windings on the field-pole structure, heavy single-phase loads may be supplied. Three groups of windings are wound 60 electrical degrees between centers (360 electrical degrees to each pair of poles). The e.m.fs. of these windings would be 60 electrical degrees apart (Fig. 56*a*), and by reversing the middle winding and consequently its e.m.f. (Fig. 56*b*), the three e.m.fs. are 120° apart. The windings can then be connected in Δ or in *Y*, the latter being more common.

A large range of **excitation** (field-circuit current) is necessary to provide constant external voltage, with variation in load and power factor (Fig. 57). As individual exciters are frequently furnished with each alternator, a portion of this control can be provided by the rheostat in the exciter field circuit, reducing the I^2R losses otherwise present in the alternator field rheostat. Certain types of automatic regulators utilize the exciter field circuit entirely for control of alternator voltage.

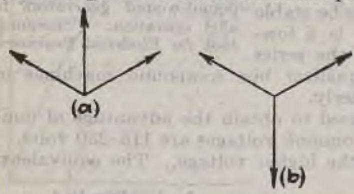


FIG. 56.—Three phase voltage vectors. (*a*) Voltages 60 deg. apart; (*b*) voltages 120 deg. apart.

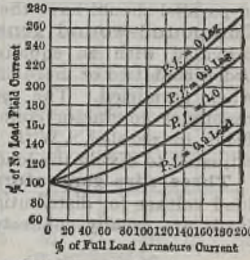


FIG. 57.—Excitation curves of a turbo-alternator. (*"Standard Handbook for Electrical Engineers."*)

Parallel operation of commercial alternators is quite satisfactory. To connect an alternator to an energized bus, it is necessary to synchronize by hand or automatically. *Synchronizing* is adjusting the time of the voltage wave, by changing the speed of the incoming alternator, to obtain the correct frequency and the correct instantaneous polarity; lamps may be used, although the synchroscope is the instrument usually used to indicate this. In addition it is necessary to have correct voltage and phase rotation.

Load division of paralleled alternators is dependent on the speed (governor) adjustments of the prime movers. When it is desired to increase the output of an alternator, the throttle is opened more, tending to increase its speed, but actually changing the phase position of its generated e.m.f. with respect to that of the other alternators. The frequency of all alternators in parallel must be the same, which thus fixes their speed.

Excitation change of paralleled alternators tends to change their **reactive current.** Since the total reactive current (at a given voltage) is fixed by the load, if the excitation of one alternator is increased, it furnishes a larger proportion of the total reactive current.

An induction motor may be driven slightly above synchronous speed; it will then deliver energy to the supply line. This **induction generator** receives its excitation from the supply line, hence ceases to generate if the supply is disconnected. The power factor of the generator is fixed by the voltage, load, and design of the machine; it cannot be adjusted.

Synchronous or **rotary converters** (Figs. 58 and 59) are used to convert alternating current to direct current. Alternating current is applied to, and direct current taken from, a common armature winding rotating within a salient pole structure. The converter operates as a synchronous motor and is started in like manner. The power factor may be adjusted by field change as in the motor, but the efficiency and output of the converter are reduced at decreased power factor. The characteristics as a direct-current generator are limited, since the speed and voltage are fixed by the frequency and the alternating voltage.

To obtain a standard direct voltage from a standard alternating voltage, it is necessary to use transformers as the *conversion ratio is fixed*. Since the efficiency and capacity increase with more phase, and the transformers are in use for proper voltage, six-phase converters are common. In large sizes the converter has the advantage over a motor-generator set of *higher efficiency*, *lower first cost*, and *smaller space*.

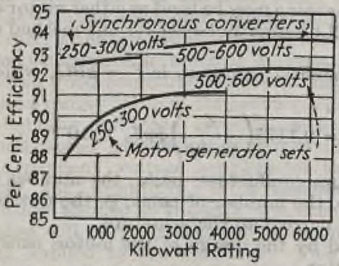


FIG. 58.—Efficiency of synchronous converters and motor-generator sets. (*Chem. Met. Eng.*, *November*, 1928.)

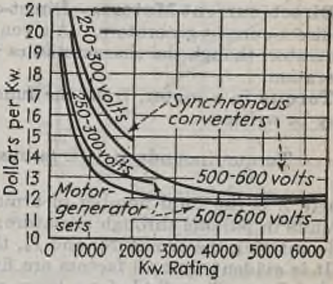


FIG. 59.—Approximate prices of synchronous converters and motor-generator sets. (*Chem. Met. Eng.*, *November*, 1928.)

The **disadvantage** of the synchronous converter is that the direct voltage is directly proportional to the alternating voltage. This difficulty disqualifies the synchronous converter in favor of the motor-generator set for many applications. If the output voltage must be adjusted, it may be (*a*) by field control, either shunt or by compounding; (*b*) by transformer taps or induction regulators; and (*c*) by a booster alternator on the shaft with the converter. These schemes decrease the efficiency and increase the cost of the installation, and in the case of (*a*) the possible variation is very small being caused by change of power factor.

The converter may be operated *inverted*, receiving direct current and delivering alternating current. If driven mechanically, it may deliver direct or alternating current, or both.

The **mercury arc rectifier** is taking the place of the synchronous converter for many installations of 600 volts and higher. The rectifier is quieter in operation, has no heavy moving parts, and requires less space and less foundation than the converter. The efficiency of the rectifier is high over a wide range of load and increases with the voltage.

The application of mercury-arc rectifiers is extended by introduction of the grid, permitting control of voltage, current, or load. Grid-controlled rectifiers, rated at 2000 and 3000 kw. to supply 500 volts to chlorine cells, operate at an over-all conversion efficiency of 94.1 per cent according to

Rimlinger [*Elec. World*, **109**, 35 (1938)]. The grid control on these rectifiers permits operation to as low as 60 per cent of normal voltage.

Table 24. Synchronous Converters

Number of phases	Alternating-phase voltage	Direct voltage	% Output, based on d.-c. generator as 100%, at 100% P.F. and 100% Eff.
1	71	100	0.85
2	71	100	1.65
3	61	100	1.33
4	50	100	1.65
6	35	100	1.93

MOTORS

Direct-current Motors. Direct-current motors are structurally the same as direct-current generators. A given machine may be used as either motor or generator, though the characteristics will be slightly better for one method of operation.

Torque in a motor, from the fundamental equation, is $F = kBlI$, which may be written

$$\text{Torque (pounds at 1-ft. radius)} = 0.1175\left(\frac{Z}{\text{paths}}\right)\phi p I_a \times 10^{-8}$$

where Z is the total number of armature conductors; *paths*, the number of circuits in parallel through armature; p, the number of poles; ϕ, the flux per pole entering the armature; and I_a, the current to the armature.

It is evident that all factors are fixed by the design of the motor, except ϕ and I_a; or, $T = K\phi I_a$ for a given motor.

Speed in a motor is dependent on the voltage and field as follows:

$$V = E + I_aR_a$$

where V is the impressed voltage, E is the induced voltage in armature, and I_aR_a is resistance drop in armature circuit.

Since

$$E = k'\phi\,\text{r.p.m.} = V - I_aR_a$$

$$\text{R.p.m.} = \left(\frac{V - I_aR_a}{k'\phi}\right)$$

For commercial motors, under normal operation, I_aR_a is less than 5 per cent of V. Hence the speed is approximately proportional to V/ϕ.

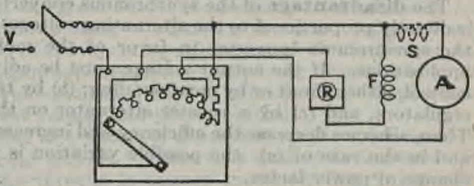


FIG. 60.—Starting box and shunt (or compound) motor connections. *R*, rheostat; *F*, shunt field; *A*, armature; *S*, series field.

Starting of direct-current motors requires a rheostat in the armature circuit, except for fractional-horsepower motors, because of the low inherent resistance of the armature. This rheostat, usually built into a starting box, increases the resistance of the armature circuit so that, during starting I_aR_a is the larger part of V.

Shunt motors are the most common type of direct-current motor, furnishing nearly constant speed (within 5 per cent) from no load to full load.

The speed of commercial motors may be increased nearly 25 per cent by field-circuit (rheostat) control; for small motors 50 per cent speed range is available (see Fig. 61 for characteristic curves). Figure 60 shows circuits for shunt motor with internal connections of starting box.

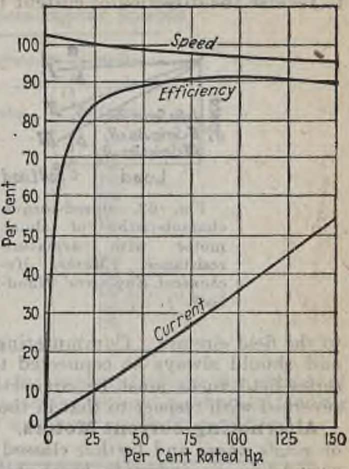


FIG. 61.—Typical characteristics of shunt motor.

Adjustable speed shunt motors are standard for 2:1, 3:1, and 4:1 speed range at increase in price. For greater speed range in a motor it is necessary to resort to other means than field-rheostat control, largely because of armature reaction effects. One scheme used successfully is to shift the armature axially, giving a flux change through the armature. In this type of motor the commutating poles are offset from the center line of the main poles, so that with the weak main field and hence greater armature reaction, the effect of the commutating poles is increased, giving good commutation over a large speed range. One of the best arrangements for increased speed range and good speed regulation is to provide independent voltage supplies for the field and armature. With constant voltage applied to the field, the motor speed will vary directly with the voltage applied to the armature. This arrangement requires a direct-current generator for each motor but it is quite common for applications requiring a wide speed range, such as rolling mills and elevators.

Armature circuit resistance may be used to *reduce the speed* of the motor with the load. Aside from the cost of the rheostat, the losses are relatively large, the regulation poor, and the speed decrease at light load is negligible. For a given decrease in speed, the rheostat loss is the same proportion of the input; *i.e.*, with 40 per cent speed reduction, the loss is 40 per cent of the input.

Potentiometer control is very convenient for obtaining reduced speed from small motors where the accompanying losses are not excessive. Referring to Fig. 63, the resistance R_1 and R_2 are proportioned to give the desired voltage to the armature. This reduced voltage will vary somewhat with load on the motor; the larger the current in R_1 relative to that in A, the more constant will be the voltage across A.

Series motors are used for loads requiring heavy starting torque, and where variable speed is permissible. The armature current is also the field current, hence the torque is approximately proportional to the current squared. The speed varies nearly inversely as the load current. Because of the excessive speed at no load, a series motor should not be connected to its load by a belt. The series motor is used for streetcars because of the heavy starting torque.

Compound-wound motors combine the characteristics of the shunt and series motors. The starting torque is good with speed regulation intermediate between that of the shunt and series motors. This type of motor is frequently used with a flywheel for heavy duty of short duration as on a punch press.

To **reverse** the direction of rotation of a direct-current motor, it is necessary to reverse the direction of current through the armature circuit with respect to the field circuits. Commutating poles are a part of the armature circuit and should always be connected to the armature in the same way. Any series-field turns must be connected so that current through them is not reversed with respect to that in the shunt field.

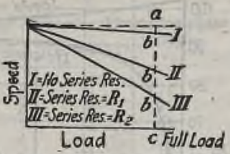


FIG. 62.—Speed-torque characteristics of shunt motor with armature resistance. (*Marks, "Mechanical Engineers' Handbook."*)

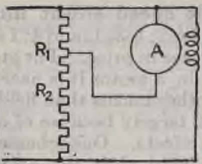


FIG. 63. Potentiometer control to operate small motor at low speed.

Alternating-current Motors. These motors may be classed as *polyphase* or *single phase* and further classed by type of connection or characteristics as *synchronous, induction, brush shifting*, etc.

The **polyphase synchronous motor** is constructed similarly to an alternator. An alternator deprived of its driving torque will continue to operate as a synchronous motor. The *speed* is fixed only by the frequency of the supply and the number of poles.

$$\text{R.p.m.} = 120\frac{f}{p}$$

where f is the frequency and p the number of poles.

Starting is facilitated by providing an amortisseur or damper winding in the pole faces of the synchronous motor. The field circuit is short-circuited through a small resistance, and low-voltage alternating current, from an autotransformer, is applied to the armature. The motor starts as an induction motor approaching synchronous speed. Direct current is applied to the field as the motor pulls in to synchronous speed, and full voltage is then applied to the armature. Modern synchronous motors have fair starting torque and with built-in clutch any reasonable load may be started.

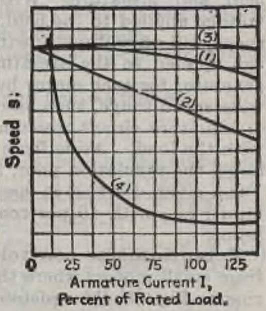


FIG. 64.—Speed-load characteristics of direct-current motors. 1, Shunt; 2, compound; 3, differential compound; 4, series. (*Marks, "Mechanical Engineers' Handbook."*)

The **excitation adjustments** by means of a field rheostat give the most important characteristics of the synchronous motor. By increasing the field current the power factor of the motor may be made leading, and by decreasing the field current the power factor may be made lagging. Synchronous motors are frequently operated with leading power factor to compensate for the lagging current of induction motors. Purchased power usually costs less if high power factor is maintained.

Table 25. Synchronous-motor Speeds

Poles	Revolutions per minute for	
	60 Cycles	25 Cycles
2	3600	1500
4	1800	750
6	1200	500
8	900	375
10	720	300
12	600	250
14	514.3	214.3
16	450	187.5
18	400	166.7
20	360	150

Synchronous condensers are used to regulate voltage by means of power-factor control. The synchronous machine without mechanical loading is called a "synchronous condenser" or "synchronous phase modifier."

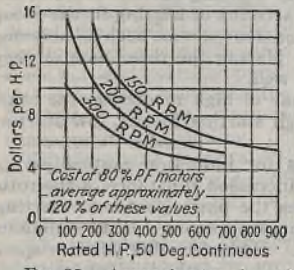


Fig. 65.—Approximate prices of synchronous motors. (*Amer. Architect, June 5, 1926.*)

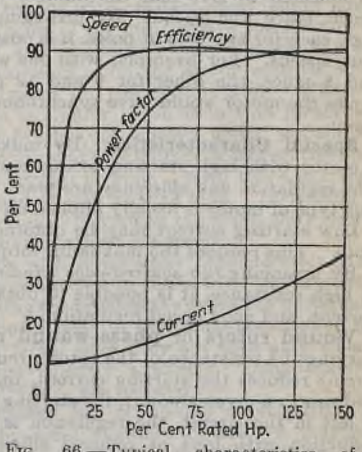


Fig. 66.—Typical characteristics of induction motor.

The **polyphase induction motor** is most useful because of its rugged construction. The **squirrel-cage** type has no moving contacts; the stator consists of a winding similar to that of the synchronous-motor armature; the rotor winding resembles a squirrel cage, bars secured to end rings. The rotor bars are sometimes bolted and soldered to the end connecting rings, sometimes welded and sometimes cast with the end rings.

The *speed characteristics* are similar to those of the direct-current shunt motor decreasing about 2 to 5 per cent from no load to full load; the no-load speed is the synchronous speed as defined under Synchronous Motors. The available speeds are limited and there is no speed adjustment. Characteristic curves are shown in Fig. 66. It will be noted that the slip (decrease in speed as a percentage of synchronous speed) increases almost directly with the load for normal loading. Hence from a measurement of the slip and knowledge of the slip at full load (name-plate data), the approximate load may be determined.

To reduce the current on *starting*, low voltage is applied from a starter which contains an autotransformer. The starting torque with normal voltage is not large and is reduced with voltage ($T = KE^2$ at constant speed). The starting voltage is usually about two-thirds of the normal. Small motors, below 5 h.p. (the largest permissible motor depends on the system) are started on full voltage. The stalling or breakdown torque of an induction motor is usually three times that of full load.

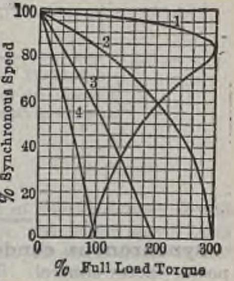


FIG. 67.—Speed-torque characteristics of induction motor with resistance in secondary circuit. (*"Standard Handbook for Electrical Engineers."*)

Multispeed motors of the squirrel-cage type are obtained by reconnecting the stator windings for different numbers of poles. A single winding may be reconnected to give half the number of poles and hence twice the speed. By arranging two windings, each for two sets of poles, it is possible to have four speeds. For example, with one winding for 4 and 8 poles, the other for 6 and 12 poles, on 60 cycles the motor would have synchronous speeds of 600, 900, 1200, and 1800 r.p.m.

Special Characteristics. By making the rotor with high resistance, a motor with high starting torque and lower starting current is obtained. The regulation and efficiency are poor. On account of heating in the rotor, this type of motor is usually applied on intermittent service such as hoisting.

Low starting current may be obtained by placing the rotor bars in deep slots. This reduces the maximum torque as well.

By arranging *two squirrel-cage windings*, one of high reactance, the other of high resistance, it is possible to obtain high starting torque, low starting current, and good speed regulation.

Wound rotors or **phase-wound rotors** for induction motors permit varying the resistance of the rotor circuit. Increased resistance of the rotor circuit reduces the starting current, improves the power factor at starting, and, up to a given amount, the starting torque is increased. If the resistance is left in the circuit, the regulation is poor. However, this permits speed reduction with load at reduced efficiency. This method of operation is exactly analogous to the operation of the shunt motor with resistance in the armature circuit. By removing the resistance, the speed is increased giving a characteristic similar to that of the squirrel-cage motor. The controller provides for starting the motor with the maximum resistance and then reducing the amount as the motor increases in speed. For certain applications water rheostats are provided; in some cases the amount of resistance is automatically adjusted so that the input to the motor is kept nearly constant.

Reversing of polyphase motors is accomplished by reversing of the relative phase rotation applied to either synchronous or induction motors. In two-phase motors this is accomplished by interchanging the leads of one phase. For a three-phase motor any two leads may be interchanged.

Speed Control. Inherently, an induction motor is a constant-speed motor. Its speed is dependent on the frequency. Voltage change will affect the speed a little, particularly in a motor with a high-resistance rotor; however, the effect is in general so small (nil at light loads) that it is not used. Pole changing and the wound-rotor motor with resistance have been discussed already.

The energy consumed in the resistor of the wound-rotor motor reduces the efficiency because it is wasted. This energy can be supplied to an alternating-current commutator motor (or converter), helping to drive the main motor (Kramer system) or to drive an induction generator feeding back to the electrical supply (Scherbius system). By suitable control devices, this commutating machine can also be used as a generator, causing the main motor to run above its synchronous speed. These devices permit adjustment of speed over a wide range, though the first cost limits the application to large motors.

Wound-rotor motors can be connected in series, *i.e.*, the secondary of one connected to the primary of the second, both motors being on the same shaft. The resultant speed is that of a motor having a number of poles equal to the sum of the poles of the two motors actually used. In practice the characteristics are not desirable, the efficiency and power factor being low with uneconomical utilization of material.

The **polyphase brush-shifting motor** with *constant-speed* (shunt motor) characteristics utilizes in one motor some of the principles of the Kramer set. The primary is placed on the rotor, and in addition there is a winding with commutator on the rotor. The secondary is on the stator with its circuit completed through two sets of brushes on the commutator. When the brushes are together, the operation is similar to an induction motor with short-circuited secondary. As the brushes are spread apart, an e.m.f. is introduced in the secondary corresponding to an IR drop; however, this e.m.f. is constant, *i.e.*, independent of the current flowing. With this e.m.f. introduced, the speed is reduced at no load as well as with load. Increasing the brush shift increases the speed reduction. Reversing the brush shift reverses the e.m.f. and permits operation above synchronous speed.

A polyphase brush-shifting motor with *series-motor characteristics* is also available. The stator windings are connected in series with the brushes on the commutating winding. By shifting the brushes the speed can be adjusted.

Single-phase motors are ordinarily used in fractional-horsepower sizes for many labor-saving devices. In larger sizes they are used only where a polyphase power supply is not conveniently available. Installations of single-phase motors of larger than 7½ h.p. are unusual.

The **single-phase induction motor** has no torque at standstill; the starting torque must be supplied by some auxiliary device. A polyphase induction motor, after being started, will continue to run with only one phase connected to the supply. The available output is reduced and hence the utilization of material is poorer; yet the power factor and efficiency for the smaller output are nearly as high as for the polyphase machine.

Split-phase starting utilizes an auxiliary winding which is disconnected after starting, usually by a centrifugal switch. This auxiliary circuit has the phase of its current displaced from that in the main winding by a series resistance or a condenser. These displaced currents are similar to those in a polyphase motor producing a small torque.

The **repulsion-start** induction motor utilizes a commutating winding on the armature with the brushes short-circuited giving torque as a repulsion motor. As the motor accelerates, a centrifugal device operates at about three-fourths speed to short-circuit the entire commutator. This makes the armature, in effect, a squirrel-cage rotor and the motor operates as an induction motor.

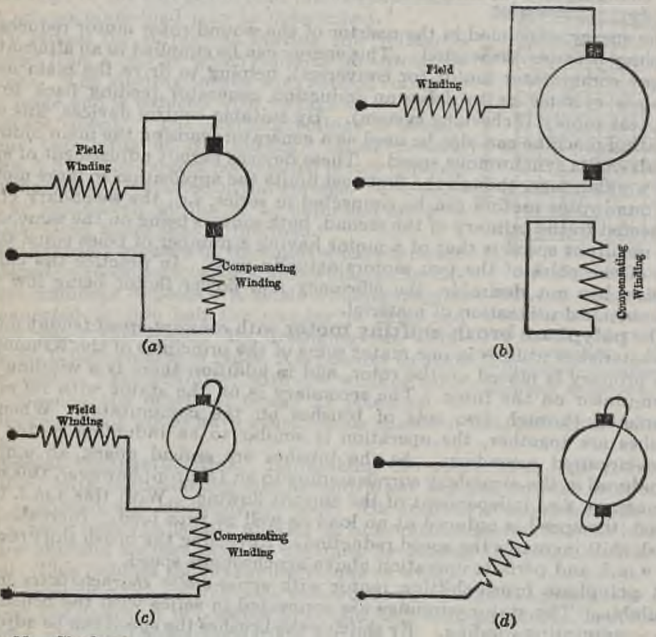


FIG. 68.—Single-phase motors with series characteristics. (*a*) Series motor conductively compensated; (*b*) series motor inductively compensated; *c* and *d*, repulsion motors. ("*Standard Handbook for Electrical Engineers.*")

Shading poles are used for starting small induction motors such as fans. A section of the pole area is enclosed by a low-resistance band. As the flux starts to change, an e.m.f. is induced in this shading band, the resulting current retarding the change of flux. Hence the flux in the shaded portion of the pole occurs after that in the unshaded portion; this phase displacement is sufficient to react on the squirrel-cage rotor and give a starting torque.

Series motors for operation on single-phase circuits require a completely laminated magnetic structure. To reduce sparking, only one armature turn is connected between commutator segments, resulting in a large number of segments. To reduce the reactance of the armature circuit a compensating winding is placed in slots in the pole faces; this winding may be short-circuited (inductive) (Fig. 68*b*), or in series with the armature (conductive) (Fig. 68*a*). To reduce the reactance of the field, fewer field turns are used compared with a normal direct-current motor, and more turns are needed on the armature. As a result the poles appear shorter and the armature larger than for the direct-current motor. Preventive resistance leads are frequently connected between the armature coils and the commutator segments to reduce sparking. The single-phase series motor is quite satisfactory on direct current and this type of motor is used as a universal motor. In the larger sizes the voltage applied when operated on direct current should be lower than when operated on alternating current.

Table 26. Approximate Full-load Current for Motors*

Horsepower	Single phase, 220 volts	Three phase				Direct current, 230 volts
		Squirrel cage		Wound rotor		
		220 volts	2200 volts	220 volts	2200 volts	
⅙	1.67					
¼	2.4					
½	3.5	2.5	..	.....	....	2.3
¾	4.7	2.8	..	.....	....	3.3
1	5.5	3.3	..	3.9	....	4.2
1½	7.6	4.7	..	.....	....	6.3
2	10	6	..	7.2	....	8.3
3	14	9	..	10	....	12.3
5	23	15	..	15	....	19.8
7½	34	22	..	25	....	28.7
10	43	27	..	28	....	38
15	.....	38	..	45	....	56
20	.....	52	..	56	....	74
25	.....	64	7	67	7.5	92
30	.....	77	8	82	9	110
40	.....	101	10	106	11	146
50	.....	125	13	128	14	180
60	.....	149	15	150	16	215
75	.....	180	19	188	19	268
100	.....	246	25	246	25	357
125	.....	310	32	310	32	443
150	.....	360	36	364	37	
200	.....	480	49	490	52	

Note. For motors of other voltages the current is inversely proportional to the voltage. Current for two-phase motors (four-wire) is about 87 per cent of the three-phase value.

* Compiled from the "National Electrical Code."

In the series motor (Fig. 68*b* and *c*) the current in the compensating winding was produced by transformer action from the armature. In a similar manner the current in the armature may be induced by transformer action from the compensating field as in Fig. 68*c*. This type of motor is the **repulsion motor** and a more suitable voltage may be obtained for the armature through the transformer action. A more common connection for the repulsion motor is shown in Fig. 68*d*, where the main field and compensating winding are combined into one. By changing the brush position, the relative amount of field and compensation component may be changed. This method of operation is common for starting single-phase induction motors.

Modifications for the series and repulsion motors have been developed with power supplied in various ways to the armature and field, some using auxiliary brushes; certain combinations give improved power factor, speed modifications, etc. The **repulsion-induction motor** is provided with two sets of windings on the armature, one with commutator giving repulsion characteristics and, at the same time, the other, a squirrel-cage, giving induction-motor characteristics.

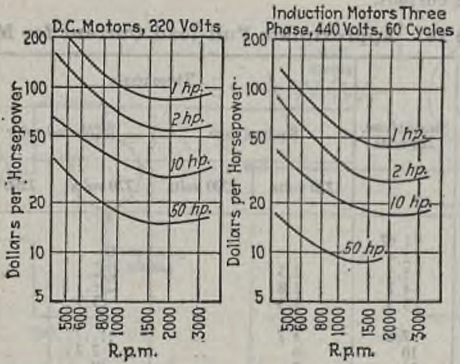


FIG. 69.—Approximate prices of motors with manually operated starting devices.

Reversing rotation of single-phase motors of the induction type is accomplished through the starting device. With a split-phase motor the connections to the starting winding may be reversed. In the repulsion motor the brushes are shifted. Shading-pole motors are built for one rotation only. Series motors are reversed by reversing the armature with respect to the field.

Condenser-type induction motors approximate polyphase characteristics in a motor operated from a single-phase supply. The motor is essentially a two-phase motor, one phase being supplied directly from the line, the other phase having a condenser in series with it, thus producing a split-phase effect. By selecting the proper condenser this circuit may be used throughout the operating range of the motor, with resulting good characteristics. The starting torque is small; for larger starting torque, additional capacitance may be connected and then disconnected as the motor comes up to speed.

SUMMARY OF MOTOR APPLICATIONS

The available power supply is the first consideration. The commercial supply of power is usually three phase, 60 cycles, and the alternating-current data given apply to this type of equipment; two-phase motors have similar characteristics and costs. Direct current is desirable for many drives because of the convenience of adjusting speeds. However, with an alternating-current supply the cost of converting to direct current (especially for one or two motors) may be so great as to make direct-current motors impractical. With an alternating-current supply of 2300 volts, motors larger than 75 h.p. may be operated at the supply voltage to eliminate cost of transformers.

Single-phase motors are used in small sizes where direct current or polyphase power is not conveniently available.

For **constant speed** the direct-current shunt motor, the induction motor, or the synchronous motor may be used. For *large starting torque*, the induction motor should have either a high-resistance squirrel-cage rotor (which causes more change in speed with load) or a wound rotor with resistance control. The synchronous-motor speed is the only one which does not change slightly with load. The direct-current shunt-motor speed may be kept constant by field control.

Table 27. Typical Prices of Motors without and with Usual Hand-control Devices
(Rated speed about 1150 r.p.m.)

Type	10 h.p.		20 h.p.		50 h.p.	
	Motor only	With control	Motor only	With control	Motor only	With control
Induction Motors, 440 volts, 60 cycles:						
Squirrel cage, normal	129	203	200	276	359	444
Squirrel cage, normal torque, low starting current	136	160	212	236	380	416
Squirrel cage, high torque, low starting current	151	175	233	257	417	453
Totally enclosed squirrel cage, fan cooled, 55°C. temperature rise, low starting current	185	209	371	395		
Wound rotor, constant speed	234	302	357	445	599	732
Wound rotor, adjustable speed	234	326	357	498	599	812
Multispeed, 1800, 1200, 900, 600 r.p.m., constant torque	289	390	407	651		
Multispeed, 1800, 1200, 900, 600 r.p.m. constant horsepower	374	460	514	693		
Two-speed, 1200, 600 constant horsepower	308	434	439	567		
Synchronous motor (0.8 P.F.) including exciter	...	...	449	726	569	855
A.-c. brush-shifting motor, adjustable speed	785	816	1075	1096	1955	2051
D.-c. shunt motor, 230 volts, constant speed	295	336	420	473	733	823
Adjustable speed, constant horsepower, 500 to 1500 r.p.m.	558	742	758	999	1334	1754
Totally enclosed, constant speed, 55°C. temperature rise	489	531	649	711	1724	1928

For **adjustable speed** the direct-current shunt motor is preferable as its change of speed with load is small, and the speed may be changed, at will, over a wide range. Similar characteristics are obtainable from the polyphase brush-shifting motor, though the change in speed with load is somewhat larger than with the direct-current motor. Where two or more fixed speeds are desired, the multispeed induction motor can be applied. For fairly constant load the wound-rotor induction motor with resistance can be utilized to give reduced speed.

For **heavy starting torque** or **intermittent heavy loads** the series direct-current motor may be used; if less speed variation is desired the compound direct-current motor should be applied. The wound-rotor motor with resistance can be similarly applied, though the losses will be greater. The double-rotor squirrel-cage motor supplies heavy starting torque, but for heavy intermittent loads the demand on the system would be great.

The **price** of a motor and its control equipment should always be considered in terms of the usefulness of the particular characteristics. For example assume a motor delivering 10 h.p. 2500 hr. per year. The cost of energy would be about $400. The motor cost might be only $200 if constant speed were permissible, or $700 if close adjustment of speed over a wide range were desired. The fixed charges on this difference in cost taken at 20 per cent would be $100; 25 per cent of the cost of the energy. Assuming, however, that the power cost was about 4 per cent of the total manufacturing cost, this additional cost of the adjustable speed motor would be economical if it reduced the total manufacturing cost or increased the value of product by more than 1 per cent. It must be assumed further that suitable electrical supply is available for either motor. In general, the *first cost* of suitable control in electrical apparatus must be considered in terms of operating value.

SECTION 25

ELECTROCHEMISTRY

BY

C. L. Mantell, Ph.D., Consulting Chemical Engineer, New York; Member American Institute of Mining and Metallurgical Engineers, American Institute of Chemical Engineers; Author of "Industrial Electrochemistry."

CONTENTS

ELECTROCHEMISTRY

REFERENCES: For general references to electrochemical processes and engineering, the reader is referred to Mantell, "Industrial Electrochemistry," 2d ed., McGraw-Hill, New York, 1940. For theoretical electrochemistry, reference should be made to Dole, "Experimental and Theoretical Electrochemistry," McGraw-Hill, New York, 1935; Foerster, "Elektrochemie wässeriger Lösungen," 4th ed., Barth, Leipzig, 1923; Walden, "Elektrochemie nichtwässeriger Lösungen," Barth, Leipzig, 1924; and Lorenz, "Elektrochemie geschmolzener Salze," Barth, Leipzig, 1909. None of the latter books is available in English. For plating, reference should be made to Blum and Hogaboom, "Principles of Electroplating and Electroforming," 2d ed., McGraw-Hill, New York, 1930; for calcium carbide to Taussig, "Die Industrie des Kalziumkarbides," Knapp, Halle, 1930. For the individual metals, reference should be made to the recent metallurgical works on this particular topic.

Electrochemical Industries. Electrochemistry has been defined as the science which treats of the chemical changes produced by the electric current and of the production of electricity from the energy of chemical reactions. Theoretically the two branches are of equal importance. Industrially, however, the chemical and physical changes produced by the use of the electric current are by far the more important.

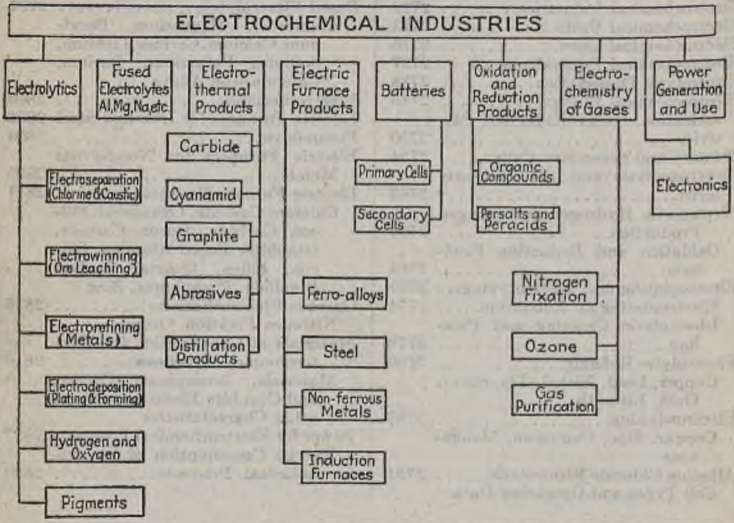


FIG. 1.—Classification of the electrochemical industries.

Electrochemical engineering is primarily a branch of chemical engineering to which portions and viewpoints of electrical engineering and metallurgy have been joined. Electrochemical engineering deals not only with all of the electrochemical theories, processes, and operations but also with the furnishing and utilization of electrical power to the industries; the design, construction, and operation of the equipment, machinery, and plants employed to produce the electrochemical products; the economic considerations involved in the competition of chemical and electrochemical methods for the preparation of the same or similar products, as well as the sale, distribution, and consumption of the materials produced.

Electrochemical industries may be roughly divided into several classes or groups, as shown in Fig. 1. Each group and subdivision has problems and applications of the chemical and electrical engineering unit processes but with the emphasis placed differently in the different groups.

The scope of the electrochemical engineering industries is very wide. To illustrate the breadth of the industries, a list of products of the electric furnace and electrolytic cell, at least as far as the major materials are concerned, is given (see Table 1).

Electrochemical processes of an endothermic (absorption of energy) nature have frequently supplanted purely chemical processes and in some cases allowed the production of new products which could hardly be obtained in any other way. Thus copper is now almost entirely refined by electrochemical means. All the chlorine used for water purification, sanitation, and bleaching is the product of electrolytic cells. Aluminum can be commercially made only by fused salt electrolysis. Calcium carbide and the synthetic abrasives of the silicon carbide or fused alumina type are not possible by other than electrothermal methods.

Chemical reactions often are made to take place by a series of steps because the most direct processes cannot be used and because of the difficulty of conversion of thermal into chemical energy. In electrochemical processes the needed energy is introduced in an electrical rather than a thermal form. When the electric current is used for heating, it may be applied at the point where it is desired. Electrochemical processes are more direct than the corresponding chemical ones. Sometimes the electrochemical method is more expensive but is preferred, inasmuch as purer products are produced.

Electrochemical processes generally operate satisfactorily only under constant conditions. They must be as simple as possible. Raw materials should be as pure as can be obtained or manufactured within the economic limitations of the process. Accumulation of impurities causes rapid decreases in efficiency.

Electrochemical Units. The **ampere** is defined electrochemically as the unvarying electric current which will deposit silver at the rate of 0.00111800 g. per sec. from a solution of $AgNO_3$ in water under a given set of conditions. The current in amperes flowing across any point, divided by the cross-sectional area of the conductor, is called the **current density** (c.d.) (amperes per square foot or amperes per centimeter square, etc.). So-called **c.d.** meters used for control purposes in electroplating consist of cathode surfaces of definite area, connected in series with an ammeter, the mounting and connections being so arranged that the meter appears above the level of the bath and the whole apparatus can be hung on the cathode rod. The meter thus indicates amperes per unit of area.

The **coulomb** or **ampere-second** is that quantity of electricity which will deposit 0.00111800 g. silver from a solution of $AgNO_3$.

The **volt** in practical use is that of the International System of Electrical Units. In this system the volt cannot be easily produced as defined, due to the definition of the ampere. The e.m.f. of a voltaic cell, however,

Table 1. Products of Electric Furnace and Electrolytic Cell

Product	Raw material	Applications of product
Alumina, fused	Bauxite (natural aluminum oxide)	Abrasives and refractories
Alumina, pure	Bauxite	Insulating material; aluminum metal
Aluminum metal	Bauxite	Electric power transmission cable; light-weight alloys for airplanes, automobiles, and trucks; deoxidizing agent for steel; aluminothermic reactions; ammonal (explosives); acid containers; cooking utensils
Aluminum, pure	Aluminum metal	Corrosion-resistant coatings
Beryllium	Beryl	Light alloys
Bismuth	Lead refining slimes	Alloys
Cadmium	Zinc electrowinning slimes	Alloys; plating
Calcium	Calcium chloride	Special uses; radio tubes; lamps
Calcium carbide	Lime and coke	Acetylene for welding, cutting, and lighting; acetone, acetic acid; airplane dope
Calcium cyanamid	Calcium carbide (nitrogen of the air)	Fertilizer; ammonia; cyanides
Carbon bisulfide	Coke and sulfur	Solvent; insecticide; carbon tetrachloride; artificial silk (viscose)
Caustic	Water, salt	Soap; paper industry; explosives
Cerium metals	Rare-earth chlorides	Pyrophoric alloys; automatic lighters; tracer bullets and shells
Chlorine gas	Water, salt	Bleaching; gas warfare; mustard gas, phosgene, chlorpicrin, silicon tetrachloride; explosives; chlorbenzol; water purification; surgery (Dakin solution); detinning; artificial plastics; hydrochloric acid; aluminum chloride for oil refining; sanitation
Chrome yellow	Lead	Paint pigment
Copper, pure	Crude copper	Electrical industry; brass
Copper, pure	Copper ore	Electrical industry; brass
Ferrochrome	Chrome ore	Special and high-speed steels; armor plate; projectiles
Ferromanganese	Manganese ore and coke	Steel; permanganates
Ferromolybdenum	Molybdenum ore	Special steels
Ferrosilicon	Iron, silica rock, coke	Steel manufacture; hydrogen production
Ferrosilicon-titanium	Bauxite	Steel deoxidizer
Ferrotitanium	Titanium ore	Scavenger in steel manufacture
Ferrotungsten	Tungsten ore	Special and high-speed steels
Ferrovanadium	Iron vanadate	Special steels; automobile steels
Gold	Copper refining slimes	Jewelry; coinage; industrial alloys
Graphite	Anthracite coal	Electrodes; lubricants; paints
Hydrogen	Water, sodium hydroxide	Ballooning; hydrogenated fats
Hypochlorite	Water, salt	Disinfectants; bleaches
Iron, pig	Iron ore	Steel industry
Iron, pure or "Swedish"	Pyrrhotite	Tubes and special steels
Lead refined	Crude lead	Alloys; fittings; acid chambers
Lithium metal	Lepidolite, lithium salts	Light alloys
Magnesium metal	Magnesium chloride	Flash-light powders; light-weight alloys; tracer bullets and flares

Table 1. Products of Electric Furnace and Electrolytic Cell—(*Concluded*)

Product	Raw material	Applications of product
Nickel, refined	Crude nickel	Alloys; plating industry; dairy equipment; utensils
Nitric acid	Air	Explosives; fertilizers
Oxygen	Water, sodium hydroxide	Oxy-welding; oxy-cutting
Ozone	Air	Sterilization of water; sanitation
Palladium	Nickel refining slimes	Industrial alloys
Perborates	Borax	Bleaching agents for textiles
Phosphoric acid	Phosphate rock, coke, and sand	Acid phosphates; cleaners; food products
Phosphorus	Phosphate rock, coke, and sand	Matches; phosphorus compounds; phosphor bronze; smoke screens
Platinum	Copper refining slimes	Electrical uses
Platinum	Nickel refining slimes	Catalysts; jewelry; industrial alloys
Potassium chlorate	Potassium chloride	Primers; matches; dyeing
Quartz, fused	Quartz rock	Silica tubes; heat-resisting materials; optical uses; lenses
Rhodium	Nickel refining slimes	Industrial alloys
Silicon	Sand and coke	Silicon steel; hydrogen for balloons; resistance units; silicides; silicon tetrachloride
Silicon carbide	Sand, sawdust, and coke	Abrasives and refractories
Silver	Copper refining slimes	Jewelry; coinage; industrial alloys
Sorbitol	Glucose	Humectants
Sodium bichromate	Chromium salts	Dyeing; tanning
Sodium metal	Caustic (Castner)	Peroxides; cyanides; bleaching; mining
Sodium metal	Salt	Alloys; tetraethyl lead; organic synthesis
Sodium perchlorate	Sodium salts, $NaClO_3$	Fireworks
Tin, refined	Impure tin, tin dross	Tin-plate industry; bronzes
White lead	Lead	Paint pigment
Zinc, pure	Zinc ore	Brass; galvanizing
Zinc metal, pure	Zinc ore	Brass industry

can be determined against the international ohm and the international ampere, and such a cell is used as a medium for realizing the international volt.

Standard cells have provided the means for making comparisons of e.m.f. The cells so used may be divided into primary standards, or normal cells, and secondary standards. The first are those by means of which the value of the volt is maintained, as at the National Bureau of Standards or elsewhere. The second are those suitable for general laboratory use.

The Weston standard cell consists of an amalgam of cadmium, a solution of cadmium sulfate having a concentration corresponding to that of a solution saturated at 4°C., and pure mercury overlaid with Hg_2SO_4. This combination has a very low temperature coefficient and is constant when properly made. However, it is not reproducible to the degree required in a primary standard. The "normal Weston" or "normal cadmium" cell, having an excess of cadmium sulfate, is therefore the standard maintained at the various government laboratories upon which rests the duty of establishing the volt. Its e.m.f. is taken as 1.0183 volts at 20°C. by international agreement. These cells are reproducible to better than 10 microvolts.

Electrochemical Laws. Conductors of electricity may be sharply divided into three classes. The first class, the *metallic* or *electronic* conductors, consists of the metals, alloys, and a few other substances such as carbon. The current passes through these without the accompaniment of any quantity of matter. Those of the second class are termed *electrolytic* conductors. They embrace in general the solutions of acids, bases, and salts, fused salts, some solid substances, and hot gases. In electrolytic conduction the passage of the current is always connected with a movement of matter. When the current leaves the electrolyte, it cannot take the matter with it; the latter is consequently set free. Chemical effects are produced. These mark the chief distinction between metallic or electronic and electrolytic conduction. In a third class with *mixed* conductors, the current passes partly in a metallic and partly in an electrolytic manner. Examples of these are the β form of silver sulphide and the solutions of the alkali and alkaline-earth metals in liquid ammonia. The greater part of electrochemistry is concerned with the chemical effects resulting from electrolytic conduction, and the corresponding electrical effects necessary for their production.

If two carbon plates be dipped into a dilute solution of HCl and connected with a source of d.c. such as a battery, electrolysis will take place. At the negative carbon plate, hydrogen gas is given off; while at the positive plate, chlorine is evolved. The gases are produced only at the carbon plates. The plates are termed *electrodes*, the negative the *cathode* and the positive one the *anode*. If a $AgNO_3$ solution be electrolyzed, silver is deposited on the cathode and oxygen evolved at the anode. Whatever the solution, chemical action takes place only at the electrodes.

Michael Faraday (1791–1867) discovered the quantitative relations between the amount of electricity which passes through a solution and the quantity of matter separated at the electrodes. His first law is: *The quantities of substances set free at the electrodes are directly proportional to the quantity of electricity which passes through the solution.* A second law expresses the fundamental relation between quantities of different substances liberated at the electrodes by the same quantity of electricity. It is: *The same quantity of electricity sets free the same number of equivalents of substances at the electrodes.* In other words, the quantity of silver liberated at the cathode by the passage of 20 coulombs electricity through a solution of a silver salt is double that which would be obtained by the passage of 10 coulombs. The passage of the same quantity of electricity through solutions of a silver salt, a copper salt, an iron salt, a zinc salt, and an acid will set free quantities of silver, copper, iron, zinc, and hydrogen which are proportional to their equivalent weights.

The neutral dissolved molecules of an electrolyte consist of two oppositely charged parts, called *ions*. Those which move toward the cathode are called *cations*, and those toward the anode, *anions*. When the current passes, the positive ions are attracted toward the negatively charged cathode where their charge is neutralized and they are set free. In a similar manner, the anions move to, and are discharged at, the anode.

The **electrochemical equivalent** of an element or a group of elements is the number of grams of that substance set free by the passage of 1 coulomb of electricity through an electrolyte. Electrochemical equivalents are proportional to chemical equivalents.

One coulomb of electricity (by definition) sets free 0.00111800 g. silver from a solution of a silver salt. If the gram equivalent of silver (its gram-

Table 2. Electrochemical Equivalents of the Elements

1		2	3	4	5	6	7	8	9
Element	Symbol	Atomic weight	Val. or chg. val.	Mg. per coulomb	Coulombs per mg.	G. per amp.-hr.	Amp.-hr. per g.	Lb. per 1,000 amp.-hr.	Amp.-hr. per lb.
Actinium..	Ac	**227***	3	0.78 4$\overline{11}$	1.27 $\overline{533}$	2.82 $\overline{280}$	0.35 4$\overline{26}$	6.22 $\overline{320}$	160.$\overline{689}$
			n	2.35 $\overline{233}$	0.42 5$\overline{11}$	8.46 $\overline{839}$	0.11 8$\overline{09}$	18.6$\overline{6}$ $\overline{961}$	53.5$\overline{63}$
Alabamine	Ab	**221***	7	0.32 7$\overline{17}$	3.05 $\overline{656}$	1.17 $\overline{779}$	0.84 9$\overline{04}$	2.59 $\overline{656}$	385.$\overline{120}$
			5	0.45 8$\overline{03}$	2.18 $\overline{326}$	1.64 $\overline{891}$	0.60 6$\overline{46}$	3.63 $\overline{523}$	275.$\overline{086}$
			3	0.76 3$\overline{39}$	1.30 $\overline{995}$	2.74 $\overline{819}$	0.36 3$\overline{88}$	6.05 $\overline{871}$	165.$\overline{052}$
			2	1.14 $\overline{508}$	0.87 3$\overline{30}$	4.12 $\overline{228}$	0.24 2$\overline{58}$	9.08 $\overline{807}$	110.$\overline{034}$
			1	2.29 $\overline{016}$	0.43 6$\overline{65}$	8.24 $\overline{456}$	0.12 1$\overline{29}$	18.1$\overline{7}$ $\overline{614}$	55.01$\overline{7}$
Aluminum.	Al	**26.97**	3	0.09 316	10.73 $\overline{415}$	0.33 53$\overline{8}$	2.98 1$\overline{71}$	0.73 93$\overline{8}$	1352.$\overline{480}$
			n	0.27 94$\overline{8}$	3.57 8$\overline{05}$	1.00 6$\overline{13}$	0.99 39$\overline{0}$	2.21 8$\overline{15}$	450.82$\overline{7}$
Antimony.	Sb	121.76	5	0.25 235	3.96 27$\overline{2}$	0.90 847	1.10 07$\overline{5}$	2.00 28$\overline{3}$	499.29$\overline{4}$
			3	0.42 059	2.37 76$\overline{3}$	1.51 41$\overline{1}$	0.66 045	3.33 80$\overline{5}$	299.57$\overline{6}$
			2	0.63 088	1.58 50$\overline{9}$	2.27 11$\overline{7}$	0.44 030	5.00 70$\overline{7}$	199.71$\overline{7}$
			n	1.26 17$\overline{6}$	0.79 254	4.54 23$\overline{4}$	0.22 015	10.01 4$\overline{15}$	99.858
Argon.....	A	**39.944**	*n*	0.41 393	2.41 58$\overline{8}$	1.49 01$\overline{4}$	0.67 108	3.28 51$\overline{9}$	304.39$\overline{6}$
Arsenic....	As	**74.91**	5	0.15 25$\overline{4}$	6.44 1$\overline{06}$	0.55 89$\overline{1}$	1.78 9$\overline{18}$	1.23 2$\overline{19}$	811.5$\overline{60}$
			3	0.25 87$\overline{6}$	3.86 4$\overline{64}$	0.93 15$\overline{2}$	1.07 3$\overline{51}$	2.05 3$\overline{66}$	486.9$\overline{36}$
			2	0.38 81$\overline{3}$	2.57 6$\overline{42}$	1.39 7$\overline{29}$	0.71 56$\overline{7}$	3.08 0$\overline{49}$	324.6$\overline{24}$
			n	0.77 62$\overline{7}$	1.28 8$\overline{21}$	2.79 4$\overline{57}$	0.35 78$\overline{4}$	6.16 0$\overline{97}$	162.3$\overline{12}$
Barium....	Ba	137.36	2	0.71 171	1.40 50$\overline{7}$	2.56 21$\overline{6}$	0.39 030	5.64 85$\overline{8}$	177.03$\overline{5}$
			n	1.42 34$\overline{2}$	0.70 253	5.12 43$\overline{1}$	0.19 515	11.29 71$\overline{7}$	88.518
Beryllium.	Be	9.02	2	0.04 67$\overline{4}$	21.3$\overline{9}$ $\overline{688}$	0.16 8$\overline{25}$	5.94 $\overline{358}$	0.37 0$\overline{92}$	269$\overline{5}$.$\overline{963}$
			n	0.09 34$\overline{7}$	10.6$\overline{9}$ $\overline{844}$	0.33 6$\overline{50}$	2.97 $\overline{179}$	0.74 1$\overline{85}$	134$\overline{7}$.$\overline{981}$
Bismuth...	Bi	209.00	5	0.43 316	2.30 86$\overline{1}$	1.55 93$\overline{8}$	0.64 128	3.43 78$\overline{4}$	290.88$\overline{0}$
			3	0.72 193	1.38 51$\overline{7}$	2.59 89$\overline{6}$	0.38 477	5.72 97$\overline{3}$	174.52$\overline{8}$
			2	1.08 29$\overline{0}$	0.92 345	3.89 84$\overline{5}$	0.25 651	8.59 46$\overline{0}$	116.35$\overline{2}$
			n	2.16 58$\overline{0}$	0.46 172	7.79 68$\overline{9}$	0.12 826	17.18 9$\overline{20}$	58.176
Boron.....	B	10.82	5	0.02 242	44.59 $\overline{337}$	0.08 073	12.38 $\overline{704}$	0.17 79$\overline{8}$	5618.$\overline{669}$
			3	0.03 737	26.75 $\overline{602}$	0.13 45$\overline{5}$	7.43 $\overline{223}$	0.29 66$\overline{2}$	3371.$\overline{201}$
			2	0.05 606	17.83 $\overline{735}$	0.20 18$\overline{2}$	4.95 4$\overline{82}$	0.44 49$\overline{5}$	2247.$\overline{467}$
			n	0.11 21$\overline{2}$	8.91 8$\overline{67}$	0.40 36$\overline{5}$	2.47 $\overline{741}$	0.88 98$\overline{9}$	1123.$\overline{734}$
Bromine...	Br	79.916	7	0.11 831	8.45 26$\overline{3}$	0.42 590	2.34 79$\overline{5}$	0.93 896	1065.0$\overline{13}$
			6	0.13 802	7.24 51$\overline{1}$	0.49 689	2.01 25$\overline{2}$	1.09 54$\overline{5}$	912.86$\overline{8}$
			5	0.16 563	6.03 75$\overline{9}$	0.59 626	1.67 70$\overline{8}$	1.31 45$\overline{4}$	760.72$\overline{4}$
			4	0.20 704	4.83 00$\overline{7}$	0.74 533	1.34 16$\overline{9}$	1.64 31$\overline{7}$	608.57$\overline{9}$
			3	0.27 605	3.62 25$\overline{5}$	0.99 377	1.00 62$\overline{6}$	2.19 09$\overline{0}$	456.43$\overline{4}$
			2	0.41 407	2.41 50$\overline{4}$	1.49 06$\overline{6}$	0.67 084	3.28 63$\overline{4}$	304.28$\overline{9}$
			1	0.82 815	1.20 75$\overline{2}$	2.98 13$\overline{2}$	0.33 542	6.57 26$\overline{9}$	152.14$\overline{5}$

Table 2. Electrochemical Equivalents of the Elements—(*Continued*)

1		2	3	4	5	6	7	8	9
Element	Symbol	Atomic weight	Val. or chg. val.	Mg. per coulomb	Coulombs per mg.	G. per amp.-hr.	Amp.-hr. per g.	Lb. per 1,000 amp.-hr.	Amp.-hr. per lb.
Cadmium..	Cd	112.41	2	0.58 244	1.71 693̅	2.09 677̅	0.47 692	4.62 258̅	216.329̅
			n	1.16 487̅	0.85 846	4.19 353̅	0.23 846	9.24 516̅	108.165̅
Calcium...	Ca	**40.08**	2	0.20 767̅	4.81 53̅7̅	0.74 761̅	1.33 76̅0̅	1.64 81̅9̅	606.72̅6̅
			n	0.41 534̅	2.40 76̅8̅	1.49 52̅1̅	0.66 880̅	3.29 63̅8̅	303.36̅3̅
Carbon....	C	**12.010**	4	0.03 111	32.13 98̅9̅	0.11 201	8.92 775̅	0.24 694	4049.55̅8̅
			2	0.06 223	16.06 99̅4̅	0.22 402	4.46 387̅	0.49 388	2024.77̅9̅
			n	0.12 446	8.03 497̅	0.44 804	2.23 194̅	0.98 776	1012.39̅0̅
Cerium....	Ce	140.13	4	0.36 303	2.75 459̅	1.30 691̅	0.76 516	2.88 125̅	347.072̅
			3	0.48 404	2.06 594̅	1.74 255̅	0.57 387	3.84 166̅	260.304̅
			1	1.45 212̅	0.68 865	5.22 765̅	0.19 129	11.52 49̅9̅	86.768
Cesium....	Cs	**132.91**	1	1.37 731̅	0.72 606	4.95 830̅	0.20 168	10.93 11̅8̅	91.481
Chlorine...	Cl	35.457	7	0.05 248 9	19.05 12̅4̅	0.18 896	5.29 201̅	0.41 659	2400.41̅7̅
			6	0.06 123 8	16.32 96̅4̅	0.22 046	4.53 601̅	0.48 603	2057.50̅0̅
			5	0.07 348 6	13.60 80̅3̅	0.26 455	3.78 001̅	0.58 323	1714.58̅3̅
			4	0.09 185 8	10.88 64̅2̅	0.33 069	3.02 401̅	0.72 904	1373.66̅7̅
			3	0.12 248	8.16 482̅	0.44 092	2.26 801̅	0.97 205	1028.75̅0̅
			2	0.18 372	5.44 321̅	0.66 137	1.51 200̅	1.45 808̅	685.833̅
			1	0.36 743	2.72 161̅	1.32 275̅	0.75 600	2.91 616̅	342.917̅
Chromium.	Cr	52.01	6	0.08 983	10.13 2̅4̅7̅	0.32 338̅	3.09 23̅5̅	0.71 293̅	1402.6̅6̅8̅
			4	0.13 474̅	7.42 16̅5̅	0.48 507̅	2.06 15̅7̅	1.06 93̅9̅	935.11̅2̅
			3	0.17 965̅	5.56 62̅4̅	0.64 676̅	1.54 61̅8̅	1.42 58̅5̅	701.33̅4̅
			2	0.26 948̅	3.71 08̅2̅	0.97 013̅	1.03 078̅	2.13 87̅8̅	467.55̅6̅
			1	0.53 896̅	1.85 54̅1̅	1.94 02̅7̅	0.51 539̅	4.27 75̅6̅	233.77̅8̅
Cobalt....	Co	58.94	3	0.20 359̅	4.91 17̅7̅	0.73 287̅	1.36 45̅0̅	1.61 57̅0̅	618.92̅5̅
			2	0.30 539̅	3.27 45̅2̅	1.09 93̅1̅	0.90 966̅	2.42 35̅6̅	412.61̅7̅
			1	0.61 078̅	1.63 72̅6̅	2.19 86̅1̅	0.45 483̅	4.84 71̅1̅	206.30̅8̅
Columbium	Cb	**92.91**	5	0.19 256̅	5.19 32̅0̅	0.69 321̅	1.44 25̅5̅	1.52 82̅8̅	654.33̅2̅
			4	0.24 070̅	4.15 45̅6̅	0.86 652̅	1.15 40̅4̅	1.91 03̅5̅	523.46̅6̅
			3	0.32 093̅	3.11 59̅2̅	1.14 70̅2̅	0.86 553̅	2.54 71̅3̅	392.59̅9̅
			2	0.48 140̅	2.07 72̅8̅	1.73 30̅4̅	0.57 702̅	3.82 06̅9̅	261.73̅3̅
			1	0.96 280̅	1.03 86̅4̅	3.46 60̅7̅	0.28 851̅	7.64 13̅8̅	130.86̅6̅
Copper....	Cu	63.57	2	0.32 938̅	3.03 60̅2̅	1.18 57̅6̅	0.84 334̅	2.61 41̅6̅	382.53̅2̅
			1	0.65 876̅	1.51 80̅1̅	2.37 152̅	0.42 167̅	5.22 83̅1̅	191.26̅6̅
Dysprosium.	Dy	162.46	3	0.56 069	1.78 351̅	2.01 849̅	0.49 542	4.44 901̅	224.886̅
			n	1.68 207̅	0.59 450	6.05 547̅	0.16 514	13.34 00̅2̅	74.962
Erbium....	Er	167.2	3	0.57 755̅	1.73 14̅5̅	2.07 91̅7̅	0.48 096̅	4.58 37̅8̅	218.16̅0̅
			n	1.73 26̅4̅	0.57 715̅	6.23 75̅1̅	0.16 032̅	13.75 1̅3̅5̅	72.720̅

Table 2. Electrochemical Equivalents of the Elements—(*Continued*)

1		2	3	4	5	6	7	8	9
Element	Symbol	Atomic weight	Val. or chg. val.	Mg. per coulomb	Coulombs per mg.	G. per amp.-hr.	Amp.-hr. per g.	Lb. per 1,000 amp.-hr.	Amp.-hr. per lb.
Europium	Eu	152.0	3	0.52 50$\overline{4}$	1.90 46$\overline{1}$	1.89 01$\overline{6}$	0.52 90$\overline{6}$	4.16 70$\overline{8}$	241.91$\overline{1}$
			n	1.57 51$\overline{3}$	0.63 487	5.67 04$\overline{7}$	0.17 63$\overline{5}$	12.40 12$\overline{4}$	80.63$\overline{7}$
Fluorine	F	19.00	1	0.19 68$\overline{9}$	5.07 89$\overline{5}$	0.70 88$\overline{1}$	1.41 08$\overline{2}$	1.56 26$\overline{5}$	639.93$\overline{7}$
Gadolinium	Gd	**156.9**	3	0.54 17$\overline{9}$	1.84 57$\overline{2}$	1.95 04$\overline{6}$	0.51 27$\overline{0}$	4.30 00$\overline{2}$	232.73$\overline{7}$
			n	1.62 53$\overline{8}$	0.61 52$\overline{4}$	5.85 13$\overline{7}$	0.17 09$\overline{0}$	12.90 00$\overline{7}$	77.57$\overline{9}$
Gallium	Ga	69.72	3	0.24 08$\overline{3}$	4.15 23$\overline{2}$	0.86 69$\overline{8}$	1.15 34$\overline{2}$	1.91 13$\overline{7}$	523.18$\overline{4}$
			2	0.36 12$\overline{4}$	2.76 82$\overline{2}$	1.30 04$\overline{8}$	0.76 89$\overline{5}$	2.86 70$\overline{6}$	348.78$\overline{9}$
			1	0.72 24$\overline{9}$	1.38 41$\overline{1}$	2.60 09$\overline{5}$	0.38 44$\overline{7}$	5.73 41$\overline{2}$	174.39$\overline{5}$
Germanium	Ge	72.60	4	0.18 80$\overline{8}$	5.31 68$\overline{0}$	0.67 71$\overline{0}$	1.47 68$\overline{9}$	1.49 27$\overline{5}$	669.90$\overline{6}$
			2	0.37 61$\overline{7}$	2.65 84$\overline{0}$	1.35 42$\overline{0}$	0.73 84$\overline{5}$	2.98 54$\overline{9}$	334.95$\overline{3}$
			n	0.75 23$\overline{3}$	1.32 92$\overline{0}$	2.70 83$\overline{9}$	0.36 92$\overline{2}$	5.97 09$\overline{9}$	167.47$\overline{6}$
Gold	Au	197.2	3	0.68 11$\overline{7}$	1.46 80$\overline{5}$	2.45 22$\overline{3}$	0.40 77$\overline{9}$	5.40 62$\overline{4}$	184.97$\overline{2}$
			2	1.02 17$\overline{6}$	0.97 87$\overline{0}$	3.67 83$\overline{4}$	0.27 18$\overline{6}$	8.10 93$\overline{6}$	123.31$\overline{4}$
			1	2.04 35$\overline{2}$	0.48 93$\overline{5}$	7.35 66$\overline{8}$	0.13 59$\overline{3}$	16.21 87$\overline{1}$	61.65$\overline{7}$
Hafnium	Hf	178.6	4	0.46 26$\overline{9}$	2.16 12$\overline{5}$	1.66 57$\overline{0}$	0.60 03$\overline{5}$	3.67 22$\overline{3}$	272.31$\overline{3}$
			n	1.85 07$\overline{8}$	0.54 03$\overline{1}$	6.66 28$\overline{0}$	0.15 00$\overline{9}$	14.68 89$\overline{5}$	68.07$\overline{8}$
Helium	He	4.003	*n*	0.04 148 $\overline{2}$	24.10 69$\overline{2}$	0.14 93$\overline{3}$	6.69 63$\overline{7}$	0.32 92$\overline{3}$	3037.42$\overline{1}$
Holmium	Ho	163.5	3	0.56 47$\overline{7}$	1.77 06$\overline{4}$	2.03 31$\overline{6}$	0.49 18$\overline{5}$	4.48 23$\overline{5}$	223.09$\overline{7}$
			n	1.69 43$\overline{0}$	0.59 02$\overline{1}$	6.09 94$\overline{8}$	0.16 39$\overline{5}$	13.44 70$\overline{5}$	74.36$\overline{6}$
Hydrogen	H	1.0081	1	0.01 044 7	95.72 46$\overline{5}$	0.03 760 8	26.59 01$\overline{8}$	0.08 291	12061.10$\overline{2}$
Deuterium†	D	**2.01471**	1	0.02 087 8	47.89 77$\overline{1}$	0.07 516 0	13.30 49$\overline{2}$	0.16 570	6035.01$\overline{1}$
Illinium	Il	**146***	3	0.50 43$\overline{2}$	1.98 28$\overline{8}$	2.72 33$\overline{2}$	0.55 08$\overline{0}$	3.90 25$\overline{6}$	272.53$\overline{9}$
			n	1.51 29$\overline{5}$	0.66 09$\overline{6}$	5.44 66$\overline{3}$	0.18 36$\overline{0}$	11.00 76$\overline{0}$	90 84$\overline{6}$
Indium	In	**114.76**	3	0.39 641	2.52 26$\overline{6}$	1.42 70$\overline{7}$	0.70 074	3.14 61$\overline{4}$	317.84$\overline{9}$
			n	1.18 92$\overline{2}$	0.84 089	4.28 12$\overline{0}$	0.23 358	9.43 84$\overline{3}$	105.95$\overline{0}$
Iodine	I	**126.92**	7	0.18 790	5.32 22$\overline{5}$	0.67 641	1.47 84$\overline{0}$	1.49 12$\overline{2}$	670.59$\overline{3}$
			6	0.21 921	4.56 19$\overline{3}$	0.78 914	1.26 72$\overline{0}$	1.73 97$\overline{6}$	574.79$\overline{4}$
			5	0.26 305	3.80 16$\overline{1}$	0.94 697	1.05 60$\overline{0}$	2.08 77$\overline{1}$	478.99$\overline{5}$
			4	0.32 881	3.04 12$\overline{9}$	1.18 37$\overline{1}$	0.84 480	2.60 96$\overline{3}$	383.19$\overline{6}$
			3	0.43 841	2.28 09$\overline{6}$	1.57 82$\overline{8}$	0.63 360	3.47 95$\overline{1}$	287.39$\overline{7}$
			2	0.65 762	1.52 06$\overline{4}$	2.36 74$\overline{2}$	0.42 240	5.21 92$\overline{7}$	191.59$\overline{8}$
			1	1.31 52$\overline{3}$	0.76 032	4.73 48$\overline{4}$	0.21 120	10.43 85$\overline{3}$	95.799

Table 2. Electrochemical Equivalents of the Elements—(*Continued*)

1		2	3	4	5	6	7	8	9
Element	Symbol	Atomic weight	Val. or chg. val.	Mg. per coulomb	Coulombs per mg.	G. per amp.-hr.	Amp.-hr. per g.	Lb. per 1,000 amp.-hr.	Amp.-hr. per lb.
Iridium....	Ir	193.1	4	0.50 02$\overline{6}$	1.99 89$\overline{6}$	1.80 09$\overline{5}$	0.55 54$\overline{6}$	3.97 03$\overline{8}$	251.86$\overline{5}$
			3	0.66 70$\overline{1}$	1.49 92$\overline{2}$	2.40 12$\overline{4}$	0.41 65$\overline{9}$	5.29 38$\overline{4}$	188.89$\overline{9}$
			1	2.00 10$\overline{4}$	0.49 97$\overline{4}$	7.20 37$\overline{3}$	0.13 88$\overline{6}$	15.88 15$\overline{1}$	62.96$\overline{6}$
Iron.......	Fe	55.84	3	0.19 28$\overline{8}$	5.18 44$\overline{6}$	0.69 43$\overline{8}$	1.44 01$\overline{3}$	1.53 08$\overline{5}$	653.23$\overline{0}$
			2	0.28 93$\overline{3}$	3.45 63$\overline{0}$	1.04 15$\overline{8}$	0.96 00$\overline{8}$	2.29 62$\overline{8}$	435.48$\overline{7}$
			1	0.57 86$\overline{5}$	1.72 81$\overline{5}$	2.08 31$\overline{5}$	0.48 00$\overline{4}$	4.59 25$\overline{6}$	217.77$\overline{4}$
Krypton...	Kr	**83.7**	*n*	0.86 73$\overline{6}$	1.15 29$\overline{3}$	3.12 24$\overline{9}$	0.32 02$\overline{6}$	6.88 39$\overline{0}$	145.26$\overline{6}$
Lanthanum	La	**138.92**	3	0.47 986	2.08 39$\overline{3}$	1.72 75$\overline{0}$	0.57 887	3.80 84$\overline{9}$	262.57$\overline{1}$
			n	1.43 95$\overline{9}$	0.69 464	5.18 25$\overline{1}$	0.19 296	11.42 54$\overline{7}$	87.524
Lead......	Pb	**207.21**	4	0.53 681	1.86 28$\overline{4}$	1.93 25$\overline{3}$	0.51 746	4.26 05$\overline{0}$	234.71$\overline{5}$
			2	1.07 36$\overline{3}$	0.93 142	3.86 50$\overline{6}$	0.25 873	8.52 09$\overline{9}$	117.35$\overline{7}$
			1	2.14 72$\overline{5}$	0.46 571	7.73 01$\overline{1}$	0.12 936	17.04 19$\overline{8}$	58.679
Lithium...	Li	6.940	1	0.07 192	13.90 49$\overline{0}$	0.25 89$\overline{0}$	3.86 24$\overline{7}$	0.57 07$\overline{8}$	1751.98$\overline{8}$
Lutecium..	Lu	175.0	3	0.60 44$\overline{9}$	1.65 42$\overline{9}$	3.26 42$\overline{5}$	0.45 95$\overline{2}$	7.34 87$\overline{4}$	208.43$\overline{7}$
			n	1.81 34$\overline{7}$	0.55 14$\overline{3}$	6.52 84$\overline{9}$	0.15 31$\overline{7}$	14.39 28$\overline{7}$	69.47$\overline{9}$
Magnesium	Mg	24.32	2	0.12 60$\overline{1}$	7.93 58$\overline{6}$	0.45 36$\overline{4}$	2.20 44$\overline{0}$	1.00 01$\overline{0}$	999.90$\overline{1}$
			n	0.25 20$\overline{2}$	3.96 79$\overline{3}$	0.90 72$\overline{7}$	1.10 22$\overline{0}$	2.00 02$\overline{0}$	499.95$\overline{1}$
Manganese	Mn	54.93	7	0.08 132	12.29 77$\overline{2}$	0.29 27$\overline{4}$	3.41 60$\overline{3}$	0.64 53$\overline{7}$	1549.48$\overline{7}$
			6	0.09 487	10.54 09$\overline{0}$	0.34 15$\overline{3}$	2.92 80$\overline{3}$	0.75 29$\overline{4}$	1328.13$\overline{2}$
			5	0.11 38$\overline{4}$	8.78 40$\overline{9}$	0.50 98$\overline{3}$	2.44 00$\overline{2}$	0.90 35$\overline{2}$	1106.77$\overline{7}$
			4	0.14 23$\overline{0}$	7.02 72$\overline{7}$	0.51 22$\overline{9}$	1.95 20$\overline{2}$	1.12 94$\overline{1}$	885.42$\overline{1}$
			3	0.18 97$\overline{4}$	5.27 04$\overline{5}$	0.68 30$\overline{5}$	1.46 40$\overline{1}$	1.50 58$\overline{7}$	664.06$\overline{6}$
			2	0.28 46$\overline{1}$	3.51 36$\overline{3}$	1.02 45$\overline{8}$	0.97 60$\overline{1}$	2.25 88$\overline{1}$	442.71$\overline{1}$
			1	0.56 92$\overline{1}$	1.75 68$\overline{2}$	2.04 91$\overline{6}$	0.48 80$\overline{0}$	4.51 76$\overline{2}$	221.35$\overline{5}$
Masurium.	Ma	**97.8***	7	0.14 47$\overline{8}$	6.90 69$\overline{5}$	0.52 12$\overline{1}$	1.91 86$\overline{0}$	1.14 90$\overline{8}$	870.26$\overline{2}$
			n	1.01 34$\overline{7}$	0.98 67$\overline{1}$	3.64 85$\overline{0}$	0.27 40$\overline{9}$	8.04 35$\overline{6}$	124.32$\overline{3}$
Mercury...	Hg	200.61	2	1.03 94$\overline{3}$	0.96 207	3.74 19$\overline{5}$	0.26 724	8.24 95$\overline{8}$	121.21$\overline{8}$
			1	2.07 88$\overline{6}$	0.48 103	7.48 39$\overline{0}$	0.13 362	16.49 91$\overline{7}$	60.609
Molybdenum.	Mo	95.95	6	0.16 57$\overline{2}$	6.03 43$\overline{9}$	0.59 65$\overline{8}$	1.67 62$\overline{2}$	1.31 52$\overline{3}$	760.32$\overline{1}$
			5	0.19 88$\overline{6}$	5.02 86$\overline{6}$	0.71 59$\overline{0}$	1.39 68$\overline{5}$	1.57 82$\overline{8}$	633.60$\overline{1}$
			4	0.24 85$\overline{8}$	4.02 29$\overline{3}$	0.89 48$\overline{7}$	1.11 74$\overline{8}$	1.97 28$\overline{5}$	506 88$\overline{1}$
			3	0.33 14$\overline{3}$	3.01 72$\overline{0}$	1.19 31$\overline{6}$	0.83 81$\overline{1}$	2.63 04$\overline{7}$	380.16$\overline{0}$
			2	0.49 71$\overline{5}$	2.01 14$\overline{6}$	1.78 97$\overline{4}$	0.55 87$\overline{4}$	3.94 57$\overline{0}$	253.44$\overline{0}$
			1	0.99 43$\overline{0}$	1.00 57$\overline{3}$	3.57 94$\overline{8}$	0.27 93$\overline{7}$	7.89 14$\overline{1}$	126.72$\overline{0}$
Neodymium.	Nd	144.27	3	0.49 834	2.00 66$\overline{5}$	1.79 40$\overline{3}$	0.55 740	3.95 51$\overline{6}$	252.83$\overline{4}$
			n	1.49 50$\overline{3}$	0.66 888	5.38 20$\overline{9}$	0.18 580	11.86 54$\overline{8}$	84.278

Table 2. Electrochemical Equivalents of the Elements—(*Continued*)

1		2	3	4	5	6	7	8	9
Element	Symbol	Atomic weight	Val. or chg. val.	Mg. per coulomb	Coulombs per mg.	G. per amp.-hr.	Amp.-hr. per g.	Lb. per 1,000 amp.-hr.	Amp.-hr. per lb.
Neon......	Ne	20.183	*n*	0.20 915	4.78 12$\overline{5}$	0.75 294	1.32 81$\overline{3}$	1.65 99$\overline{5}$	602.42$\overline{8}$
Nickel.....	Ni	58.69	3	0.20 27$\overline{3}$	4.93 $\overline{270}$	0.72 98$\overline{2}$	1.37 0$\overline{19}$	1.60 8$\overline{99}$	621.5$\overline{09}$
			2	0.30 40$\overline{9}$	3.28 $\overline{846}$	1.09 4$\overline{74}$	0.91 34$\overline{6}$	2.41 3$\overline{48}$	414.3$\overline{40}$
			1	0.60 81$\overline{9}$	1.64 4$\overline{23}$	2.18 9$\overline{47}$	0.45 67$\overline{3}$	4.82 6$\overline{96}$	207.1$\overline{70}$
Nitrogen ..	N	14.008	5	0.02 903 2	34.44 4$\overline{46}$	0.10 452	9.56 79$\overline{5}$	0.23 042	438.99$\overline{5}$
			4	0.03 629 0	27.55 5$\overline{68}$	0.13 064	7.65 43$\overline{6}$	0.28 802	347.19$\overline{6}$
			3	0.04 838 7	20.66 6$\overline{76}$	0.17 419	5.74 07$\overline{7}$	0.38 403	260.39$\overline{7}$
			2	0.07 258 0	13.77 $\overline{784}$	0.26 129	3.82 71$\overline{8}$	0.57 604	173.59$\overline{8}$
			1	0.14 516	6.88 89$\overline{2}$	0.52 258	1.91 35$\overline{9}$	1.15 20$\overline{9}$	86.799
Osmium...	Os	190.2	8	0.24 63$\overline{7}$	4.05 8$\overline{89}$	0.88 69$\overline{4}$	1.12 7$\overline{47}$	1.95 5$\overline{37}$	447.4$\overline{85}$
			6	0.32 85$\overline{0}$	3.04 4$\overline{16}$	1.18 2$\overline{59}$	0.84 56$\overline{0}$	2.60 7$\overline{17}$	383.5$\overline{58}$
			5	0.39 42$\overline{0}$	2.53 $\overline{680}$	1.41 9$\overline{11}$	0.70 46$\overline{7}$	3.12 $\overline{860}$	319.63$\overline{2}$
			4	0.49 27$\overline{5}$	2.02 9$\overline{44}$	1.77 3$\overline{89}$	0.56 37$\overline{3}$	3.91 07$\overline{5}$	255.7$\overline{06}$
			3	0.65 7$\overline{00}$	1.52 2$\overline{08}$	2.36 5$\overline{18}$	0.42 28$\overline{0}$	5.21 4$\overline{33}$	191.7$\overline{79}$
			2	0.98 54$\overline{9}$	1.01 4$\overline{72}$	3.54 7$\overline{77}$	0.28 18$\overline{7}$	7.82 1$\overline{50}$	127.8$\overline{53}$
			1	1.97 0$\overline{98}$	0.50 73$\overline{6}$	7.09 5$\overline{54}$	0.14 09$\overline{3}$	15.64 $\overline{299}$	63.92$\overline{6}$
Oxygen....	O	16.0000	2	0.08 290 2	12.06 25$\overline{0}$	0.29 845	3.35 069	0.65 796	1519.85$\overline{0}$
			n	0.16 580	6.03 125	0.59 689	1.67 535	1.31 592	759.925
Palladium .	Pd	106.7	4	0.27 64$\overline{2}$	3.61 7$\overline{62}$	0.99 51$\overline{3}$	1.00 4$\overline{89}$	2.19 3$\overline{88}$	455.8$\overline{12}$
			3	0.36 85$\overline{7}$	2.71 $\overline{332}$	1.32 6$\overline{84}$	0.75 36$\overline{7}$	2.92 5$\overline{18}$	340.8$\overline{59}$
			2	0.55 28$\overline{5}$	1.80 8$\overline{81}$	1.99 0$\overline{26}$	0.50 24$\overline{5}$	4.38 $\overline{777}$	227.9$\overline{06}$
			1	1.10 5$\overline{70}$	0.90 44$\overline{0}$	3.98 0$\overline{52}$	0.25 12$\overline{2}$	8.77 5$\overline{54}$	113.9$\overline{53}$
Phosphorus	P	31.02	5	0.06 429	15.55 $\overline{448}$	0.23 14$\overline{4}$	4.32 0$\overline{69}$	0.51 02$\overline{5}$	1959.$\overline{832}$
			3	0.10 71$\overline{5}$	9.33 2$\overline{69}$	0.38 57$\overline{4}$	2.59 2$\overline{41}$	0.85 04$\overline{1}$	1175.$\overline{899}$
			2	0.16 07$\overline{3}$	6.22 1$\overline{79}$	0.57 86$\overline{1}$	1.72 8$\overline{28}$	1.27 5$\overline{62}$	783.9$\overline{33}$
			n	0.32 14$\overline{5}$	3.11 $\overline{090}$	1.15 7$\overline{22}$	0.86 41$\overline{4}$	2.55 1$\overline{24}$	391.9$\overline{66}$
Platinum ..	Pt	195.23	4	0.50 578	1.97 71$\overline{6}$	1.82 08$\overline{0}$	0.54 921	4.01 41$\overline{7}$	249.11$\overline{7}$
			2	1.01 15$\overline{5}$	0.98 858	3.64 16$\overline{0}$	0.27 460	8.02 83$\overline{4}$	124.55$\overline{9}$
			n	2.02 31$\overline{1}$	0.49 429	7.28 31$\overline{9}$	0.13 730	16.05 6$\overline{69}$	62.279
Polonium ..	Po	**210***	6	0.36 2$\overline{69}$	2.75 $\overline{714}$	1.30 $\overline{570}$	0.76 5$\overline{87}$	2.87 $\overline{857}$	347.$\overline{394}$
			4	0.54 4$\overline{04}$	1.83 $\overline{810}$	1.95 $\overline{855}$	0.51 0$\overline{58}$	4.31 $\overline{786}$	231.$\overline{596}$
			2	1.08 $\overline{808}$	0.91 9$\overline{05}$	3.91 $\overline{710}$	0.25 5$\overline{29}$	8.63 $\overline{572}$	115.$\overline{798}$
			n	2.17 $\overline{617}$	0.45 9$\overline{52}$	7.83 $\overline{420}$	0.12 7$\overline{65}$	17.2$\overline{7}$ $\overline{145}$	57.8$\overline{99}$
Potassium .	K	39.096	1	0.40 514	2.46 82$\overline{8}$	1.45 85$\overline{0}$	0.68 563	3.21 54$\overline{5}$	310.99$\overline{8}$
Praseodymium.	Pr	140.92	3	0.48 677	2.05 43$\overline{6}$	1.75 23$\overline{7}$	0.57 065	3.86 33$\overline{2}$	258.84$\overline{5}$
			n	1.46 03$\overline{1}$	0.68 479	5.25 71$\overline{2}$	0.19 022	11.58 9$\overline{96}$	86.282
Protoactinium.	Pa	**231**	5	0.59 8$\overline{45}$	2.08 $\overline{874}$	1.72 $\overline{352}$	0.58 0$\overline{21}$	3.79 $\overline{972}$	263.$\overline{177}$
			3	0.79 $\overline{793}$	1.25 $\overline{325}$	2.87 $\overline{254}$	0.34 8$\overline{12}$	6.33 $\overline{286}$	157.$\overline{906}$
			2	1.19 $\overline{689}$	0.83 5$\overline{50}$	4.30 $\overline{881}$	0.23 2$\overline{08}$	9.49 $\overline{929}$	105.$\overline{271}$
			n	2.39 $\overline{378}$	0.41 7$\overline{75}$	8.61 $\overline{762}$	0.11 6$\overline{04}$	18.9$\overline{9}$ $\overline{859}$	52.6$\overline{35}$

Table 2. Electrochemical Equivalents of the Elements—(*Continued*)

1		2	3	4	5	6	7	8	9
Element	Symbol	Atomic weight	Val. or chg. val.	Mg. per coulomb	Coulombs per mg.	G. per amp.-hr.	Amp.-hr. per g.	Lb. per 1,000 amp.-hr.	Amp.-hr. per lb.
Radium...	Ra	**226.05**	2	1.17 12$\overline{4}$	0.85 379	4.21 64$\overline{8}$	0.23 716	9.29 57$\overline{4}$	107.57$\overline{6}$
			n	2.34 24$\overline{9}$	0.42 690	8.43 29$\overline{5}$	0.11 858	18.59 $\overline{148}$	53.788
Radon.....	Rn	222	*n*	2.30 $\overline{052}$	0.43 4$\overline{68}$	8.28 $\overline{187}$	0.12 0$\overline{75}$	18.2$\overline{5}$ $\overline{839}$	54.7$\overline{69}$
Rhenium..	Re	**186.31**	7	0.27 581	3.62 56$\overline{8}$	0.99 292	1.00 71$\overline{3}$	2.18 90$\overline{1}$	456.82$\overline{8}$
			6	0.32 178	3.10 77$\overline{2}$	1.15 84$\overline{0}$	0.86 326	2.55 38$\overline{4}$	391.56$\overline{7}$
			5	0.38 613	2.58 97$\overline{7}$	1.39 00$\overline{8}$	0.71 938	3.06 46$\overline{1}$	326.30$\overline{6}$
			4	0.48 267	2.07 18$\overline{2}$	1.73 76$\overline{1}$	0.57 550	3.83 07$\overline{7}$	261.04$\overline{4}$
			3	0.64 356	1.55 38$\overline{6}$	2.31 68$\overline{1}$	0.43 163	5.10 76$\overline{9}$	195.78$\overline{3}$
			2	0.96 537	1.03 59$\overline{1}$	3.47 52$\overline{1}$	0.28 775	7.66 15$\overline{3}$	130.52$\overline{2}$
			1	1.93 06$\overline{7}$	0.51 795	6.95 04$\overline{2}$	0.14 388	15.32 3$\overline{06}$	65.261
Rhodium..	Rh	102.91	4	0.26 661	3.75 08$\overline{5}$	0.95 978	1.04 19$\overline{0}$	2.11 59$\overline{6}$	472.59$\overline{9}$
			3	0.35 547	2.81 31$\overline{4}$	1.27 97$\overline{1}$	0.78 143	2.82 12$\overline{8}$	354.44$\overline{9}$
			2	0.53 321	1.87 54$\overline{3}$	1.91 95$\overline{6}$	0.52 095	4.23 19$\overline{2}$	236.30$\overline{0}$
			1	1.06 64$\overline{2}$	0.93 771	3.83 91$\overline{3}$	0.26 048	8.46 38$\overline{3}$	118.15$\overline{0}$
Rubidium.	Rb	**85.48**	1	0.88 58$\overline{0}$	1.12 8$\overline{92}$	3.18 8$\overline{89}$	0.31 35$\overline{9}$	7.03 0$\overline{30}$	142.24$\overline{1}$
Ruthenium	Ru	101.7	8	0.13 17$\overline{4}$	7.59 0$\overline{95}$	0.47 42$\overline{5}$	2.10 8$\overline{60}$	1.04 $\overline{554}$	856.4$\overline{44}$
			6	0.17 56$\overline{5}$	5.69 3$\overline{22}$	0.63 23$\overline{3}$	1.58 $\overline{145}$	1.39 4$\overline{05}$	717.3$\overline{33}$
			5	0.21 07$\overline{8}$	4.74 4$\overline{35}$	0.75 88$\overline{0}$	1.31 7$\overline{87}$	1.67 2$\overline{86}$	597.7$\overline{78}$
			4	0.26 34$\overline{7}$	3.79 5$\overline{48}$	0.94 85$\overline{0}$	1.05 4$\overline{30}$	2.09 1$\overline{08}$	478.2$\overline{22}$
			3	0.35 13$\overline{0}$	2.84 6$\overline{61}$	1.26 4$\overline{66}$	0.79 07$\overline{2}$	2.78 8$\overline{10}$	358.6$\overline{67}$
			2	0.52 69$\overline{4}$	1.89 $\overline{774}$	1.89 6$\overline{99}$	0.52 71$\overline{5}$	4.18 2$\overline{16}$	239.1$\overline{11}$
			1	1.05 3$\overline{89}$	0.94 88$\overline{7}$	3.79 3$\overline{99}$	0.26 35$\overline{7}$	8.36 4$\overline{31}$	119.5$\overline{56}$
Samarium.	Sm	150.43	3	0.51 962	1.92 44$\overline{8}$	1.87 06$\overline{3}$	0.53 458	4.12 40$\overline{4}$	242.48$\overline{1}$
			n	1.55 88$\overline{6}$	0.64 149	5.61 19$\overline{0}$	0.17 819	12.37 2$\overline{11}$	80.827
Scandium..	Sc	45.10	3	0.15 57$\overline{9}$	6.41 9$\overline{07}$	0.56 08$\overline{3}$	1.78 3$\overline{07}$	1.23 64$\overline{2}$	808.7$\overline{89}$
			n	0.46 73$\overline{6}$	2.13 9$\overline{69}$	1.68 24$\overline{9}$	0.59 43$\overline{6}$	3.70 9$\overline{25}$	269.5$\overline{96}$
Selenium..	Se	**78.9**	6	0.13 63$\overline{7}$	7.33 2$\overline{83}$	0.49 09$\overline{4}$	2.03 6$\overline{90}$	1.08 2$\overline{34}$	923.9$\overline{21}$
			4	0.20 45$\overline{6}$	4.88 8$\overline{55}$	0.73 64$\overline{1}$	1.35 7$\overline{93}$	1.62 3$\overline{52}$	615.94$\overline{7}$
			2	0.40 91$\overline{2}$	2.44 4$\overline{28}$	1.47 2$\overline{83}$	0.67 89$\overline{7}$	3.24 7$\overline{03}$	307.9$\overline{74}$
			n	0.81 82$\overline{4}$	1.22 2$\overline{14}$	2.94 5$\overline{66}$	0.33 94$\overline{8}$	6.49 4$\overline{06}$	153.98$\overline{7}$
Silicon....	Si	28.06	4	0.07 269	13.75 $\overline{624}$	0.26 17$\overline{0}$	3.82 1$\overline{18}$	0.57 69$\overline{5}$	1733.$\overline{257}$
			n	0.29 07$\overline{8}$	3.43 9$\overline{06}$	1.04 6$\overline{80}$	0.95 52$\overline{9}$	2.30 7$\overline{79}$	433.3$\overline{14}$
Silver.....	Ag	107.880	1	1.11 793‡	0.89 451	4.02 454	0.24 848	8.87 259	112.707
Sodium....	Na	22.997	1	0.23 831	4.19 62$\overline{0}$	0.85 792	1.16 56$\overline{1}$	1.89 13$\overline{9}$	528.71$\overline{2}$
Strontium	Sr	87.63	2	0.45 40$\overline{4}$	2.20 2$\overline{44}$	1.63 4$\overline{55}$	0.61 17$\overline{9}$	3.60 3$\overline{56}$	277.5$\overline{03}$
			n	0.90 80$\overline{8}$	1.10 1$\overline{22}$	3.26 9$\overline{10}$	0.30 58$\overline{9}$	7.20 7$\overline{13}$	138.7$\overline{52}$

Table 2. Electrochemical Equivalents of the Elements—(*Continued*)

1		2	3	4	5	6	7	8	9
Element	Symbol	Atomic weight	Val. or chg. val.	Mg. per coulomb	Coulombs per mg.	G. per amp.-hr.	Amp.-hr. per g.	Lb. per 1,000 amp.-hr.	Amp.-hr. per lb.
Sulphur....	S	32.06	7	0.04 746	21.06 98$\overline{7}$	0.17 08$\overline{6}$	5.85 27$\overline{4}$	0.37 66$\overline{8}$	2654.75$\overline{9}$
			6	0.05 537	18.05 98$\overline{9}$	0.19 93$\overline{4}$	5.01 66$\overline{4}$	0.43 94$\overline{6}$	2275.50$\overline{8}$
			5	0.06 645	15.04 99$\overline{1}$	0.23 92$\overline{0}$	4.18 05$\overline{3}$	0.52 73$\overline{6}$	1896.25$\overline{7}$
			4	0.08 306	12.03 99$\overline{3}$	0.29 90$\overline{1}$	3.34 44$\overline{2}$	0.65 91$\overline{9}$	1517.00$\overline{5}$
			3	0.11 07$\overline{4}$	9.02 99$\overline{4}$	0.39 86$\overline{7}$	2.50 83$\overline{2}$	0.87 89$\overline{2}$	1137.75$\overline{4}$
			2	0.16 61$\overline{1}$	6.01 99$\overline{6}$	0.59 80$\overline{1}$	1.67 22$\overline{1}$	1.31 83$\overline{9}$	758.50$\overline{3}$
			1	0.33 22$\overline{3}$	3.00 99$\overline{8}$	1.19 60$\overline{2}$	0.83 61$\overline{1}$	2.63 67$\overline{7}$	379.25$\overline{1}$
Tantalum..	Ta	**180.88**	5	0.37 488	2.66 75$\overline{1}$	1.34 95$\overline{7}$	0.74 098	2.97 52$\overline{9}$	336.10$\overline{1}$
			4	0.46 860	2.13 40$\overline{1}$	1.68 69$\overline{6}$	0.59 278	3.71 91$\overline{2}$	268.88$\overline{1}$
			3	0.62 480	1.60 05$\overline{1}$	2.24 92$\overline{8}$	0.44 459	4.95 88$\overline{2}$	201.66$\overline{1}$
			2	0.93 720	1.06 70$\overline{1}$	3.37 39$\overline{3}$	0.29 639	7.43 82$\overline{4}$	134.44$\overline{0}$
			1	1.87 44$\overline{0}$	0.53 350	6.74 78$\overline{5}$	0.14 820	14.87 64$\overline{7}$	67.220
Tellurium .	Te	**127.61**	6	0.22 040	4.53 72$\overline{6}$	0.79 343	1.26 03$\overline{7}$	1.74 92$\overline{1}$	571.68$\overline{6}$
			4	0.33 060	3.02 48$\overline{4}$	1.19 01$\overline{5}$	0.84 023	2.62 38$\overline{2}$	381.12$\overline{4}$
			2	0.66 119	1.51 24$\overline{2}$	2.38 02$\overline{9}$	0.42 012	5.24 76$\overline{4}$	190.56$\overline{2}$
			n	1.32 23$\overline{8}$	0.75 621	4.76 05$\overline{8}$	0.21 006	10.49 52$\overline{8}$	95.281
Terbium...	Tb	159.2	3	0.54 99$\overline{1}$	1.81 84$\overline{7}$	1.97 96$\overline{9}$	0.50 51$\overline{3}$	4.36 44$\overline{7}$	229.11$\overline{3}$
			n	1.64 97$\overline{4}$	0.60 61$\overline{6}$	5.93 90$\overline{7}$	0.16 83$\overline{8}$	13.09 34$\overline{0}$	76.37$\overline{1}$
Thallium..	Tl	204.39	3	0.70 601	1.41 64$\overline{1}$	2.54 16$\overline{4}$	0.39 345	5.59 00$\overline{2}$	178.57$\overline{1}$
			1	2.11 80$\overline{3}$	0.47 214	7.62 49$\overline{1}$	0.13 115	16.80 00$\overline{5}$	59.524
Thorium...	Th	232.12	4	0.60 135	1.66 29$\overline{3}$	2.16 48$\overline{5}$	0.46 193	4.77 26$\overline{8}$	209.52$\overline{6}$
			n	2.40 53$\overline{9}$	0.41 573	8.65 94$\overline{0}$	0.11 548	19.09 07$\overline{0}$	52.382
Thulium...	Tm	169.4	3	0.58 51$\overline{5}$	1.70 89$\overline{7}$	2.10 65$\overline{3}$	0.47 47$\overline{1}$	4.64 41$\overline{0}$	215.32$\overline{7}$
			n	1.75 54$\overline{4}$	0.56 96$\overline{6}$	6.31 95$\overline{9}$	0.15 82$\overline{4}$	13.93 23$\overline{0}$	71.77$\overline{6}$
Tin........	Sn	118.70	4	0.30 751	3.25 19$\overline{0}$	1.10 70$\overline{5}$	0.90 330	2.44 06$\overline{2}$	409.73$\overline{2}$
			2	0.61 503	1.62 59$\overline{5}$	2.21 40$\overline{9}$	0.45 165	4.88 12$\overline{4}$	204.86$\overline{6}$
			n	1.23 00$\overline{5}$	0.81 297	4.42 81$\overline{9}$	0.22 583	9.76 24$\overline{8}$	102.43$\overline{3}$
Titanium.	Ti	47.90	4	0.12 40$\overline{9}$	8.05 84$\overline{6}$	0.44 67$\overline{4}$	2.23 84$\overline{6}$	0.98 48$\overline{8}$	1015.34$\overline{8}$
			3	0.16 54$\overline{6}$	6.04 38$\overline{4}$	0.59 56$\overline{5}$	1.67 88$\overline{4}$	1.31 31$\overline{8}$	761.51$\overline{1}$
			1	0.49 63$\overline{7}$	2.01 46$\overline{1}$	1.78 69$\overline{4}$	0.55 96$\overline{1}$	3.93 95$\overline{3}$	353.83$\overline{7}$
Tungsten..	W	183.92	6	0.31 765	3.14 81$\overline{1}$	1.14 35$\overline{4}$	0.87 447	2.52 10$\overline{8}$	396.65$\overline{5}$
			5	0.38 118	2.62 34$\overline{2}$	1.37 22$\overline{5}$	0.72 873	3.02 53$\overline{0}$	330.54$\overline{6}$
			4	0.47 648	2.09 87$\overline{4}$	1.71 53$\overline{2}$	0.58 298	3.78 16$\overline{2}$	264.43$\overline{7}$
			3	0.63 530	1.57 40$\overline{5}$	2.28 70$\overline{9}$	0.43 724	5.04 21$\overline{7}$	198.32$\overline{7}$
			2	0.95 295	1.04 93$\overline{7}$	3.43 06$\overline{3}$	0.29 149	7.56 32$\overline{5}$	132.21$\overline{8}$
			1	1.90 59$\overline{1}$	0.52 468	6.86 12$\overline{6}$	0.14 575	15.12 65$\overline{0}$	66.109
Uranium ..	U	**238.07**	6	0.41 117	2.43 20$\overline{6}$	1.48 02$\overline{3}$	0.67 557	3.26 16$\overline{8}$	306.59$\overline{1}$
			5	0.49 341	2.02 67$\overline{1}$	1.77 62$\overline{7}$	0.56 298	3.91 60$\overline{1}$	255.49$\overline{2}$
			4	0.61 676	1.62 13$\overline{7}$	2.22 03$\overline{4}$	0.45 038	4.89 25$\overline{2}$	204.39$\overline{4}$
			3	0.82 235	1.21 60$\overline{3}$	2.96 04$\overline{6}$	0.33 779	6.52 33$\overline{5}$	153.29$\overline{5}$
			2	1.23 35$\overline{2}$	0.81 069	4.44 06$\overline{8}$	0.22 519	9.78 50$\overline{3}$	102.19$\overline{7}$
			1	2.46 70$\overline{5}$	0.40 534	8.88 13$\overline{7}$	0.11 260	19.57 00$\overline{6}$	51.098

Table 2. Electrochemical Equivalents of the Elements—(*Concluded*)

1		2	3	4	5	6	7	8	9
Element	Symbol	Atomic weight	Val. or chg. val.	Mg. per coulomb	Coulombs per mg.	G. per amp.-hr.	Amp.-hr. per g.	Lb. per 1.000 amp.-hr.	Amp.-hr. per lb.
Vanadium.	V	**50.95**	5	$0.10\ 56\overline{0}$	$9.47\ \overline{007}$	$0.38\ 01\overline{5}$	$2.63\ 0\overline{57}$	$0.83\ 80\overline{8}$	$1193.\overline{209}$
			4	$0.13\ 19\overline{9}$	$7.57\ \overline{606}$	$0.47\ 51\overline{8}$	$2.10\ \overline{446}$	$1.04\ 7\overline{60}$	$954.56\overline{7}$
			3	$0.17\ 59\overline{9}$	$5.68\ \overline{204}$	$0.63\ 35\overline{8}$	$1.57\ 8\overline{34}$	$1.39\ 6\overline{79}$	$715.9\overline{25}$
			2	$0.26\ 39\overline{9}$	$3.78\ \overline{803}$	$0.95\ 03\overline{6}$	$1.05\ 2\overline{23}$	$2.09\ 5\overline{19}$	$477.2\overline{84}$
			1	$0.52\ 79\overline{8}$	$1.89\ \overline{401}$	$1.90\ 0\overline{73}$	$0.52\ 61\overline{1}$	$4.19\ 0\overline{38}$	$238.6\overline{42}$
Virginium .	Vi	**224***	1	$2.32\ \overline{124}$	$0.43\ 0\overline{80}$	$8.35\ \overline{648}$	$0.11\ 96\overline{7}$	$18.4\overline{2}\ \overline{288}$	$54.2\overline{80}$
Xenon.....	Xe	**131.3**	n	$1.36\ 0\overline{62}$	$0.73\ 49\overline{6}$	$4.89\ 8\overline{24}$	$0.20\ 41\overline{6}$	$10.79\ \overline{877}$	$92.60\overline{3}$
Ytterbium.	Yb	**173.04**	3	$0.59\ 772$	$1.67\ 30\overline{2}$	$2.15\ 17\overline{9}$	$0.46\ 473$	$4.74\ 38\overline{9}$	$210.79\overline{7}$
			n	$1.79\ 31\overline{6}$	$0.55\ 767$	$6.45\ 53\overline{8}$	$0.15\ 491$	$14.23\ 1\overline{67}$	70.266
Yttrium...	Y	88.92	3	$0.30\ 71\overline{3}$	$3.25\ 5\overline{74}$	$1.10\ 5\overline{74}$	$0.90\ 43\overline{7}$	$2.43\ 7\overline{74}$	$410.2\overline{16}$
			n	$0.92\ 14\overline{5}$	$1.08\ 5\overline{25}$	$3.31\ 7\overline{22}$	$0.30\ 14\overline{6}$	$7.31\ 3\overline{22}$	$136.7\overline{39}$
Zinc.......	Zn	65.38	2	$0.33\ 87\overline{6}$	$2.95\ \overline{197}$	$1.21\ 9\overline{52}$	$0.81\ 99\overline{9}$	$2.68\ 8\overline{59}$	$371.9\overline{42}$
			n	$0.67\ 75\overline{1}$	$1.47\ 5\overline{99}$	$2.43\ 9\overline{05}$	$0.41\ 00\overline{0}$	$5.37\ 7\overline{18}$	$185.9\overline{71}$
Zirconium.	Zr	91.22	4	$0.23\ 63\overline{2}$	$4.23\ \overline{153}$	$0.85\ 07\overline{6}$	$1.17\ 5\overline{42}$	$1.87\ 5\overline{60}$	$533.1\overline{64}$
			n	$0.94\ 52\overline{8}$	$1.05\ 7\overline{88}$	$3.40\ 3\overline{03}$	$0.29\ 38\overline{6}$	$7.50\ 2\overline{39}$	$133.2\overline{91}$

NOTE.—Atomic weights in boldface type indicate those in which changes have been made since the last revision of this table in 1929, or new additions to the list since that time.

Digits overscored may, if desired, be dropped from the values, rounding them off to the nearest preceding digit; such digits have been carried as a matter of convenience and uniformity in calculating and tabulating but are in excess of the number of significant figures in the primary data and hence do not add to the true accuracy of the results.

* Best value known; not included in the official list.

† This is the second isotope of hydrogen and is the only isotope included in the table, as no others have as yet been isolated to a sufficient degree to have their atomic weights determined.

‡ This value varies from the basic figure of 1.1180 mg. because of the rounding off of the value of the Faraday to 96,500 coulombs; other values also differ in the same proportion.

atomic weight divided by its valence) be divided by the electrochemical equivalent of silver, we find that

$$107.88 \div 0.00111800 = 96{,}494 \text{ coulombs}$$

This quantity of electricity is called a **faraday** (F). It is taken as 96,500 coulombs or 26.8 amp.-hr. If 1 faraday be passed through an electrolytic conductor, 1 g. equivalent of some substance will be liberated at each electrode. If 1 faraday be passed through a solution containing several electrolytes, the summation of the quantities of the different products set free at each electrode, when expressed in equivalents, will be unity.

In practice more than 1 faraday is needed for the liberation of a gram equivalent of a substance. This is not due to the failure of Faraday's laws but to other causes. Side reactions may take place. The products of the electrolysis may suffer mechanical loss. Secondary reactions may take place at the electrodes. In addition there may be current leaks, short circuits, and losses in the form of heat. The ratio of the theoretical to the actual

Table 3. Atomic Numbers, Atomic Weights, and Isotopes of the Elements

Atomic No.	Symbol	Mean atomic weight	Known isotopes	Atomic weight of isotopes
1	H	1.0078	3	1-2-3
2	He	4.002	2	4
3	Li	6.94	2	6-7
4	Be	9.02	2	(8)-9
5	B	10.82	2	10-11
6	C	12.01	2	12-13
7	N	14.008	2	14-15
8	O	16.0000	3	16-17-18
9	F	19.00	1	19
10	Ne	20.183	3	20-21-22
11	Na	22.997	3	23(1L-1G)
12	Mg	24.32	3	24-25-26
13	Al	26.97	1	27
14	Si	28.06	3	28-29-30
15	P	31.02	1	31
16	S	32.06	3	32-33-34
17	Cl	35.497	3	35-37-39
18	A	39.944	3	36-38-40
19	K	39.096	3	39-40-41
20	Ca	40.08	6	40-42-43-44-46-48
21	Sc	45.10	1	45
22	Ti	47.90	5	46-47-48-49-50
23	V	50.95	1	51
24	Cr	52.01	4	50-52-53-54
25	Mn	54.93	7	55(3L-3G)
26	Fe	55.84	4	54-56-57-58
27	Co	58.94	2	57-59
28	Ni	58.69	5	58-60-62-64
29	Cu	63.57	2	63-65
30	Zn	65.38	5	64-66-67-68-70
31	Ga	69.72	2	69-71
32	Ge	72.60	5	70-72-73-74-76
33	As	74.91	1	75
34	Se	78.96	6	74-76-77-78-80-82
35	Br	79.916	2	79-81
36	Kr	83.7	6	78-80-82-83-84-86
37	Rb	85.48	2	85-87
38	Sr	87.63	4	84-86-87-88
39	Y	88.92	1	89
40	Zr	91.22	5	90-91-92-94-96
41	Cb	92.91	1	93
42	Mo	96.0	8	92-94-95-96-97-98-100-(102)
43	Ma	97.8		
44	Ru	101.7	7	96-99-100-101-102-104
45	Rh	102.91	2	101-103
46	Pd	106.7	6	102-104-105-106-108-110
47	Ag	107.880	2	107-109
48	Cd	112.41	9	106-108-110-111-112-113-114-116
49	In	114.76	2	113-115
50	Sn	118.70	10	112-114-115-116-117-118-119-120-(121)-122-124
51	Sb	121.76	2	121-123
52	Te	127.61	8	120-122-123-124-125-126-128-130
53	I	126.92	1	127
54	Xe	131.3	9	124-126-128-129-130-131-132-134-136
55	Cs	132.91	5	133(2L-2G)
56	Ba	137.36	7	130-132-134-135-136-137-138
57	La	138.92	1	139

Table 3. Atomic Numbers, Atomic Weights, and Isotopes of the Elements—(*Concluded*)

Atomic No.	Symbol	Mean atomic weight	Known isotopes	Atomic weight of isotopes
58	Ce	140.13	4	136-138-140-142
59	Pr	140.92	1	141
60	Nd	144.27	5	142-143-144-145-146
61	Il	146.		
62	Sm	150.43	7	144-147-148-149-150-152-154
63	Eu	152.0	2	151-153
64	Gd	156.9	5	155-156-157-158-160
65	Tb	159.2	1	159
66	Dy	162.46	4	161-162-163-164
67	Ho	163.5	1	165
68	Er	167.64	4	166-167-168-170
69	Tm	169.4	1	169
70	Yb	173.04	5	171-172-173-174-176
71	Lu	175.0	1	175
72	Hf	178.6	5	176-177-178-179-180
73	Ta	180.88	1	181
74	W	184.0	4	182-183-184-186
75	Re	186.31	2	185-187
76	Os	191.5	6	186-187-188-189-190-192
77	Ir	193.1	2	191-193
78	Pt	195.23	5	192-194-195-196-198
79	Au	197.2		
80	Hg	200.61	8	196-198-199-200-201-202-203-204
81	Tl	204.39	8	201-203-205-207-209-211-213-215
82	Pb	207.21	16	201-202-203-204-205-206-207-208-209-210-211-212-213-214-215-216
83	Bi	209.00	14	205-206-207-208-209-210-211-212-213-214-215-216-217-219
84	Po	210.		
85	Ab	221.		
86	Rn	222.		
87	Vi	224.		
88	Ra	226.05	4	226-228-230-232
89	Ac	227.		
90	Th	232.12	8	229-230-231-232-233-234-235-236
91	Pa	231.		
92	U	238.07	8	233-234-235-236-237-238-239-240

Note.—Parentheses indicate an uncertain value.

quantity of current used is the **current efficiency**. In a similar manner, the actual amount of a product formed from a definite amount of current, divided by the theoretical amount, also gives us the current efficiency. In commercial practice, current efficiencies may vary from as low as 25 to 30 per cent in the decomposition of certain fused salts and in chromium plating to as high as 92 to 95 per cent in copper refining and 100 per cent in the electrolytic oxidation of anthracene to anthraquinone.

The **current concentration** is the current divided by the volume upon which the current acts. If a high concentration is to be produced of some compound subject to chemical decomposition, a high current concentration is desirable.

Electrochemical Equivalents

The electrochemical equivalents of the elements are tabulated. The use of the table, Table 2, has been simplified in that not only the ordinary valences of the elements are included but also the changes of valence which occur in oxidation-reduction reactions. For example, while iron exhibits the valence

Table 4. Conversion Data for Metric and Avoirdupois Units

	1	2	3	4	5	6	7	8	
	Mg. per sec. ↓ ÷	G. per hr. ↓ ÷	Kg. per day ↓ ÷	Metric tons per yr. ↓ ÷	Oz. per hr. ↓ ÷	Lb. per hr. ↓ ÷	Lb. per day ↓ ÷	Net tons per yr. ↓ ÷	
1. Mg. per sec. → ×	1.	3.6	0.0864	0.031557	0.12699	0.0079367	0.19048	0.034786	÷ ← Sec. per mg.
2. G. per hr. → ×	0.27778	1.	0.024	0.0087658	0.035274	0.0022046	0.052911	0.0096626	÷ ← Hr. per g.
3. Kg. per day → ×	11.57407	41.66667	1.	0.36524	1.46975	0.091860	2.20462	0.40261	÷ ← Days per kg.
4. Metric tons per yr. → ×	31.68877	114.07955	2.73791	1.	4.02404	0.25150	6.03606	1.10231	÷ ← Yr. per metric ton
5. Oz. per hr. → ×	7.87487	28.34953	0.68039	0.24851	1.	0.0625	1.5	0.27393	÷ ← Hr. per oz.
6. Lb. per hr. → ×	125.99790	453.59243	10.88622	3.97611	16.	1.	24.	4.38291	÷ ← Hr. per lb.
7. Lb. per day → ×	5.24991	18.89969	0.45359	0.16567	0.66667	0.041667	1.	0.18262	÷ ← Days per lb.
8. Net tons per yr. → ×	28.74767	103.49124	2.48379	0.90719	3.65051	0.22816	5.47582	1.	÷ ← Yr. per net ton
	Sec. per mg. × ↑	Hr. per g. × ↑	Days per kg. × ↑	Yr. per metric ton × ↑	Hr. per oz. × ↑	Hr. per lb. × ↑	Days per lb. × ↑	Yr. per net ton × ↑	

Table 5. Conversion Data for Metric and Troy Units

	1	2	3	4	5	6	7	
	M. per sec. ↓ ÷	G. per hr. ↓ ÷	Kg. per day ↓ ÷	Metric tons per yr. ↓ ÷	Oz. per hr. ↓ ÷	Oz. per day ↓ ÷	1000 oz. per yr. ↓ ÷	
1. Mg. per sec. → ×	1.	3.6	0.0864	0.031557	0.11574	2.77778	1.01458	÷ ← Sec. per mg.
2. G. per hr. → ×	0.27778	1.	0.024	0.008766	0.032151	0.77162	0.28183	÷ ← Hr. per g.
3. Kg. per day → ×	11.57407	41.66667	1.	0.36524	1.33941	32.15072	11.74281	÷ ← Days per kg.
4. Metric tons per yr. → ×	31.68877	114.07955	2.73791	1.	3.66774	88.02578	32.15072	÷ ← Yr. per metric ton
5. Oz. per hr. → ×	8.63986	31.10359	0.74649	0.27265	1.	24.	8.76581	÷ ← Hr. per oz.
6. Oz. per day → ×	0.36	1.29598	0.031104	0.011360	0.041667	1.	0.365224	÷ ← Days per oz.
7. 1000 oz. per yr. → ×	0.98563	3.54823	0.085157	0.031104	0.11408	2.73791	1.	÷ ← Yr. per 1000 oz.
	Sec. per mg. × ↑	Hr. per g. × ↑	Days per kg. × ↑	Yr. per metric ton × ↑	Hr. per oz. × ↑	Days per oz. × ↑	Yr. per 1000 oz. × ↑	

of 2 or 3, in the reduction of ferric to ferrous iron there is a valence change of 1. In Table 2, valence changes of 1, when they are not a normal valence or normal valence change of the element, are indicated as n. The calculations in Table 2 are based on the 1938 atomic weights, being the work of Roush [*Trans. Am. Electrochem. Soc.*, **73**, 285 (1938)]. Mass and energy conversion factors allowing greater ease of use of the electrochemical equivalent tables are given in Tables 4, 5, and 6.

Table 6. Energy Conversion Data

$$\text{Kg. per hp.-hr.} = \frac{\text{g. per amp.-hr.} \times 0.7465}{\text{voltage of the reaction}}$$

$$\text{Hp.-hr. per kg.} = \text{amp.-hr. per g.} \times 1.3411 \times \text{voltage}$$

$$\text{Lb. per hp.-hr.} = \frac{\text{g. per amp.-hr.} \times 1.6457}{\text{voltage of the reaction}}$$

$$\text{Hp.-hr. per lb.} = \text{amp.-hr. per g.} \times 0.60786 \times \text{voltage}$$

$$\text{Kg. per hp.-yr.} = \frac{\text{g. per amp.-hr.} \times 6543.8}{\text{voltage of the reaction}}$$

$$\text{Lb. per hp.-yr.} = \frac{\text{g. per amp.-hr.} \times 14{,}426.5}{\text{voltage of the reaction}}$$

Coulometers. Galvanometers are used for the *detection* of current and occasionally for the *measurement* of small currents. For large currents, ammeters are generally used and time recorded. Coulometers find application for the measurement of current (in coulombs) in small-scale experimental work. Measurement is made of the electrode products due to the passage of the current. From Faraday's laws, the quantity of electricity passed through can be calculated. Coulometers depend upon operation under 100 per cent current efficiency. Side reactions must be eliminated.

Coulometers are of several classes. In the **weight coulometer** the gain in weight of the cathode of an electrolytic cell, due to the deposition of metal from a solution of its salt, is measured. **Volume coulometers** are those in which the volume of a gas liberated as the result of electrolysis, or the volume of mercury set free during electrolysis of a suitable mercury salt, is measured. In **titration coulometers** the change in concentration or the amount of a substance set free at one of the electrodes is determined by analytical methods.

The errors of coulometers are those inherent in the measurement of weight and volume or in titration, and also those due to imperfections in the coulometer itself. The silver-weight and the iodine-titration coulometers are the most accurate, partly because of the high equivalent weights of iodine and silver (1 g. Ag corresponds to 894.53, 1 g. I to 760.33 coulombs).

The silver coulometer has been much studied. On the accuracy of the measurement of quantities of electricity depend not only the value of the faraday and the definition of the ampere but also the value of the e.m.f. of the normal Weston cell which is employed as a universal standard of e.m.f. The errors in the silver coulometer have been so completely eliminated that the results obtained for the e.m.f. of this cell by investigators in several countries, using three types of the coulometer, agree to about 1 part in 100,000. [See Rosa and Vinal, *Bur. Standards Bull.*, **13**, 479 (1916); *Natl. Bur. Standards, Sci. Paper* 285.]

Electrolytic Dissociation

In dilute solutions of nonelectrolytes the gas law equation

$$PV = nRT$$

applies to osmotic pressure. There, P is the osmotic pressure, V the volume of the solution, n the number of mols in the solution, R the gas constant, and T the absolute temperature. $R = 0.0821$ l.-atm. per °K (the absolute temperature scale) or 1.985 cal. per deg. For electrolytes, P is always greater than the value calculated. For electrolytes the expression becomes

$$PV = inRT$$

where i, always greater than unity, represents a degree of abnormality.

The presence of a solute causes a depression of the freezing point of a solvent. For nonelectrolytes the extent of this lowering is given by the expression

$$\Delta = K\frac{n}{N}$$

where Δ is the lowering of the freezing point, K is a constant, n is the number of formula weights of solute present in N formula weights of solvent. The value becomes 1.858 °K. per g. mol solute per liter. Electrolytes always give values which exceed 1.858.

In order to explain this behavior of electrolytes, Arrhenius in 1887 formulated a theory of **electrolytic dissociation.** It was assumed that the molecules of electrolytes break up into equivalent quantities of positively and negatively electrified particles or ions when dissolved in water. The solutions of electrolytes are electrically neutral. The abnormal osmotic effects produced by electrolytes may then be accounted for by the increase in the number of particles of solute present in a solution. The theory does not assume that all of the molecules in solution are dissociated. Let α equal the degree of dissociation or the fraction of each formula weight dissociated into ions, and n the number of ions into which each molecule dissociates, then

$$\begin{aligned} 1 &= (1 - \alpha) + n\alpha \\ &= 1 + (n - 1)\alpha \end{aligned}$$

$$\alpha = \frac{(i - 1)}{(n - 1)}$$

Thus α may be calculated from osmotic pressure or from freezing-point determinations.

When n is the valence of the ion, the quantity of electricity carried by any gram ion is nF. One gram ion contains Avogadro's number of ions, which is 6.06×10^{23}. A single ion must carry the charge equivalent to the amount carried by the gram ion divided by the number of ions present, or a simple multiple n of this quantity if the ion have a valence of more than 1. This ultimate quantity of negative electricity is called the **electron.** It amounts to

$$\frac{96{,}500}{(6.06 \times 10^{23})} = 1.59 \times 10^{-19} \text{ coulomb}$$

Solutions of electrolytes in solvents other than water conduct the electric current. It may be inferred that electrolytic dissociation takes place in these solvents. Substances which show conduction of the electric current in nonaqueous solutions are not necessarily dissociated in water. This solvent, however, is more effective in bringing about dissociation than almost all others. Molten salts exhibit the same phenomena as solutions of electrolytes.

Solvents with high dielectric constants, like water, possess a high dissociating power, while those with low dielectric constants dissociate dissolved material to a less degree. The attraction of electric charges for each other is inversely proportional to the dielectric constant of the surrounding medium.

In the newer theories, the electric charges upon the ions are assumed to set up electrostatic fields which do not allow the ions to behave independently as demanded by the gas laws. The anomaly of strong electrolytes is to be attributed entirely to these interionic attractions.

Debye and Hückel [*Physik. Z.*, **24**, 185, 344 (1923); *Ergebn. d. exakt. Naturwiss.*, **3**, 199 (1925); *Physik. Z.*, **25**, 97 (1924)] developed a theory that, owing to electrical attractions, an ion of a given sign will, on the average, be surrounded by more ions of unlike sign than by ions of like sign. When such a solution is diluted to a very large volume, the ions become separated to such an extent that their mutual electrostatic attractions are no longer of significance. If the excess electrical work involved in this isothermal dilution due to the rearrangement of the relative position of the ions be determined, the extent of the deviation from the gas laws in terms of the activity coefficient of the ions can be calculated.

Conductivity. In industrial operations, owing to the resistance of the electrolyte, certain amounts of electrical energy are converted into heat. From a practical viewpoint the resistance of the electrolyte is important in that it represents one of the ways in which electrical energy is consumed.

The resistance of any conductor of uniform cross section is

$$R = r\left(\frac{l}{a}\right)$$

where l is the length, a the cross-sectional area, r the specific resistance, and R the total resistance. The **specific resistance** is defined as the resistance of a unit cube of a conductor of a given material, expressed in ohms per centimeter cube or ohms per inch cube. The reciprocal of resistance is conductivity, and the reciprocal of specific resistance is the **specific conductivity**, denoted by K which may be defined by the expression

$$\mathrm{K} = \frac{1}{r} = \frac{l}{Ra} = \frac{lI}{Ea}$$

where I is the current and E the potential drop between the electrodes.

Conductivity of a solution is a function of the nature of the electrolyte, the solvent, the concentration, and the temperature. Specific conductivity varies directly with concentration up to a maximum point, after which there is a decrease. The $\mathrm{K} - t$ relation

$$\mathrm{K}_t = \mathrm{K}_{18}[1 + b(t - 18)]$$

is almost linear, where b is 0.02 to 0.025 for salts and bases, and 0.01 to 0.016 for acids.

The **equivalent conductivity** (Λ) equals $\mathrm{K}V$, where V is the volume of the solution containing 1 g. equivalent of the solute. Λ varies directly with V. K decreases with dilution except in very concentrated solutions, while Λ increases. At first the change in value is rapid, gradually diminishing until, at sufficiently high dilutions, a practically constant maximum termed Λ_∞, or equivalent conductivity at infinite dilution, is reached. It can be shown that $\Lambda_v/\Lambda_\infty = \alpha$, giving an electrical method for determination of the degree of ionization at any dilution. α increases with dilution, approaching 1 as the limit. Equivalent conductivities of a number of inorganic acids, bases, and salts are shown in Table 7. The values for the concentrations of 0.001 N are in many cases equal to or approximately the same as Λ_∞, so that α values may be calculated from the tables. From the relations $\Lambda_v = \mathrm{K}V$

Table 7. Equivalent Conductivities of Salts, Acids, and Bases

Salts	°C.	0.001 *N*	0.01 *N*	0.1 *N*	0.2 *N*	0.5 *N*	1 *N*	2 *N*	3 *N*	4 *N*	5 *N*	6 *N*	7 *N*	10 *N*	20 *N*	Reference
$AgNO_3$	..	113.0	107.62	94.2	87.8	77.3	67.5	55.9	48.4	42.3						1,2,3
Ag_2SO_4	..	116.8	103.3													4,5
$AlCl_3$	..					65.0	56.2	44.5	34.2	27.1						1
$Al(NO_3)_3$	25	125.	108.	87.9	80.5	69.6										6
$Al_2(SO_4)_3$	25	107.0	67.3													7
$BaCl_2$	..	115.44	106.52	90.65	85.23	77.18	70.04	60.5	51.5							2,8
$Ba(NO_3)_2$	..	111.56	100.82	78.83	70.08	56.52										2,3,9
$CaCl_2$	..	111.8	103.23	88.07	82.68	74.82	67.45	58.0	49.7	42.4	35.6		23.5			9
$Ca(NO_3)_2$	..	108.34	99.39	82.37	75.84	65.61	55.79	42.7	33.5	26.5		14.2		5.		2,10
$CdCl_2$	..	104.8	82.9	50.0	41.0	29.6	21.6	14.1	10.	7.	4.8		3.	1.2		11,12,13
$Cd(NO_3)_2$	..	108.2	96.2	79.7	73.5	63.4	53.9	40.8	31.4	24.2	18.4	13.7	11.			12,13
$CdSO_4$	..	97.58	70.23	47.15	35.84	28.7	23.56	17.9	14.	10.8	8.2					2,12
$CoCl_2$	..						60.8	49.2	39.7	32.1			12.			1,14
$Co(NO_3)_2$	..					66.2	58.0	46.8	37.6	29.9	23.3					1
$CoSO_4$	25	113.4	82.5	51.4	43.9	35.3	29.3									6
$Cr(NO_3)_3$	..		108.6	80.2	74.0	64.8	55.9	43.2	33.3	25.5	19.					1,15
$CuCl_2$	..			83.2	77.1	67.0	56.9	43.4	33.6	26.0						8,16
$Cu(NO_3)_2$	..			81.9	76.0	67.2	57.8	45.5	36.3	28.4				4.6		16
$CuSO_4$	..	98.42	71.64	43.8	37.6	30.7	25.74	20.	16.2							2,10
$FeCl_3$	..					69.4	60.6	48.1	38.8	30.8	24.0	18.1				1
$FeSO_4$	..					30.8	25.8	19.5	15.3	11.9						17

[1] Heydweiller, *Z. anorg. allgem. Chem.*, **116**, 42 (1921).

[2] Kohlrausch and Grüneisen, *Sitzb. preuss. Akad. Wiss.*, p. 1215 (1904).

[3] Noyes and Melcher, *Carnegie Inst. Wash. Pub.* 63, p. 71 (1907).

[4] Drucker, *Z. physik. Chem.*, **96**, 381 (1920).

[5] Hunt, *J. Am. Chem. Soc.*, **33**, 795 (1911).

[6] Jones *et al.*, *Carnegie Inst. Wash. Pub.* 170 (1912).

[7] Walden, *Z. physik. Chem.*, **2**, 49 (1888).

[8] Clausen, *Ann. Physik*, **37**, 51 (1912).

[9] Noyes and Falk, *J. Am. Chem. Soc.*, **34**, 454 (1912).

[10] Kohlrausch and Holborn, "Das Leitvermögen der Elektrolyte," Teubner, Leipzig, 1916.

[11] Grotrian, *Ann. physik. Chem.*, **151**, 378 (1874).

[12] Grotrian, *Ann. physik. Chem.*, **18**, 177 (1883).

[13] Wershoven, *Z. physik. Chem.*, **5**, 481 (1890).

[14] Mazzetti, *Gazz. chim. ital.*, **54**, 891 (1924).

[15] Bjerrum, *Z. physik. Chem.*, **59**, 336 (1907).

[16] Heydweiller, *Ann. Physik*, **37**, 739 (1912).

[17] Klein, *Ann. physik. Chem.*, **27**, 151 (1886).

Table 7. Equivalent Conductivities of Salts, Acids, and Bases—(*Continued*)

Salts	°C.	0.001 *N*	0.01 *N*	0.1 *N*	0.2 *N*	0.5 *N*	1*N*	2*N*	3*N*	4 *N*	5 *N*	6 *N*	7 *N*	10 *N*	20 *N*	Reference
$Fe(NO_3)_3$	..	.....	.....	.....	.....	75.9	63.8	48.5	37.5	28.5	.....	.....	.....	....	....	1
$HgCl_2$	..	.....	2.6	1.5	.....	1.0	.....	.....	.....	.....	.....	.....	.....	....	....	12
KBr	..	129.1	124.13	113.98	110.17	105.06	101.2	96.5	91.9	86.9	.....	.....	.....	....	....	5,18,19,20,21
KCl	..	127.07	122.18	111.79	107.74	102.25	98.08	92.3	88.1	.....	.....	.....	.....	....	....	22,23,24
$KClO_3$	..	116.7	111.5	99.0	93.6	.....	.....	.....	.....	.....	.....	.....	.....	....	....	21,25
KF	..	108.45	103.85	93.63	89.1	82.1	75.7	.....	.....	.....	.....	.....	.....	....	....	9,18,26
KNO_3	..	123.37	117.93	104.56	98.53	89.12	80.33	69.0	61.3	.....	.....	.....	.....	....	....	9,22,25
K_2SO_4	..	126.7	115.6	94.8	87.8	78.5	71.5	.....	.....	.....	.....	.....	.....	....	....	2,3,9,17,27
LiBr	..	.....	.....	84.3	80.8	74.1	67.2	57.7	50.5	44.2	.....	.....	.....	....	....	19
LiCl	..	96.2	91.84	82.76	77.69	70.5	63.18	53.0	45.2	.....	33.3	.....	.....	11.3	....	22
$LiNO_3$	..	92.58	88.33	78.95	74.79	67.8	60.61	.....	.....	.....	.....	.....	.....	....	....	9,22
$MgCl_2$	0	.....	65.7	55.2	51.8	46.0	40.6	.....	.....	.....	.....	.....	.....	....	....	8,28,29,30,31
$Mg(NO_3)_2$	..	102.5	94.52	80.4	75.2	67.2	59.5	47.9	39.2	32.1	.....	.....	.....	....	....	5,8,9
$MgSO_4$	..	99.9	76.0	49.57	43.0	34.8	28.99	21.4	16.1	12.0	8.8	.....	.....	....	....	2,3,32,33
$MnCl_2$	..	.....	.....	.....	.....	66.4	57.4	45.3	35.7	28.1	21.6	16.3	.....	....	....	1
$Mn(NO_3)_2$	..	.....	.....	.....	.....	66.8	58.9	47.5	37.9	30.3	23.5	.....	.....	....	....	1
$MnSO_4$	..	.....	.....	.....	.....	30.0	24.7	18.5	14.2	10.8	8.1	5.8	3.8	....	....	17
$Na_2B_4O_7$	25	86.	78.6	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	....	....	34
NaBr	..	.....	.....	95.9	91.7	84.5	78.0	69.1	60.5	53.0	.....	.....	.....	....	....	19
NaCl	..	106.27	101.72	91.82	87.53	80.76	74.19	64.6	56.4	42.6	.....	.....	.....	....	....	22

[18] Gropp, *Dissertation*, Rostock (1915).
[19] Heydweiller, *Ann. Physik*, **30**, 873 (1909).
[20] Heydweiller, "Gesammelte Abhandlungen von Fr. Kohlrausch," vol. 2, Barth, Leipzig (1911).
[21] Kohlrausch and Steinwehr, *Sitzb. preuss. Akad. Wiss.*, p. 581 (1902).
[22] Kohlrausch and Maltby, *Sitzb. preuss. Akad. Wiss.*, p. 665 (1899).
[23] Parker and Parker, *J. Am. Chem. Soc.*, **46**, 312 (1924).
[24] Weiland, *J. Am. Chem. Soc.*, **40**, 131 (1918).
[25] Walden and Ulich, *Z. physik. Chem.*, **106**, 49 (1923).
[26] Kohlrausch, *Sitzb. preuss. Akad. Wiss.*, p. 1002 (1900).
[27] Sherrill, *J. Am. Chem. Soc.*, **32**, 741 (1910).
[28] Jones *et al.*, *Carnegie Inst. Wash. Pub.* 180 (1913).
[29] Kohlrausch and Grotrian, *Nachr. kgl. Ges. Wiss. Göttingen*, p. 405 (1874).
[30] Kohlrausch and Grotrian, *Ann. physik. Chem.*, **154**, 1 (1875).
[31] Kohlrausch and Grotrian, *Ann. physik. Chem.*, **154**, 215 (1875).
[32] Foster, *Phys. Rev.*, **8**, 257 (1899).
[33] Harkins and Paine, *J. Am. Chem. Soc.*, **41**, 1155 (1919).
[34] Walden, *Z. physik. Chem.*, **1**, 529 (1887).

Table 7. Equivalent Conductivities of Salts, Acids, and Bases—(*Continued*)

Salts	°C.	0.001 *N*	0.01 *N*	0.1 *N*	0.2 *N*	0.5 *N*	1 *N*	2 *N*	3 *N*	4 *N*	5 *N*	6 *N*	7 *N*	10 *N*	20 *N*	Reference
$NaClO_3$	..	95.5	91.0	80.8	76.4	69.1	61.8	51.7	43.5	36.1	.....	...	.....	....	....	3,16,35
Na_2CO_3	..	112.	96.1	72.8	65.	54.4	45.4	.....	.....	.....	.....	.....	.....	....	....	36,37
Na_2CrO_4	..	......	......	82.1	75.8	66.3	57.8	46.6	37.8	30.6	.....	.....	.....	....	....	16,38
NaF	..	87.65	83.33	72.9	67.8	59.8	51.8	.....	.....	.....	.....	.....	.....	....	....	9,21
$NaNO_3$	..	102.60	97.93	87.04	82.09	73.88	65.72	54.6	46.1	39.2	.....	.....	.....	....	....	8,9,22
Na_3PO_4	25	......	118.	......	......	......	......	.....	.....	.....	.....	.....	.....	....	....	34
Na_2SO_4	..	105.8	96.1	77.6	70.4	59.4	50.3	39.6	.....	.....	.....	.....	.....	....	....	1,2,9,17,20,26, 27,36,37
NH_4F	..	......	......	89.9	83.4	74.3	65.6	.....	.....	.....	.....	.....	.....	....	....	16
$NiCl_2$	..	......	......	......	......	70.8	62.1	50.6	41.0	33.3	.....	.....	.....	....	....	1
$Ni(NO_3)_2$	..	......	......	......	......	66.9	57.6	46.0	37.2	29.3	22.7	16.	.....	....	....	1
$NiSO_4$	..	94.4	69.8	43.8	37.9	30.6	25.4	19.3	15.1	.....	.....	.....	.....	....	....	10,17,39,40
$PbCl_2$	..	118.98	102.0	......	......	......	......	.....	.....	.....	.....	.....	.....	....	....	5,9
$Pb(NO_3)_2$	..	116.0	103.44	77.18	67.29	53.15	41.97	30.6	.....	.....	.....	.....	.....	....	....	2,9,41
$SrCl_2$	..	114.3	105.3	90.4	85.1	76.1	67.9	57.5	49.7	.....	.....	.....	.....	....	....	2,29,30,31
$Sr(NO_3)_2$	..	108.16	98.90	80.82	73.70	62.64	52.00	38.4	28.9	21.1	16.4	.....	.....	....	....	2,9
$ZnCl_2$	..	......	......	86.5	79.3	68.7	56.2	39.5	29.6	22.9	18.4	15.2	12.9	8.	0.7	16,41
$Zn(NO_3)_2$	..	......	......	80.5	75.2	67.2	59.2	47.9	38.8	31.0	.....	.....	.....	....	....	16
$ZnSO_4$	..	98.5	72.8	45.4	39.1	31.4	26.0	20.0	15.6	11.9	8.9	.....	.....	....	....	2,10
$FeCl_3$	..	......	......	......	......	66.5	53.1	37.5	27.8	20.9	15.9	12.4	.....	....	....	1
Acids:																
HBr	..	......	......	355.2	347.7	328.4	301.1	.....	.....	.....	.....	....	.....	....	....	16
HCl	..	377.	369.3	350.1	341.5	326.6	300.5	253.8	214.7	182.	152.	.....	.....	64.3	....	10,37,42,43,44
$HClO_3$	..	......	......	343.6	334.6	316.6	291.4	.....	.....	.....	.....	.....	.....	....	....	8,16,19

---

[35] Flügel, *Z. physik. Chem.*, **79**, 577 (1912)
[36] Kohlrausch, *Ann. physik. Chem.*, **6**, 145 (1879).
[37] Kohlrausch, *Ann. physik. Chem.*, **26**, 161 (1885).
[38] Claussen, *Dissertation*, Rostock (1911).
[39] Pfanhauser, *Z. Elektrochem.*, **7**, 698 (1901).
[40] Murata, *Chem. Soc. Japan, Bull.*, **3**, 47 (1928).
[41] Long, *Ann. physik. Chem.*, **11**, 37 (1880).
[42] Goodwin and Haskell, *Phys. Rev.*, **19**, 369 (1904).
[43] Lorenz and Ostwald, *Z. anorg. allgem. Chem.*, **114**, 209 (1920).
[44] Noyes and Cooper, *Carnegie Inst. Wash. Pub.* 63, p. 115 (1907).
[45] Brownson and Cray, *J. Chem. Soc., London*, **127**, 2923 (1925).

Table 7. Equivalent Conductivities of Salts, Acids, and Bases—(*Concluded*)

Salts	°C.	0.001 *N*	0.01 *N*	0.1 *N*	0.2 *N*	0.5 *N*	1 *N*	2 *N*	3 *N*	4 *N*	5 *N*	6 *N*	7 *N*	10 *N*	20 *N*	Reference
$HClO_4$	25	......	405.	384.	373.	359.	342.	291.	226.	185.	150.	118.	91.	41.	......	46,47,48
H_2CrO_4	25	......	193.	186.	......	......	......	......	......	......	......	......	......	......	......	7
HNO_3	..	372.	364.	345.7	336.	321.	305.5	259.	220.	184.	154.	129.1	108.6	64.9	5.5	10,31,42,49,50
H_3PO_4	..	......	202.7	96.4	......	......	......	24.	......	......	......	14.	13.	11.4	6.4	49
H_2SO_3	..	......	130.	80.	......	......	......	24.	......	......	......	......	......	......	......	51
H_2SO_4	..	360.	308.6	232.9	......	......	198.6	182.7	166.5	150.5	134.6	119.1	105.	70.	12.	5,49,52,53,54
Bases:																
$Ba(OH)_2$	..	217.	207.	179.7	......	......	......	......	......	......	......	......	......	......	......	36,49
$Ca(OH)_2$	25	......	220.	......	......	......	......	......	......	......	......	......	......	......	......	47
KOH	..	234.	228.	213.	207.	197.	184.	160.2	140.4	122.0	105.6	90.6	77.1	39.6	......	36,37,55
LiOH	..	......	......	180.5	172.2	156.9	139.	113.3	94.4	78.7	65.3	......	......	......	......	8,16,19
NaOH	..	......	208.4	195.3	189.0	175.5	158.4	129.4	104.2	86.7	68.9	54.2	43.	20.5	4.4 (19 *N*)	36,37,55,56,57,58

---

[46] Linde, *Z. Elektrochem.*, **30**, 255 (1924).
[47] Ostwald, "Lehrbuch der allgemeinen Chemie," Engelmann, Leipzig, 1891–1903.
[48] Smith, *J. Am. Chem. Soc.*, **45**, 360 (1923).
[49] Noyes and Eastman, *Carnegie Inst. Wash. Pub.* 63, 239 (1907).
[50] Veley and Manley, *Phil. Trans. Roy. Soc. (London)*, **A**, **191**, 365 (1898).
[51] Lindner, *Monatsh.*, **33**, 613 (1912).
[52] Gibson and Gibson, *Proc. Roy. Soc.* (Edinburgh), **30**, 254 (1909).
[53] Kohlrausch, *Ann. physik. Chem.*, **159**, 233 (1876).
[54] Kohlrausch, *Ann. physik. Chem.*, **17**, 69 (1882).
[55] Heydweiller, *Ann. Physik*, **48**, 681 (1915).
[56] Bousfield and Lowry, *Phil. Trans. Roy. Soc. (London)*, **A**, **204**, 253 (1905).
[57] Noyes and Kato, *Carnegie Inst. Wash. Pub.* 63, 151 (1907).
[58] Raikes, Yorke, and Ewart, *J. Chem. Soc. London*, p. 630 (1926).

and $K = 1/r$, $\Lambda_v = V/r$, and $r = V/\Lambda_v$, the resistance of electrolytes at various concentrations may be calculated from the tables. V can be calculated from the normalities or concentrations given. If, instead of normalities, concentrations C in mil-equivalents per liter be used, the normalities must be multiplied by 10^3 and $\Lambda_v = 10^6\ K/C$, from which specific conductivities and resistances may be calculated. The values in Table 7 are reciprocal ohms. This table has been condensed from those of Washburn and Klemenc and Parker and Klemenc in the "International Critical Tables," vol. 6, by permission.

Electrochemical Energy

Differences of potential are electrochemically produced by voltaic or galvanic cells, which may be defined as any arrangement by which the energy of chemical reactions or of certain physical processes, such as diffusion, is converted into electrical energy.

If the reaction involved in a voltaic cell be written completely as a thermochemical equation, and on an equivalent basis for the components involved, and the thermal energy be converted into electrical units, the electrical energy (in watts) divided by the faraday (in coulombs), the quotient of watts per equivalent divided by coulombs per equivalent will be volts. This will be the rough calculation of the theoretical decomposition voltage of the reaction of the cell under the conditions and limitations of the reaction.

The effective voltage of a primary cell is a function of the amount of current drawn from the cell. The greater the current, the lower will be the voltage across the terminals. When the current flowing is infinitely small, the voltage will have its maximum value, a figure which is termed the e.m.f. of the cell, or voltage on open circuit. Conversely, the greater the current forced into the cell, the higher the necessary voltage which must be applied across the terminals. With an infinitely small current the minimum value of applied voltage approaches the e.m.f. of the cell as a limit.

The maximum amount of electrical energy can be developed only when a cell operates isothermally and its reactions are completely reversible.

In a reversible process involving the conversion of energy, $U = A - Q$, where Q is the heat absorbed by the system (following the custom of thermodynamics, heat developed by a reaction is taken as negative), and A the external work done by the system when its total energy decreases by U. Electrical energy converted from chemical energy has a maximum value equal to A. In electrical units $A = nFE$, where n is the number of equivalents involved. If Q be negligible,

$$U = nFE$$

and

$$E = \frac{U}{nF}$$

The e.m.f. of a cell can be calculated from either thermochemical data or the theoretical decomposition voltage of a compound.

The relation between electrical energy of a system and the heat of reaction is given by the Gibbs-Helmholtz equation

$$A - U = \frac{TdA}{dT}$$

in which T is the absolute temperature. In chemical reaction systems, the external work is small. Since A approaches zero as a limit, $U = -Q$. By substitution in the Gibbs-Helmholtz equation for A and U, we obtain

$$nFE + Q = \frac{TnFdE}{dT}$$

whence

$$E = \frac{-Q}{nF} + \frac{TdE}{dT}$$

If E be expressed in volts and Q in calories, then

$$E = \frac{-JQ}{nF} + \frac{TdE}{dT}$$

where $J = 4.182$ is the electrical equivalent of heat. It will be observed that when dE/dT is positive, the e.m.f. of a reversible voltaic cell increases with rise in temperature; when zero, the electrical energy is equal to the chemical energy. By the use of this equation, heats of reaction may be determined by e.m.f. measurements. The equation also allows the determination of the theoretical voltage of a cell for the calculation of the theoretical energy and energy efficiency.

The existence of a definite tendency toward the passage from the atomic to the ionic state is designated as electrolytic solution pressure (p). In general, the values decrease in the same order as the increase of the electropositive character of the metal. The nonmetals are also assumed to have electrolytic solution pressures, the order in the case of the halogens and sulphur being fluorine, chlorine, bromine, iodine, and sulphur. If a bar of zinc be placed in water, some zinc atoms give up two electrons each to the bar of metal, passing into the water as positively charged zinc ions. An electric double layer is thus formed at the interface of the metal and the liquid. The electrostatic attraction of the negative charges accumulating on the metal surface opposes the passage of atoms to the ionic state. If a piece of zinc be immersed in a solution of a copper salt, copper ions will discharge and deposit as copper atoms on the zinc. The negative charges on the zinc will be reduced, and more zinc atoms will be able to assume the ionic state. The amount of zinc entering the solution will be equivalent to the amount of copper deposited. When any metal A is immersed in a solution of a salt of another metal B having a lower electrolytic solution pressure, B is deposited and A enters solution. Electrolytic solution explains the displacement of metals by others from solution, and the solution of metals in acids (displacement of hydrogen). When a metal of a high electrolytic solution pressure is in contact with a solution of its own ions, the tendency of the atoms to pass into the ionic state is opposed by the osmotic pressure P of the metal ions in solution. If $p > P$, the metal will dissolve; if $P > p$, the metal will deposit.

The **single-electrode potential** is the difference of potential between the electrode and the solution around it. The e.m.f. of a cell is equal to the difference of the single-electrode potentials of the electrodes of the cell.

$$E = e_1 - e_2$$

where e_1 and e_2 are single-electrode potentials. The normal hydrogen electrode is the standard and a single potential of zero is assigned to it. A normal hydrogen electrode consists of a platinized-platinum plate, half immersed in a normal H^+ solution and half surrounded by pure hydrogen gas which is bubbled through the solution. The hydrogen is dissolved in the platinized platinum and behaves like a metal. A definite and reproducible potential difference is set up between the electrode and the H^+ solution. The single potential of an electrode can be obtained by measuring the e.m.f. of a cell in which the hydrogen electrode is combined with the electrode in question. The sign of this single potential will be positive or negative according as the electrode in question is the positive or negative pole of the cell. Such values are single-electrode potentials on the hydrogen scale.

Table 8. Electrode Potentials

	Volts	Reference		Volts	Reference
$Li^+ + e = Li$	−2.959	1			
$Rb^+ + e = Rb$	−2.925	1			
$K^+ + e = K$	−2.924	1			
$Ca^{++} + 2e = Ca$	−2.76	2			
$Na^+ + e$	−2.714	1			
			$Ba^{++} + 2e = BaHg$	−1.570	6
			$NaCl + e = NaHg (+Cl^-$ in sat. NaCl	−1.837	7
			$Sr^{++} + 2e = SrHg$	−1.793	6
$Zn^{++} + 2e = Zn$	−0.761	1			
$Cr^{++} + 2e = Cr$	−0.557	3			
$Fe^{++} + 2e = Fe$	−0.44	1			
$Cd^{++} + 2e = Cd$	−0.401	1			
			$Tl^+ + e = TlHg$	−0.336	1
$Ni^{++} + 2e = Ni$	−0.23	4			
$Sn^{++} + 2e = Sn$	−0.136	1			
$Pb^{++} + 2e = Pb$	−0.122	1			
$H^+ + e = \frac{1}{2}H_2$	0.000				
			$Sb_2O_3 + 6H + 6e = 2Sb + 3H_2O$	0.144	8
			$BiOCl + 2H^+ + 3e = Bi + Cl^- + H_2O$	0.158	9,10
			$As_2O_3 + 6H^+ + 6e = 2As + 3H_2O$	0.234	11
$Cu^{++} + 2e = Cu$	0.344	1			
$Ag^+ + e = Ag$	0.797	1			
$Hg_2^{++} + 2e = 2Hg$	0.798	1			
$Au^{+++} + 3e = Au$	1.36	5			

---

[1] Lewis and Randall, "Thermodynamics and the Free Energy of Chemical Substances," McGraw-Hill, New York, 1923.

[2] Drucker and Luft, *Z. physik. Chem.*, **121**, 307 (1926).

[3] Grube and Breitinger, *Z. Elektrochem.*, **33**, 112 (1927).

[4] Haring and Van den Bosche, *J. Phys. Chem.*, **33**, 161 (1929).

[5] Jirsa and Jellinek, *Chem. Listy*, **18**, 1 (1924); *Z. Elektrochem.*, **30**, 286 (1924).

[6] Danner, *J. Am. Chem. Soc.*, **46**, 2385 (1924).

[7] Danner, *J. Am. Chem. Soc.*, **44**, 2832 (1922).

[8] Schuhmann, *J. Am. Chem. Soc.*, **46**, 52 (1924).

[9] Jellinek and Kühn, *Z. physik. Chem.*, **105**, 337 (1923).

[10] Noyes and Chow, *J. Am. Chem. Soc.*, **40**, 739 (1918).

[11] Schuhmann, *J. Am. Chem. Soc.*, **46**, 1444 (1924).

In order to indicate the direction of the polarity between a metal and a solution, the sign of the charge on the metal is placed before the potential difference between the two phases—the so-called potential of the metal or the electrode potential.

This convention, according to Bancroft [*Trans. Am. Electrochem. Soc.*, **33**, 79 (1918)] the only one which can be adopted universally, has been adopted officially by the Bunsen Gesellschaft, the American Electrochemical Society, and the National Bureau of Standards and is employed by most European electrochemists and largely in this country. Another convention, advocated by Lewis [*J. Am. Chem. Soc.*, **35**, 1 (1913)], is extensively employed in this country. According to this, the potential difference of a metal-solution junction is considered as positive when there is a tendency for positive electricity to flow from left to right through the junction as written, and as negative when there is a tendency for positive electricity to flow from right to left.

In solutions containing their own ions, noble metals (*e.g.*, with electrolytic solution pressures lower than that of hydrogen) acquire a positive potential, while base metals (*e.g.*, with electrolytic solution pressures greater than that of hydrogen) acquire a negative potential. In accordance with this convention, the potential of a metal in contact with a solution containing its own ions is positive when $p < P$, and negative when $p > P$.

A short arrow will sometimes be placed above a metal-solution junction to indicate the direction in which the positive current tends to flow. When the arrow points toward the metal, its potential is positive; when it points away from it, its potential is negative.

The magnitude of a single-electrode potential will be a function of p and P. For reactions of the reversible type,

$$M^{n\oplus} \rightleftharpoons M + n\oplus$$

Nernst developed the equation

$$neF = -RT \log_e \frac{p}{P}$$

on the assumption that the gas laws are valid in respect to ions in strong electrolytes, where p is the electrolytic solution pressure of metal M; P the osmotic pressure of the M ions in solution; e the single-electrode potential corresponding to the equilibrium; R the gas constant; and T the absolute temperature. For dilute solutions, P is a function of c, the concentration, and $P = kc$. Then

$$neF = -RT \log_e \frac{p}{kc}$$

$$e = -\frac{RT}{nF} \log_e \frac{p}{k} + \frac{RT}{nF} \log_e c$$

For a pure metal at a given temperature

$$-\frac{RT}{nF} \log_e \frac{p}{k} = e_o$$

or the normal electrode potential of the specific equilibrium. If $T = 291$ (18° C.), $R = 8.32$ joules and, using Briggs' logarithms for the cation,

$$e_c = e_o + \frac{(0.058 \log c)}{n}$$

and for the anion,

$$e_a = e_o - \frac{(0.058 \log c)}{n}$$

if $c = 1$, $e = e_o$. The potential of an electrode equilibrium is then the potential difference of the electrode material and a molar solution of the ion involved.

A table of electrode potentials will furnish data as to the quantitative aspect of electrode equilibriums in a concise form. A table of the more important single potentials is given, the values being taken from Gerke, "International Critical Tables," vol. 6, McGraw-Hill, by permission.

In addition to electrode reactions between metal electrodes and metal ions, gas electrodes exist in that hydrogen, the halogens, and oxygen also are known to ionize. Nitrogen, however, does not ionize. The gases are bubbled against the surface of a platinized-platinum electrode in an electrolyte containing the ion concerned. The gases are dissolved or sorbed by the metal and can ionize. The P of a gas dissolved in platinum is a function of the concentration of the gas in the metal and the gas pressure on the entire system.

Table 9. Reduction Reactions

	Volts	Reference
$Cr^{+++} + e = Cr^{++}$	−0.40	1,2
$Cu_2O + H_2O + 2e = 2Cu + 2OH^-$	−0.34	3
$CuS + 2H^+ + e = Cu + H_2S$	−0.259	4
$SbO + 2H^+ + 3e = Sb + H_2O]$	−0.212	5
$PbS + 2H^+ + 2e = Pb + H_2S$	0.07	4
$PbO_2 + H_2O + e = PbO + 2OH^-$	0.27	6
$Tl^{+++} + e = Ti^{++}$	0.37	7
$Cu^{++} + 2Cl^- + e = CuCl_2^-$	0.455	8
$K_3Fe(CN)_6 + K^+ + e = K_4Fe(CN)_6$	0.486	9
$H_3AsO_4 + 2H^+ + 2e = H_3AsO_3 + H_2O$	0.57	10
$MnO_4^- + e = MnO_4^{--}$	0.664	11
$Fe^{+++} + e = Fe^{++}$	0.747	12,13
$Tl^{+++} + 2e = Tl^+$	1.21	14
$Sn^{++++} + 2e = Sn^{++}$	1.25	15
$MnO_2 + 4H^+ + 2e = Mn^{++} + 2H_2O$	1.33	16
$Ce^{++++} + e = Ce^{+++}$	1.55	17
$MnO_4^- + 4H^+ + 3e = MnO_2 + 2H_2O$	1.58	18
$Co^{+++} + e = Co^{++}$	1.81	19,20

[1] Forbes and Richter, *J. Am. Chem. Soc.*, **39**, 1140 (1917).
[2] Grube and Breitinger, *Z. Elektrochem.*, **33**, 112 (1927).
[3] Allmand, *J. Chem. Soc., London*, **95**, 2151 (1909).
[4] Knox, *Trans. Faraday Soc.*, **4**, 29 (1908).
[5] Schuhmann, *J. Am. Chem. Soc.*, **46**, 52 (1924).
[6] Glasstone, *J. Chem. Soc., London*, **121**, 1456 (1922).
[7] Forbes and Hall, *J. Am. Chem. Soc.*, **46**, 385 (1924).
[8] Carter and Lea, *J. Chem. Soc., London*, **127**, 499 (1925).
[9] Lewis and Sargent, *J. Am. Chem. Soc.*, **31**, 355 (1909).
[10] Foerster and Pressprich, *Z. Elektrochem.*, **33**, 176 (1927).
[11] Sackur and Taegener, *Z. Elektrochem.*, **18**, 718 (1912,.
[12] Noyes and Brann, *J. Am. Chem. Soc.*, **34**, 1016 (1912).
[13] Popoff and Kunz, *J. Am. Chem. Soc.*, **51**, 382 (1929).
[14] Grube and Hermann, *Z. Elektrochem.*, **26**, 291 (1920).
[15] Forbes and Bartlett, *J. Am. Chem. Soc.*, **36**, 2030 (1914).
[16] Tower, *Z. physik. Chem.*, **32**, 566 (1900).
[17] Baur and Glaessner, *Z. Elektrochem.*, **9**, 534 (1903).
[18] Brown and Tefft, *J. Am. Chem. Soc.*, **48**, 1128 (1926).
[19] Jahn, *Z. anorg. allgem. Chem.*, **60**, 292 (1908).
[20] Lamb and Larson, *J. Am. Chem. Soc.*, **42**, 2024 (1920).

Many reactions, when resolved into their constituent oxidation and reduction processes, give electrode reactions which involve transference of electricity from ion to ion. Cells involving these reactions are oxidation-reduction cells, and the electrode systems oxidation-reduction electrodes. The essential reactions are the changes in the amount of electricity associated with the substances. For a reaction of the type $M^x \rightleftharpoons M^y + (x - y) \oplus$, the potential difference varies with ionic concentration according to the relation

$$e = e_0 + \frac{0.058}{n} \log \frac{c}{c'}$$

where c = concentration of M^x ions and c' = concentration of M^y ions. When $c = c' = 1$, $\log \frac{c}{c'} = 0$. Table 9 gives representative values of oxidation-reduction potentials for some common cells.

In concentration cells, potential differences are set up between electrodes in contact with different concentrations of the same electrolyte. The e.m.f. of a concentration cell is a function of the ratio of the ionic concentrations at the two electrodes. Only when the concentration ratio is large are the e.m.f. values large.

Polarization, Overvoltage, and Passivity

1. Polarization. If current from an external source be sent into a cell, the voltage of the cell terminals is raised above its static value. The cell is said to be polarized. When, however, an appreciable current is passing across the boundary between an electrode and a solution, the value of the potential difference between the two is changed from its equilibrium value as given by the Nernst equation. The difference between these two values is called **polarization.** Complete or partial removal of this difference is termed **depolarization.** Any agent which does this work is called a **depolarizer.**

A primary cell becomes polarized when discharged so that its e.m.f. falls below its static value. When the potential of an electrode is raised, the electrode is anodically polarized; when the potential is lowered or becomes more negative, the electrode is cathodically polarized. The amount of polarization is the difference between the actual and the equilibrium values of the electrode potential.

Owing to irreversibility of electrode processes and ohmic resistances in the cell at various points, the working voltage of a cell always exceeds the theoretical decomposition voltage. The percentage ratio between the theoretical quantity of energy necessary for the production of a given amount of a substance and the actual quantity of energy, the latter always being the larger, is termed the **energy efficiency** of the process. The energy efficiency may also be described as the product of the **current efficiency** and the percentage ratio of the theoretical to the actual voltage.

Electrode processes occur essentially at the surface of the electrode in contact with the electrolyte or in thin films adjacent to these electrodes. The bulk of the electrolyte may be considered merely as a reservoir for ions and as a conducting medium. At the cathode the discharge of an ion would tend to reduce the concentration of the ions in the cathode film. As a result, the single potential of the electrode tends to become more negative and cathodic polarization begins. The higher the current density, the greater will be the tendency for the concentration of the ions in the cathode film to be reduced. This tendency is opposed by diffusion of the ions from the region of higher concentration in the electrolyte to that of lower concentration around the cathode. Convection currents, mechanical agitation, and other compensating processes will tend to reduce this form of concentration polarization. A similar set of conditions holds true for the anode, with the exception that the single potential of the anode hence becomes more positive.

If the current become so large that the concentration of metal ions on the cathode surface is reduced practically to zero, no larger current can pass, however great the potential difference be made, unless some other ions begin to deposit. This value is termed the **limiting** or **maximum current.** It depends upon the concentration, temperature, and rate of stirring of the solution.

If the polarization occurring at ordinary current density be not greater than that accounted for by concentration polarization, the process is considered as reversible. Otherwise the process is considered to be irreversible.

Polarization may be caused by factors which interfere with the main electrode process. Films of nonconducting substances may form on the electrode surfaces, and the current density at the uncoated sections of the electrodes will increase. Enormous polarization may set in at electrode surfaces completely covered with films.

Commercial utilization of this type of effect is found in anodic oxidation of aluminum and its alloys, tantalum, and other metals in rectifiers, electrolytic lightning arresters, etc.

Electrolytic Rectifiers (see Electrical Section). Electrodes which, as anodes, have become covered with an insulating film show different behaviors when employed as cathodes. With some the insulating layer is easily removed, but in the case of metals having difficultly reducible oxides (Al and Ta) the film continues for some time although H^+ ions may be discharged through it. Such electrodes are **electrolytic valves.** They possess the property of permitting current passage only in one direction. A cell with an aluminum plate as cathode and a lead sheet as anode in a suitable electrolyte may be placed in series with and between a generator and a storage cell battery. The interposed cell does not interfere with the charging current to the cells, but prevents the storage cells from "feeding back" to the generator when that machine is shut down.

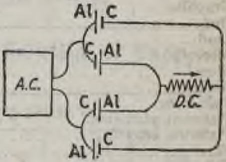


Fig. 2.—Electrolytic rectifier.

Electrolytic valves may be employed as rectifiers for a.c.-d.c. conversion. A typical arrangement is shown in Fig. 2. Four rectifier cells, each containing an aluminum electrode and one of carbon, iron, or platinum, are connected as shown. The film formation on aluminum allows the passage of appreciable current only when the electrode is used as cathode. From the figure it can be verified that the current produced on the right-hand circuit will be d.c., and that both positive and negative waves of the a.c. are utilized. The energy efficiency is about 60 per cent.

It so happens that in some cases polarization is definitely necessary for the production of the desired electrode process. In chromium deposition, the cathodic hydrogen film is an important factor in the production of the chromium metal. Polarization effects may markedly affect the type of deposits obtained. When these cannot be eliminated by variation of current and voltage, depolarizers are employed. In high-quality nickel plating, hydrogen adsorbed by the nickel affects the ductility of the metal. Oxidizers like hydrogen peroxide are added to convert the hydrogen codeposited with the nickel into water. Chlorides in various plating baths and in refining solutions find application in overcoming anode polarization by increasing the rate of anode corrosion.

2. Overvoltage. Hydrogen Overvoltage. Even at low current density, some polarization is usually needed to cause cathodic hydrogen evolution. At higher current density, these polarizations may be of considerable magnitude. The polarization voltage needed for the evolution of a gas at an electrode material is known as the overvoltage of the material for the gas under the conditions stated. The hydrogen overvoltage of an electrode is the difference between the (actual) cathode potential for hydrogen evolution and the equilibrium (theoretical) potential of hydrogen in the same electrolyte. Hydrogen overvoltage of an electrode material may be obtained from the current-density-electrode-potential curves for hydrogen evolution at a cathode of that material. The values vary directly with the current density, the extent of this increase usually being greater than accountable by concentration changes in the electrolyte. Hydrogen overvoltages at very high current density for nearly all substances approach limiting values of about 1.3 volts. A table of hydrogen overvoltages for various metals (Table 10) is given above. This is condensed from Knobel's data in "International Critical Tables," McGraw-Hill, by permission.

Table 10. Overvoltage, H_2 at 25°C.; Electrolyte 2*N* H_2SO_4*, †, ‡, §, ‖, ¶

	Amp. per sq. cm.				
	0.001	0.01	0.1	0.5	1.0
Aluminum	0.56	0.83	1.0	1.24	1.29
Bismuth	0.78	1.05	1.14	1.21	1.23
Brass	0.50	0.65	0.91	1.23	1.25
Cadmium	0.98	1.13	1.22	1.25	1.25
Carbon	....	0.70	0.9	1.1	1.17
Copper	0.48	0.58	0.8	1.19	1.25
Duriron	0.20	0.29	0.61	0.86	1.02
Gold	0.24	0.39	0.59	0.77	0.80
Graphite	0.60	0.78	0.98	1.17	1.22
Iron	0.40	0.56	0.82	1.26	1.29
Lead	0.52	1.09	1.18	1.24	1.26
Mercury	0.9	1.04	1.07	1.1	1.12
Monel	0.28	0.38	0.62	0.86	1.07
Nickel	0.56	0.75	1.05	1.21	1.24
Palladium	0.12	0.3	0.7	1.	1.
Platinum, platinized	0.015	0.03	0.04	0.05	0.05
Platinum, smooth	0.024	0.07	0.29	0.57	0.68
Silver	0.47	0.76	0.88	1.03	1.09
Tellurium	0.4	0.45	0.48	0.54	0.6
Tin	0.86	1.08	1.22	1.24	1.23
Zinc	0.72	0.75	1.06	1.20	1.23

* Knobel, *J. Am. Chem. Soc.*, **46**, 2613 (1924).
† Knobel, *J. Am. Chem. Soc.*, **46**, 2751 (1924).
‡ Knobel, Caplan, and Eiseman, *Trans. Am. Electrochem. Soc.*, **43**, 55 (1923).
§ Knobel and Joy, *Trans. Am. Electrochem. Soc.*, **44**, 443 (1923).
‖ Newbery, *J. Chem. Soc., London*, **105**, 2419 (1914).
¶ Newbery, *Trans. Faraday Soc.*, **15** I., 126 (1919).

Table 11. Overvoltage, O_2 at 25°C.; Electrolyte 1*N* KOH*

	Amp. per sq. cm.				
	0.001	0.01	0.1	0.5	1.0
Copper	0.42	0.58	0.66	0.74	0.79
Gold	0.67	0.96	1.24	1.53	1.63
Graphite	0.53	0.9	1.09	1.19	1.24
Nickel, smooth	0.35	0.52	0.73	0.82	0.85
Nickel, spongy	0.41	0.56	0.69	0.74	0.76
Platinum, platinized	0.40	0.52	0.64	0.71	0.77
Platinum, smooth	0.72	0.85	1.28	1.43	1.49
Silver	0.58	0.73	0.98	1.08	1.13

*References as under Table 10.

Hydrogen overvoltage values for most cathode materials are of practical importance even at moderate current density. The highest hydrogen overvoltages are shown by the soft metals of low melting point, while those of high melting point have relatively small values.

Overvoltage depends upon a large number of factors such as condition of the electrode surface, purity of the electrode, temperature, impurities and colloidal substances in the electrolyte, external pressure, imposed a.c., current density, and time of electrolysis.

Table 12. Overvoltage, Cl_2 at 25°C.; Electrolyte Saturated Solution of NaCl or KCl*

	Amp. per sq. cm.				
	0.001	0.01	0.1	0.5	1.0
Graphite	.....	.....	0.25	0.42	0.5
Platinum, platinized	0.006	0.016	0.026	0.05	0.08
Platinum, smooth	0.008	0.03	0.054	0.16	0.24

Overvoltage, Br_2 at 25°C.; Electrolyte Saturated Solution of NaBr or KBr

	Amp. per sq. cm.			
	0.01	0.1	0.5	1.0
Graphite	0.002	0.027	0.16	0.33
Platinum, platinized	0.002	0.012	0.07	0.2
Platinum, smooth	0.002	.....	0.26	0.4

Overvoltage, I_2 at 25°C.; Electrolyte Saturated Solution of NaI or KI

	Amp. per sq. cm.			
	0.01	0.1	0.5	1.0
Graphite	0.013	0.1	0.4	0.8
Platinum, platinized	0.006	0.03	0.09	0.2
Platinum, smooth	0.004	0.03	0.12	0.22

*References are same as under Table 10.

Overvoltage necessitates increased energy expenditure in the separation of electrolytic gases. In addition it may actually change the nature of an electrode process as the result of the change of electrode potential.

Oxygen Overvoltage. In a manner similar to hydrogen overvoltage, oxygen overvoltage exists in the course of anodic evolution of oxygen. Only the noble metals, and a few others like those of the iron group which can be "ennobled," can be satisfactorily employed for the evolution of oxygen. A table of values is given in Table 11 (condensed from Knobel's tables in "International Critical Tables.")

Halogen Overvoltage. Overvoltages for the halogens exist of a nature comparable to hydrogen and oxygen overvoltages. On continued electrolysis these frequently attain very large values. Values for chlorine, bromine, and iodine overvoltages are given in Table 12 (condensed from Knobel's tables, in "International Critical Tables," McGraw-Hill, by permission).

3. **Passivity.** In the case of the iron-group metals and chromium, solution of these metals as anodes may be displaced by other reactions without the formation of visible films on the anode. Normally the anodic solution of this group of metals requires a considerable polarization, but at higher current density, oxygen evolution sets in, the metal is "passivated" or ennobled. Reduction of current density does not immediately eliminate the passive state of the metal. Similar phenomena occur with other metals, particularly with the noble ones whose reversible potentials and oxygen-evolution potentials are close together. In general, halogen ions interfere with passivity while oxidizing ions favor its inception. Chromium and its alloys are easily rendered passive and are thus resistant to anodic solution or corrosion. Passivity may markedly affect the type of anode reactions taking place.

The passive state in metals when used as anodes in electrolytic cells is analogous to chemical passivity, a state which is of considerable importance in the corrosion resistance of metals when used as structural materials.

Many theories of anodic metal solution and passivity have been proposed. In general, the anodic polarization required for the solution of certain metals is attributed to the formation of metal-oxygen complexes at the electrode surface which make the single potential of the electrode more positive and hinder its solution. The value of this quantity may become so great that other anode reactions set in. The passivation of a metal, whether anodic or chemical, will be favored by those conditions which tend to produce a high oxygen concentration in the surface layers of the metal.

Superimposed A.C. on D.C. Superimposing an a.c. on d.c. in electrolysis causes a decrease in electrode potential, reduces any irreversibility of the reaction, acts as a depolarizer, reduces hydrogen overvoltage [Goodwin and Knobel. *Trans. Am. Electrochem. Soc.*, **37**, 617 (1920)], affects oxygen evolution at anodes [Grube and Dulk, *Z. Elektrochem.*, **24**, 237 (1918)], lowers chlorine overvoltage, aids anodic solution of metals, and allows higher current density of d.c. at an electrode.

The electrochemical effects are a function of the direct current, and the ratio of a.c. to d.c. and the frequency of the a.c. determine the magnitude of its effects. The modified Wohlwill gold-refining process is the only industrial use of superimposed a.c. on d.c.

Primary and Secondary Cells

Batteries. The conversion of chemical into electrical energy is the function of primary cells or batteries. The term *batteries* is usually applied to an assembly of identical units or cells, but is often loosely used for designating a single unit. In a **primary battery** the chemically reacting parts require renewal or replacement, while in a **secondary** or **storage battery,** reactions being reversible to a high degree, the chemical conditions are restored after partial or complete discharge by reversal of the current flow, *i.e.*, by sending electric current into the cell. The high cost of primary batteries makes the production of electricity in large quantities from them impractical, so that they are largely used for services of an intermittent nature, or for those demanding electric current for short times.

From the table of single-electrode potentials (Table 8), we can determine the e.m.f. of a primary cell resulting from the combination of two electrode systems, one with a high positive or oxidizing and the other with a strongly negative or reducing potential. The e.m.f. of the resulting cell is then the difference between the two potentials. A large number of systems may be theoretically set up, but in the case of most of them they will be found unsuited for practical use because of chemical activity of the anode material, costliness, or tendency to passivity, as well as other reasons. In practice zinc is almost invariably the soluble anode in a primary cell.

On the assumption that the electrode system and its resulting reactions are reversible, the e.m.f. of the cell may be calculated from the Gibbs-Helmholtz equation. If the cell has a zero temperature coefficient, the theoretical e.m.f. may be calculated from the heats of reaction of the materials of the cell. In practice, however, electrode systems never behave absolutely reversibly. At times the degree of irreversibility is considerable. Inasmuch as soluble anode materials are commonly used and oxygen evolution does not occur, passivity is the only irreversible effect to be feared. This is eliminated by the proper choice of metal and electrolyte. At the cathode where the process is a discharge of a metallic ion to metal, the reaction takes place very nearly reversibly. If, however, the cathode system is an oxidation-reduction electrode consisting of an oxidizing agent in contact with an indifferent electrode and serving to depolarize the discharge of hydrogen ions, a considerable overvoltage may be needed for the hydrogen discharge which will in turn lower the e.m.f. of the cell. If depolarizers are used, the e.m.f. of the cell is increased above the value corresponding to reversible cathodic hydrogen discharge. The reaction occurs so quickly between the depolarizer and the discharged hydrogen that the electrode never becomes saturated with the gas, and hydrogen discharge takes place at a less negative cathode potential. The more effective and rapid the action of the depolarizer and hydrogen, the lower is the hydrogen concentration in the electrode, the more positive the cathode potential, and the more nearly it reaches its equilibrium value.

Liquid depolarizers act more rapidly than do solids but tend to diffuse toward the anode which they may strongly attack. This may necessitate a diaphragm between the anode and the cathode, which increases the internal resistance of the cell. For satisfactory use solid depolarizers, in addition to reacting quickly with the discharged hydrogen, should have high electrical conductivity and make good contact. This may be obtained, as in the case of the dry cells, by adding graphite to the MnO_2 depolarizer.

Internal resistances in the cell lower the e.m.f. of the unit. These may be due to the type of construction, diaphragms, concentration changes, electrolyte resistances, as well as those of a film or polarization nature not taken care of by depolarizers.

Scores of different primary batteries were on the market a generation ago, but now only a few of commercial importance. At the present time the list includes the **Daniell** cell, the **Lalande**, and the **Leclanché** of which the dry cell and the **Féry** cell are modifications. Of the entire group, the dry cell is by far the most important, followed by the Lalande, often termed the caustic soda primary battery. Table 13 lists laboratory and commercial cell systems.

The Daniell cell consists of the system

$$Zn \mid ZnSO_4 \mid CuSO_4 \mid Cu$$

and the chemical reaction is

$$Zn + CuSO_4 = ZnSO_4 + Cu$$

The cell e.m.f. is a function of the concentration of the $ZnSO_4$ solution, having a maximum value of about 1.14 volts. If the $ZnSO_4$ solution be not acid with H_2SO_4, the voltage is 1.07. Inasmuch as the chemical reactions between the constituents of the cell continue whether the cell be used or not, the setup is not adapted to stand on open circuit.

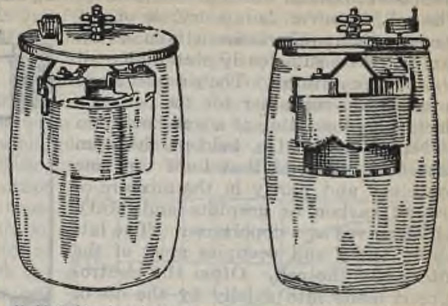
Table 13. Primary Batteries

Type	System	E.m.f.
Bunsen	HgZn | 1 part H_2SO_4 + 12 parts H_2O | concentrated HNO_3 | C	1.94
Bichromate or Poggendorf	HgZn | 1 part H_2SO_4 + 12 parts H_2O | concentrated solution $Na_2Cr_2O_7 + H_2SO_4$ | C	2.00
Daniell or gravity	HgZn | 5% solution $ZnSO_4.6H_2O$ | saturated solution $CuSO_4.5H_2O$ | Cu	1.07
Féry	Zn | 12% NH_4Cl solution | depolarizing C	1.2
Grove	HgZn | 1 part H_2SO_4 + 12 parts H_2O | fuming HNO_3 | Pt	1.66
Leclanché	HgZn | 20% NH_4Cl solution—MnO_2 | C	1.5
Lalande	Zn | 18–19% NaOH solution-oxides of Cu | Cu	0.95
Dry cell	Zn | NH_4Cl—$ZnCl_2$—MnO_2 | C	1.53
Le Carbone	Zn | 20% NaOH | special C	1.4
Air cell	Zn | solution NaOH | special C	1.25

A modification of the Daniel cell is known as the **gravity battery**, shown diagrammatically in Fig. 3. The heavier $CuSO_4$ solution is placed at the bottom of the cell in contact with a spread-out copper-sheet electrode to which a rubber-covered wire connection is made. The zinc electrode is frequently in the form of a cast crowfoot suspended at the top of the jar, surrounded by $ZnSO_4$ solution which has been carefully poured on top of the $CuSO_4$ solution. The edges of the jars are ordinarily coated with paraffin to prevent creepage of $ZnSO_4$ crystals over the top, and evaporation is reduced by covering the solution with a thin layer of mineral oil. It is estimated that under average conditions only about 30 per cent of the zinc is electrochemically utilized.

FIG. 3.—Gravity cell.

The **Lalande** cell has a soluble zinc anode, an alkaline electrolyte, and a solid cathodic depolarizer. The system is Zn | alkali solution-oxides of copper | Cu. The electrolyte is usually 18 to 19 per cent NaOH. The construction of a typical cell is shown in Fig. 4. The CuO acts as a depolarizer and is made either in a compressed or in a loose form. In the manufacture of the compressed form, the CuO is first made in a very fine powder, mixed with a binder, and compressed under heavy pressures, then given a baking treatment, after which the outer surface is metallized by partial reduction to lower the resistance, inasmuch as CuO alone is a very poor conductor. The compressed oxide element is usually a flat plate or a hollow cylinder. In commercial cells the zinc electrode is usually cast in a cylindrical form and amalgamated with mercury, so that the electrode contains as much as 2.5 per cent of the latter metal. The CuO and zinc electrodes are mounted in a glass jar made of heavy construction to withstand the corrosive action of the caustic solution, and the electrolyte covered with a layer of mineral oil to prevent absorption of CO_2 from the atmosphere by the solution as well as the evaporation of the solution. Purity of materials is quite important for the successful operation of the cell. In the **Edison** type, the electrodes are flat plates, the compressed copper oxide being in the center of the cell on each side of which is a zinc plate cast with ribs. The construction of the cell is shown in Fig. 5.


Unit Cylinder Cell, conforming to A.R.A. Specifications

Cut of Exhausted Cell; Normal Electrolyte

Fig. 4.—Lalande cell.

The chemical reaction of the cell may be given as

$$Zn + 2NaOH + CuO = Na_2ZnO_2 + H_2O + Cu$$

The e.m.f. of the cell is about 0.95, but the terminal voltage drops to less than two-thirds of this when heavy currents are furnished. The internal resistance of the cell is low. It is adapted to both closed- and open-circuit work. Because of its simple construction, ready working, relative cheapness, and despite its low voltage, it has found extensive use for the operation of signal systems and for railway work. Batteries are commercially manufactured in sizes from 75 amp.-hr. up to cells with a rated capacity of 1,000 amp.-hr. When cells are run down, fresh zinc plates are added and the oxide electrode, which has been largely reduced to copper, washed and reoxidized by heating at 150°C.

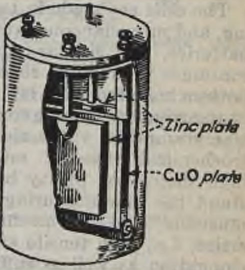


Fig. 5.—Edison cell.

The **Leclanché** cell is of the system

$$Zn|NH_4Cl - MnO_2|C.$$

The electrolyte is a strong NH_4Cl solution, usually about 20 per cent, to which various hygroscopic substances, such as glycerin, $ZnCl_2$, or at times $CaCl_2$, may be added to lessen the tendency of the cell to lose water. The reaction of the cell is given as

$$Zn + 2NH_4Cl + 2MnO_2 = Zn(NH_3)_2Cl_2 + H_2O + Mn_2O_3$$

In the original form, the carbon rod was contained in a porous cup filled with crushed carbon and MnO_2, the mixture being tamped to obtain intimate contact. In later forms the MnO_2 and carbon were molded by the use of a binder into a cylindrical form, the zinc rod being suspended centrally as in Fig. 6. The cell e.m.f. is about 1.5 volts, but the terminal voltage drops rapidly when heavy currents are drawn from the cell, showing values of 1.1 to 1.2 volts at currents as low as 0.1 to 0.2 amp. for an ordinary cell. The cell is suitable for open- or closed-circuit work if large currents be drawn intermittently and only for a short time. The dry-battery modification of the Leclanché cell has largely replaced this unit in commercial work.

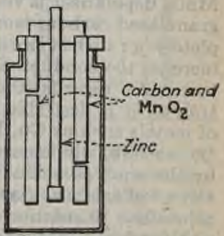


Fig. 6.—Leclanché cell.

Dry Cells. The dry cell is so built that its electrolyte is contained in an absorbent material which prevents its spilling out with the cell in any position. The cell, however, is not dry, as one of the essential requirements in its construction is to be sufficiently wet under all ordinary conditions. The zinc electrode is made the container for the cell, the electrolyte consisting of a water solution of NH_4Cl and $ZnCl_2$, held partly in an adsorbent material that lines the zinc container and partly in the mixture of ground carbon or graphite and MnO_2 which serves as a depolarizer. The latter is bulky and occupies most of the interior of the cell. Often the electrolyte is made into a jelly by the use of colloidal materials such as gum tragacanth, agar-agar, flour, or, more commonly, starch. In American practice the cell is completely sealed at the top, the newer types having a vent for the escape of gas.

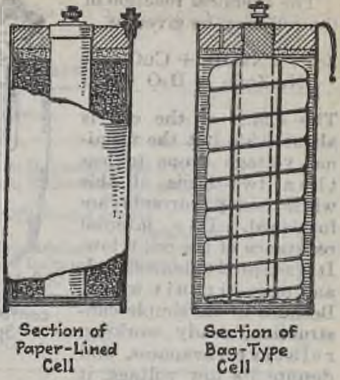


Fig. 7.—Dry cells.

The cells are made in two general sizes, the larger ones for ignition, signaling, and miscellaneous intermittent use, the smaller ones for flashlights, radio batteries, and similar purposes. In the case of the larger cells the zinc container is made of sheet with the bottom soldered with lap seams, but in the smaller cells the containers are zinc stampings. The zinc, for electrochemical reasons, must have a high degree of purity but to withstand the manufacturing operations must also have good mechanical properties, *i.e.*, high tensile strength and elongation as well as stiffness. The use of pure zinc avoids galvanic couples and local corrosion of the metal.

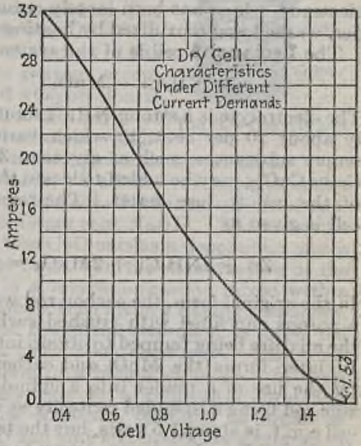


Fig. 8.—Dry-cell characteristics.

The electrical conductivity of the MnO_2 depolarizer is very low, so that granulated carbon, more or less completely graphitized, is added to increase the conductivity of the mixture. As regards the NH_4Cl electrolyte, it is desirable that it be free of metals such as Cu, Pb, Fe, As, Ni, Co, and Sb, which may be plated out by the zinc, as well as free from negative radicals, such as sulfates, which form compounds less soluble than the chlorides. Insulation and sealing compounds are usually resin, sealing wax, or bituminous pitches with fillers such as ground silica, fibrous talc, and coloring matter added. The entire cell is ordinarily insulated by a paper container or carton surrounding it.

Two general methods of manufacture are employed. In the first, used for the large cells, the zinc container is lined with a sulphite- and ground-wood-pulp board into which the MnO_2—NH_4Cl—$ZnCl_2$ mixture is tamped around the central carbon electrode, after which the cell is sealed with a sealing compound. In the manufacture of cells of the flashlight type, the carbon rod with its surrounding mixture of depolarizer and electrolyte is wrapped in a muslin bag and tied with a string, forming a unit which is placed in the zinc can, leaving sufficient space between the two for the electrolyte in the form of a paste. A solution of ammonium chloride and zinc chloride used for the electrolyte is thickened with flour or starch. The so-called **desiccated cell** is manufactured dry, being either of the paper-lined or of the bag type, the cell being provided with an opening in the seal or the center of the carbon rod, through which the water necessary to make the cell active may be introduced. Daniels [*Trans. Am. Electrochem. Soc.*, **53**, 45 (1928)] has calculated that the energy of a 90-g. *D*-cell is equal to almost 13,000 watt seconds. Such a unit will deliver about 2.73 amp.-hr. before its voltage falls to 1.13 volts on a discharge through 83.3 ohms for 4 hr. a day. These conditions correspond to radio use. The average voltage during this discharge is 1.3. Table 14 gives the commercially important dry-cell sizes and voltages. Figure 8 shows dry-cell characteristics, and Fig. 9 shows the effect of temperature on the voltage and amperage of No. 6 dry cells.

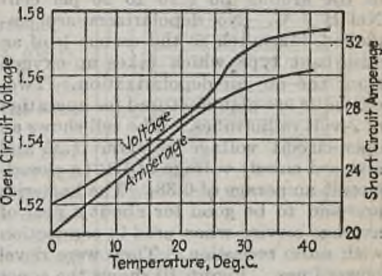


FIG. 9.—Temperature-current-voltage relations of No. 6 dry cells.

Table 14. Dry Cells

Type	Sizes				
	Diameter, in.	Height, in.	Diameter, mm.	Height, mm.	Minimum voltage
A	5/8	1 7/8	16	48	1.47
B	3/4	2 1/8	19	54	1.48
C	1 5/16	1 13/16	24	46	1.49
D	1 1/4	2 1/4	32	57	1.50
E	1 1/4	2 7/8	32	73	1.50
F	1 1/4	3 7/16	32	87	1.50
6	2 1/2	6	63	152	1.50

The **Féry** cell is a modification of the Leclanché type for use where very low currents are required, designed to avoid the use of cathodic depolarizers completely, relying on air dissolved in the carbon for depolarizing the hydrogen discharge. The type used on the French railways consists of a glass jar at the bottom of which a zinc-plate anode rests. External connection is made by a copper wire insulated from the electrolyte. The anode plate carries a cross-shaped insulator of synthetic plastic or ebonite on top of which the hollow carbon-cylinder cathode rests. The cathode is about half the diameter of the jar and is pierced with holes. It projects several inches above the surface of the electrolyte which is 12 per cent NH_4Cl. Oxygen for depolarization is adsorbed from the atmosphere by the carbon. A cell weighing 2.1 kg. has a 90 amp.-hr. capacity and an open-circuit voltage of about 1.2. In practice the cells are discharged at currents of 20 to 50 milliamp. and have a life of about 6 months.

The most recent development in primary cells has been the manufacture of so-called **breather batteries** or air cells of the system Zn | 10 to 20 per cent NaOH | C. No depolarizers are employed, inasmuch as the carbon is of an adsorbent type which takes up oxygen from the air for depolarization. Two-cell units are manufactured for operation of 2-volt radio tubes. The cell shows an open-circuit voltage of about 1.25 and a closed-circuit voltage of 1.12; a closed-circuit amperage of 0.38. The batteries are said to be good for about a year of average service when used in connection with radio reception. They were developed for rural districts not served by power lines. Figure 10 shows the construction of the air cell of the National Carbon Co. Figure 11 shows the voltage characteristics of this battery.

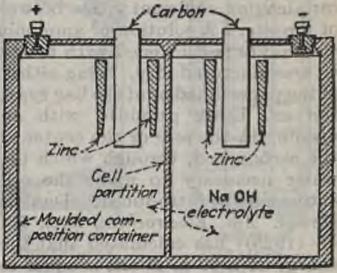


FIG. 10.—Air cell—two-cell battery.

Primary batteries of the adsorbent-carbon depolarizing type are also built for semaphore, highway flashing systems, lighthouses, railway-signal and similar work. A typical one is the **Le Carbone** cell, the construction of which is shown in Fig. 12, and the characteristics in Fig. 13. A cell whose cylindrical glass jar is approximately 10 in. high and 7 in. in diameter is rated at 500 amp.-hr., with initial closed-circuit voltage of over 1 volt at rates of discharge not greater than 3 amp. The positive or carbon element has a capacity of 2,000 to 2,500 amp.-hr., while the zinc circular element weighing about 825 g. lasts 500 amp.-hr., after which it is renewed. The cell holds about 4 l. of a 20 per cent NaOH electrolyte, which is renewed every 500 amp.-hr. The open-circuit voltage of the cell is 1.4 to 1.5 volts.

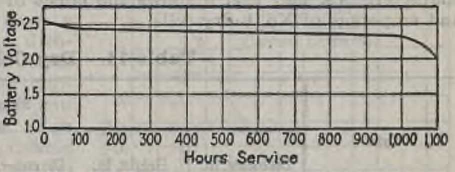


FIG. 11.—Voltage characteristics of air cell.

Secondary Cells. Cells which are reversible to a high degree, in that the chemical conditions may be restored by causing current to flow into the cell on charge, are used as storage batteries or electric accumulators. The Daniell or gravity cell as well as the Lalande is reversible to a high degree, but these cells have practical disadvantages preventing their use as storage batteries. The form in widest commercial use is the Pb—H_2SO_4 type. The only other form of any prominence is the Ni-Fe-caustic cell, often termed the **Edison** battery.

The positive pole of a battery is that from which the current flows into the external circuit. In storage-battery practice, a positive plate is one which is connected to the positive pole and a negative plate one which is connected to the negative pole.

The Pb—H_2SO_4 battery system is $PbO_2 \mid H_2SO_4 \mid$ sponge Pb. On discharge of the battery both the peroxide on the positive plate and the lead on the negative plate are quantitatively converted into $PbSO_4$ according to the reactions

$$PbO_2 + H_2SO_4 = PbSO_4 + H_2O + O$$

and

$$Pb + H_2SO_4 = PbSO_4 + 2H$$

which may be combined into the reaction,

$$PbO_2 + Pb + 2H_2SO_4 = 2PbSO_4 + 2H_2O$$

which, when read from left to right, is the equation of discharge and, inversely, the reactions during charge.

The active materials, the PbO_2 on the positive plate and the sponge lead on the negative plate, are crystalline in structure and the intergrowth of the crystals holds the masses together. It is probable that the positive active material is a hydrated peroxide of lead as it exists in the cell. Many commercial modifications of plates have been proposed and are in use either for forming the active materials in place or for applying them and holding them in place by some mechanical structure. In general the plates may be of the Planté type comprising a mass of lead, usually of flat form, with a highly developed surface on which the active material is electrochemically formed as a coherent layer, or pasted plates in which the active materials are cemented masses supported in a grid, usually of lattice form.

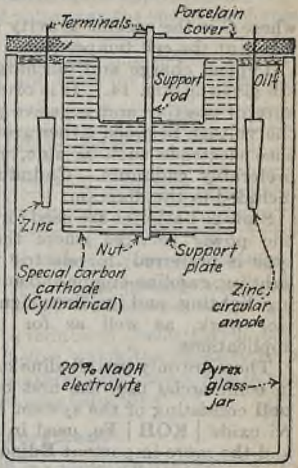


FIG. 12.—Le Carbone caustic cell.

The capacity of a storage cell is stated in ampere-hours at some normal rate of discharge, the 8-hr. rate being standard with lead cells of the stationary type. The capacity of a cell with a definite type and thickness of plate is in proportion to the plate area. The e.m.f., or open-circuit voltage, of any storage cell depends wholly upon its chemical constituents and not in any way upon the number, or total area, of the plates. It varies further with the strength of the electrolyte, temperature, and to a minor extent with the state of charge of the plates, internal resistance of the cell, polarization, and acid-concentration effects. Per ampere-hour of discharge, the amount of active material converted into $PbSO_4$ is 0.135 oz. sponge lead and 0.156 oz. PbO_2, independent of the rate of discharge. The amount of active material actually present in the plate is some three to six times that which under normal discharge of the cell is converted into $PbSO_4$. Part of this excess is present to give long life to the plates.

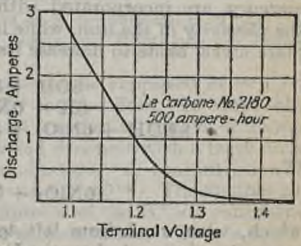


FIG. 13.—Current-voltage relations of Le Carbone cell.

The open-circuit voltage of a lead cell varies from 2.06 to 2.14 according to the strength of the electrolyte and the temperature and may be calculated from the formula

$$E = 1.850 + 0.917\ (G - g)$$

where G is the specific gravity of the electrolyte and g the specific gravity of water at the cell temperature.

Typical charge and discharge curves for the stationary type of lead cell are given in Fig. 14. It is often desirable to determine the relative performance of positive and negative plates in a cell. This may be done by taking the voltage between either group and a reference electrode such as zinc, sponge lead, or preferably cadmium. Cadmium curves are included in the diagram.

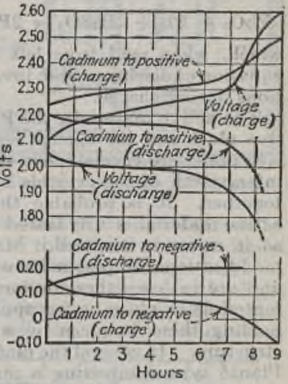


FIG. 14.—Typical charge-discharge curves of lead storage battery.

Storage batteries are used for stand-by electric power services where the pasted-plate type is preferred; for electric vehicles; train lighting; gasoline-engine and automobile starting, lighting, and ignition; signaling and control work, as well as for many other applications.

There are only two alkaline storage batteries in commercial use, the first being the **Hubbell** consisting of the system Ni threads and Ni oxide | KOH | Fe, used in miners' lamps, and the more important **Edison** battery consisting of the following system: finely divided Ni + Ni peroxide | 21 per cent KOH | finely divided Fe. The active materials of the Edison battery consist of nickel peroxide for the positive plate and finely divided iron for the negative. Small amounts of LiOH are added to the electrolyte, and certain amounts of mercury are incorporated with the iron of the negative plate to overcome the passivity of the iron, while layers of flake nickel are added to the positive plate nickel oxide to increase its conductivity. The reactions of the cell are:

$$8OH + 3Fe = Fe_3O_4 + 4H_2O$$
$$8K + 6NiO_2 = 2Ni_3O_4 + 4K_2O$$
$$8KOH + 6NiO_2 + 3Fe = Fe_3O_4 + 2Ni_3O_4 + 8KOH$$

or

$$6NiO_2 + 3Fe = Fe_3O_4 + 2Ni_3O_4$$

which, when read from left to right, are the reactions of discharge and, inversely, those of charge. It is probable that the iron and nickel oxides are both hydrated. In charging the battery, the electrolyte density does not change as in the lead storage cell, but concentration changes of the electrolyte in the pores of the active materials do occur with perhaps the formation of higher oxides of nickel.

The positive plate consists of a nickel-plated steel frame into which are pressed perforated tubes filled with alternate layers of nickel hydrate and metallic nickel in very thin flakes. The tube is made from a thin sheet of steel, nickel-plated and perforated, and has a spirally lapped joint. The negative plate consists of a grid of nickel steel with oblong openings into which perforated steel boxes containing finely divided iron with mercury are placed.

Characteristic normal charge, and discharge, curves for the Edison battery are given (Fig. 15). The average voltage on discharge is approximately 1.2, the initial open-circuit voltage 1.5, and the final voltage at the end of discharge a little less than 1.

Edison cells are used for ignition and lighting of gasoline motor cars but because of their high internal resistance are not used for motor starting. They find application in electric vehicles, storage-battery street cars, mining locomotives, and industrial trucks. In contradistinction to lead storage batteries, they are not used for load regulation in power systems because of their heavy voltage drop at high discharge rates. The commercial cells show ampere-hour efficiencies of 82 per cent and watt-hour efficiencies of 60 per cent, and an average capacity of about 13 watt-hr. per lb. of cell. The Edison cell is especially sensitive to reduction of electrolyte temperature, showing a critical point about 50°F. below which the capacity falls off very rapidly. Its performance at low temperatures, such as are met in the northern parts of the United States, is therefore unsatisfactory.

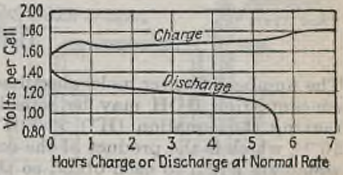


FIG. 15.—Charge-discharge curves for Edison alkaline battery.

Electrolysis and pH Measurement

Electroanalysis. Electrochemical methods are employed to a considerable extent in analytical work for the determination of metals from solution, the separation of these metals one from the other by deposition at controlled voltages. Gauze electrodes and rotating anodes are employed so that high current densities may be used to shorten the necessary time for analysis. In other cases mercury cathodes, which adsorb metals of the salt solution being electrolyzed, are advantageous. For specific methods, reference should be made to the standard works on analytical chemistry.

pH and Its Measurement. This section is a condensation of the discussion of pH in various publications of the Leeds and Northrup Co. and is used by their permission.

The concentration of H^+ ion in any solution may be expressed in terms of the normal H^+ ion solution containing 1 g. H^+ ion per liter. In most solutions the H^+ ion concentration is only a small fraction of that in a normal solution and is expressed as powers of 10 to avoid decimals with a large number of ciphers, or fractions with large denominators. A concentration of 0.0001 g. H^+ ion per liter thus becomes $1/10^4$ or 1×10^{-4}. The symbol pH is used to designate the logarithm of the reciprocal of the H^+ ion concentration, or the negative logarithm of the H^+ ion concentration. Thus a 0.0001 N solution of H^+ ion equals 1×10^{-4}, or $-\log (H^+) = 4 = \log (1/H^+)$; therefore $pH = 4$. Concentrations that are uneven decimal fractions of normal can also be expressed in pH units. Thus for a solution in which the concentration is 2.73×10^{-4} N, the pH number is 3.566. This can be proved by the use of a logarithm table, which shows that $\log 2.73 = +0.434$ and $\log 10^{-4} = -4.000$. Since the numbers are to be multiplied, the logarithms are added, so $-4.000 + 0.434 = -3.566$. Therefore $\log (H^+) = -3.566$, so $-\log (H^+) = 3.566$, and $pH = 3.566$.

If it be desired to know the actual figure for the H^+ ion concentration when only the pH value is given, it can be found by the reverse of the calculation just given. Thus, 9.63 pH means that $(H^+) = 1 \times 10^{-9.63}$. The exponent $-9.63 = -10 + 0.37$, hence $10^{-9.63} = 10^{-10} \times 10^{+0.37}$. The logarithm table shows that the exponent 0.37 corresponds to the number 2.34. Therefore $(H^+) = 10^{-9.63}$ is the same as $(H^+) = 2.34 \times 10^{-10}$.

In a similar manner the alkalinity of a base may be expressed as $pOH = \log 1/(OH^-)$. In an acid-base equilibrium water is formed and neutrality is reached when

$$\frac{(H^+) \times (OH^-)}{(HOH)} = K$$

The number of water molecules dissociated is so small in comparison that the concentration HOH may be considered as constant and combined with K, making the equation $(H^+) \times (OH^-) = K(HOH) = Kw$. At 25°C., Kw is 10^{-14}, which is the product of the concentration of the H^+ ions and OH^- ions which are equal to each other, so that the concentration of each is 10^{-7} and in water, $pH = 7$. In an acid solution the concentration of H^+ ions is greater than that of water, due to the dissociation of the acid and according to its dissociation constant. In an analogous manner in basic solutions the OH^- ion concentration is greater than that of water. But since water is present, there must be some OH^- ions in any acid solution and some H^+ ions in any basic solution, the concentration of one of these varying inversely with the concentration of the other and the product of the two concentrations being equal to 10^{-14}. Then $(H^+)(OH^-) = 10^{-14}$ and $\log (1/H^+) + \log (1/OH^-) = 14$, and $pH + pOH = 14$.

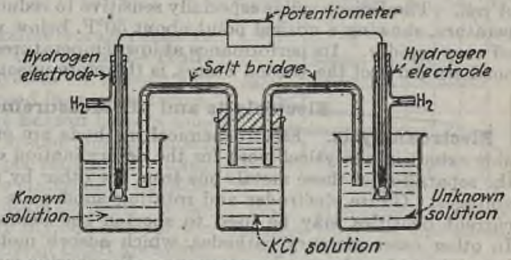


Fig. 16.—Theoretical hydrogen ion concentration setup. (*Leeds and Northrup.*)

It is evident that if the concentration of either ion is known, that of the other can be computed so that the reaction of any solution, whether acid, alkaline, or neutral, can be expressed in terms of pH. An acid-alkaline scale can be set up in terms of H^+ ion concentrations with a $pH = 0$ at one end representing a normal H^+ ion solution, and a $pH = 14$ at the other end representing the H^+ ion concentration of a normal OH^- ion solution. Any pH number from zero to 7 thus indicates acidity with decreasing acidity as the number increases. pH 7 indicates neutrality and any pH number between 7 and 14 indicates an alkaline solution with increasing alkalinity (or decreasing acidity) as the number increases.

H^+ ion concentration in a water solution can be determined electrically by the use of two hydrogen electrodes in contact with the solution and the voltage developed by the two electrodes determined by the use of a potentiometer. Such an arrangement is shown in Fig. 16. From the following formulas the H^+ ion concentration or pH can be calculated.

$$\frac{V}{0.0001983T} = \log \frac{C_n}{C}$$

where T is the absolute temperature, V the voltage, C_n the known concentration, and C the unknown. At 25°C. for the temperature of the two solutions, the equation becomes

$$\frac{V}{0.0591} = \log \frac{C_n}{C}$$

If the known solution is a normal H^+ ion solution, then $C_n = 1$ and $\log 1/C = \text{pH}$, and the equation becomes

$$\frac{V}{0.0591} = \text{pH}$$

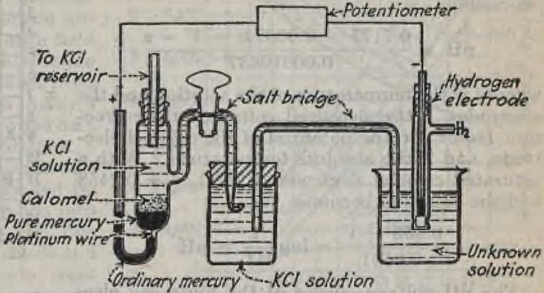


FIG. 17.—Calomel electrode setup for hydrogen ion measurement. (*Leeds and Northrup.*)

A more convenient arrangement substitutes a calomel electrode for one of the hydrogen electrodes as a result of which the arrangement becomes that of Fig. 17. The formula then becomes

$$\frac{V - v}{0.0591} = \log \frac{1}{C} = \text{pH}$$

The potential of the calomel electrode varies with its temperature as well as with the KCl concentration, as tabulated.

Table 15. Variation of Calomel-electrode Potential with Temperature and KCl Concentration

Calomel electrode	Potential at 20°C.	Potential at 25°C.	Potential at 30°C.
Tenth normal	0.3379	0.3376	0.3371
Normal	0.2860	0.2848	0.2835
Saturated	0.2496	0.2458	0.2420

For many practical purposes in industry the quinhydrone electrode is substituted for the hydrogen electrode. The basis of this electrode is a piece of platinum or gold exactly the same as used for the hydrogen electrode, but the surface of the metal is not platinized and it is not supplied with gaseous hydrogen. Instead, a small quantity of quinhydrone (benzoquinhydrone) is dissolved in the solution, and, within certain pH ranges, the electrode in the solution acquires a potential that is definitely related to the H^+ ion concentration of the solution. The potential is measured against that of a calomel electrode, and the pH value is found from the measured voltage in the same manner as with a hydrogen electrode. Quinhydrone is so slightly soluble in acid that only small quantities are necessary to saturate the solution. In solutions more alkaline than pH = 8, quinhydrone is more soluble, dissociates, and becomes oxidized. The quinhydrone electrode potential is altered to such an extent that the measured voltage is no longer a linear function of pH. The quinhydrone electrode is therefore not suitable for use in highly alkaline solutions.

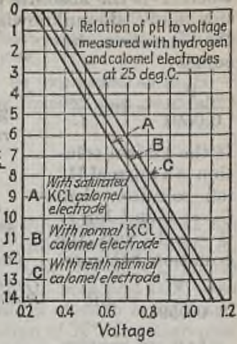


FIG. 18.—pH-voltage relation of hydrogen electrode. (*Leeds and Northrup.*)

The relation of voltage to pH is given by the expression

$$\mathrm{pH} = \frac{0.7177 - 0.00074t - V - v}{0.0001983T}$$

where t is the temperature of the solution and the electrodes, V the measured voltage, v the correction factor for the potential of the calomel electrode, and T the absolute temperature. With a saturated calomel electrode at 25°C., $v = 0.2458$ and the equation becomes

$$\frac{0.453 - V}{0.0591} = \log\frac{1}{\mathrm{H}^{+}} = \mathrm{pH}$$

The pH-voltage curves of the hydrogen electrode, calomel electrode arrangement and the quinhydrone electrode, saturated KCl calomel electrode, are given in Figs. 18 and 19.

Glass electrode is a Na_2O, CaO, SiO_2 glass bulb, thin walled, blown on the end of a glass tube and filled with pH 1 HCl. A quinhydrone electrode in this solution is the electrode terminal. Voltage measurements through the glass wall are made with a calomel reference electrode in the solution under test.

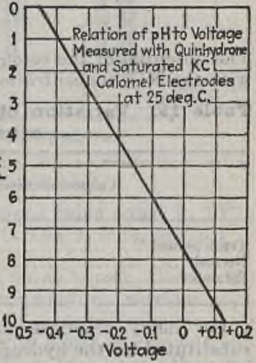


FIG. 19.—pH-voltage relation of quinhydrone electrode. (*Leeds and Northrup.*)

Antimony electrode, used for industrial controls, consists of prepared Sb with its end immersed in the solution whose pH is being measured.

Electrolytic Hydrogen and Oxygen Production

Hydrogen and Oxygen Production. Commercial hydrogen and oxygen cells are built entirely of iron or steel, insulating materials, and asbestos cloth for diaphragms. Every effort is made to have simple and inexpensive construction. Great care is taken to reduce all contact voltages to a minimum, in order to obtain the lowest possible cell voltages. Present-day cells are so well designed that per unit of current almost theoretical yields of the gases are obtained. The electrolyte ordinarily employed is an alkaline solution of either 15 per cent NaOH or its equivalent KOH. Sulphuric acid electrolytes are no longer used. Theoretical data for electrolytic hydrogen production are tabulated, and operating data for a typical cell are plotted in Fig. 20.

In general the cells use iron electrodes, the anode of which is ordinarily nickel plated to reduce the oxygen overvoltage. In the **Levin** cell the cathode is plated with cobalt to reduce the hydrogen overvoltage.

Two typical cells will be described. The **Knowles** cell consists of an outer tank containing a number of sheet-iron, or steel, gas-collecting hoods, rectangular in cross section and of considerably greater length than breadth. They alternately contain anodes and cathodes, each held in its hood by two conducting bolts which are insulated from the bells. Alternate hoods carry asbestos extensions which serve as diaphragms.

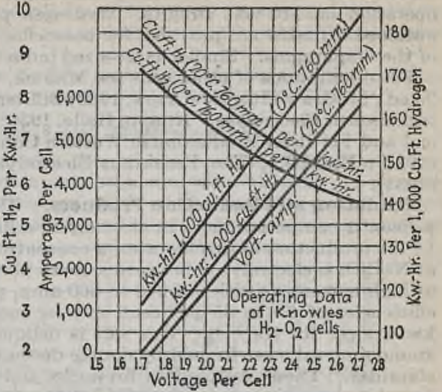


FIG. 20.—Current, voltage, and gas volume relations for hydrogen-oxygen cells.

Table 16. Electrolytic Hydrogen-cell Constants (*Knowles*)

Quantity of hydrogen liberated per 1000 amp.-hr.		Amp.-hr. required to liberate unit mass or volume of hydrogen, per	
Grams	37.65	Gram	26.56
Pounds	0.0830	Pound	12,050
Cubic feet at 20°C. and 760 mm. pressure saturated with water vapor	16.25	Cubic foot at 20°C. and 760 mm. pressure saturated with water vapor	61.55
Cubic feet at 0°C. and 760 mm. pressure dry	14.79	Cubic foot at 0°C. and 760 mm. pressure dry	67.61
Cubic meters at 20°C. and 760 mm. pressure saturated with water vapor	0.4604	Cubic meter at 20°C. and 760 mm. pressure saturated with water vapor	2171
Cubic meters at 0°C. and 760 mm. pressure dry	0.4189	Cubic meter at 0°C. and 760 mm. pressure dry	2387

Table 17. Levin Cell Characteristics

	Type *A*	Type *B*	Type M-1250
Dimensions, in.	30 × 25 × 6¼	43 × 37 × 8½	
Weight, lb.	145	325	
Current, amp.	250	600	1250
Capacity per hr.:			
Oxygen, cu. ft.	2	4.8	10
Hydrogen, cu. ft.	4	9.6	20

The Levin cell consists of a thin vertical tank divided by two asbestos diaphragms into three compartments, the two outer ones being the anode and the inner the cathode compartment. The gases produced pass upward through sight indicators to collecting manifolds. The entire tank is closed. For laboratory use, multiple or filter-press types of cells are used in which the plates act as bipolar electrodes. They are separated by asbestos diaphragms with rubber packings at the point where they insulate adjacent plates. Each plate except the end ones has three holes, one on each side at the top to lead off the gases and one at the bottom for the electrolyte which may be a 15 to 30 per cent NaOH. They are designed with sufficient plates to allow operation on 110-volt circuits. Hydrogen purer than 99 per cent can be obtained if means are provided for balancing the pressures on opposite sides of the diaphragms. Similar large-sized industrial units are now in operation.

For descriptions of other cells see Mantell, "Industrial Electrochemistry," 2d ed., McGraw-Hill, New York, 1940; Billiter, "Die neueren Fortschritte der technischen Electrolyse," Knapp, Halle, 1930; Kirkpatrick, Pioneering Chemical and Fertilizer Production in Western Canada, *Chem. & Met. Eng.*, **38**, 626 (1931); Elworthy, The Pechkranz Electrolyzer, *Chem. & Met. Eng.*, **38**, 714 (1931).

Oxidation and Reduction Products. The *perchlorates*, *persulphates*, and *perborates* can be produced in electrolytic cells by anodic oxidation processes. In the production of perchlorates, a concentrated (60 to 70 per cent) solution of $NaClO_3$ is electrolyzed between smooth platinum anodes and iron cathodes at high current densities of 270 to 500 amp. per sq. ft., with over-all current efficiencies of 70 to 85 per cent and an energy consumption of 1.5 to 1.8 kw.-hr. per lb. $NaClO_4$. This salt is deliquescent, and the potassium and ammonium salts are formed by double decomposition from the corresponding chlorides. These are used in fireworks and the explosives industries. The persulphates are made by the use of $(NH_4)_2SO_4$ solution in an anodic compartment with H_2SO_4 in the cathodic compartment separated by a diaphragm. The electrolyte is kept at about 15°C., the anode platinum operated at a high current density, and the cathode lead of a much larger surface. The current efficiency exceeds 70 per cent. Simpler processes avoid the use of diaphragms by the addition of about 0.2 per cent K_2CrO_4 to prevent cathodic reduction, or else employ $KHSO_4$ and the persulphate product $K_2S_2O_8$ is precipitated. Current densities are of the order of 400 to 600 amp. per sq. ft. and energy consumptions for $(NH_4)_2S_2O_8$ about 1 to 1.2 kw.-hr. per lb. **Sodium perborate** is prepared either by the interaction of borax and H_2O_2 or by the electrolysis of sodium borate containing Na_2CO_3 at anodic current densities of 200 to 350 amp. per sq. ft., between platinum anodes and iron cathodes, with the addition of chromate to the electrolyte. Energy consumption is about 3 kw.-hr. per lb. sodium perborate.

Table 18. Operating Data

Type of cell	diaphragm, bifluid
Electrolyte	sodium acetate + Na_2CO_3
Feed	individual
Diaphragm	linen
Anodes:	
Material	lead
Number	18
Size	21 × 30½ × 1 in.
Weight	250 lb.
Support	cast lugs
Electrode connection	parallel
Cathodes:	
Material	steel sheet
Size	1⁄32 in. thick
Cell material	concrete
Cell connection	series

A small number of **organic compounds** are made electrolytically by cathodic reduction or anodic oxidation. Among these are bromoform, iodoform, chloral, and paraaminophenol.

Sorbitol and **mannitol** [hexahydric alcohols $C_6H_6(OH)_6$] are the electrolytic reduction products of glucose (U. S. Patents 1,612,361; 1,653,004; 1,712,951-2; 1,990,582) and are produced by batch processing by cathodic reduction in alkaline glucose solutions containing Na_2SO_4 with Pb anodes and Pb-Hg cathodes.

White lead is made by the Sperry process by the electrolytic corrosion of lead anodes in a sodium acetate electrolyte containing Na_2CO_3.

Electroplating and Electrotyping

Electroplating is concerned with the production of metallic coatings on metal objects and, to a limited extent, on nonmetallic objects. In general, the metal plated is dissolved from an anode of the material and deposited on the object to be plated as cathode, although this arrangement may be modified and the metal plated may be obtained from a dissolved salt in the plating bath. Where soluble anodes are used, electrochemical action at the cathode is the reverse of that at the anode. The e.m.f. applied at the terminals of the cell is used to overcome concentration differences, voltage drops through the films adjacent to the anode and cathode, the ohmic resistance of the solution, the anodes, cathodes, connectors, and external circuit, plus the necessary overvoltages for the deposition of the particular ion in question. In the case of insoluble anodes where oxygen may be evolved, the overvoltage of oxygen is a factor as well as the decomposition voltage of the electrolyte. In general, the terminal e.m.f. of a plating tank is low and electroplating generators are built so as to supply either 6- or 12-volt high-amperage current. The capacity and cost per kilowatt relation of motor-generator sets are given in Fig. 21. The upper curve gives the cost to the user of standard size 6-volt motor-generator sets, including starting equipment but not including exciters commonly used on all sizes, 12 kw. and larger. Where more than 6 volts are required, it is customary to connect the two commutators of a generator in series and operate at 12 volts. From about 6 kw. up to the maximum sizes, the cost of the 12-volt single-commutator machines is approximately

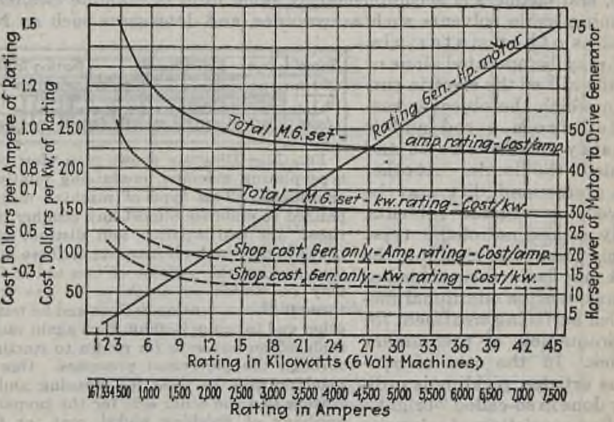


FIG. 21.—Characteristics of electroplating motor-generator sets.

85 per cent of that of the 6-volt units. The dotted-line curve gives the relation between the shop cost of the generator only and capacity, but does not include generator base, couplings, motor, and manufacturing overhead.

In commercial work, preparation of articles for plating is as important as the plating itself for the production of high-quality finishes. No general statement can be made as to methods used for preparing articles for plating, in that the material of the object, its condition, shape, size, composition, kind of surface, and other factors must be considered. Inasmuch as the plating will follow exactly the contours of the object to be plated, smooth finishes can be produced only on smooth-surface objects. In general, rough surfaces may be cut down or made smoother by sand-blasting, mechanical cleaning methods, grinding, and buffing with abrasives; or the object may be pickled, generally in H_2SO_4 or occasionally in HCl solutions. Removal of dirt, grease, and lacquers is accomplished by some form of alkaline cleaner, often containing oxide solvents such as cyanides, and detergents such as Na_3PO_4, as well as other materials.—These may be operated alone or with the aid of the electric current in which the object is suspended from a rod and is alternately made anode and cathode in the circuit. At other times grease and dirt may be removed by organic solvents, usually of the petroleum type. In some cases after cleaning, the object may be further pickled or else may receive additional mechanical surfacing treatment for the production of preliminary polishes. In the case of nonferrous articles, pickling is ordinarily done in so-called "bright" acid dips consisting of mixtures of concentrated H_2SO_4 and HNO_3 with the addition of small amounts of salt. Depending upon the object and the type of surface to be produced on the article, the preliminary flow sheet may vary widely in different industries. In order to avoid contamination of solutions as a result of the material being carried over when articles are transferred, and the introduction of acids into cyanide plating baths or vice versa, all operations, irrespective of their nature, are followed by washing or rinsing.

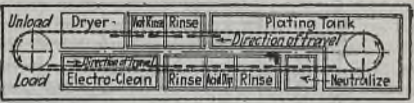


FIG. 22.—Diagram shows plan view of return type plating machine, containing relatively few operations. This type of machine may be expanded to embrace almost any number of operations. Or two separate and distinct operating routines may be built into one of these machines, thus serving the purpose of two units; *i.e.*, rack may be loaded on conveyor at one end, pass through the various operations and be removed at other end for color buffing, then again racked and replaced on conveyor for return to starting point through the prescribed processes. One side of machine may be used for cleaning and copper plating and the other side for the preparing and applying of finishing nickel coat, or for final chromium deposit. The adaptability of this machine is almost unlimited.

Plating baths may be considered to consist of a number of different parts: (1) the salt containing the metallic ion or radical; (2) an additional salt whose function is to increase the conductivity of the bath; (3) a material to effect the anode corrosion and prevent passivity of the anode; (4) a so-called addition agent to effect the type of deposit produced; and (5) a buffer material to maintain the proper pH of the solution. Some baths may contain all these while others may not. For example, a common nickel-plating solution will have the metal ion in the form of $NiSO_4.7H_2O$ (105 g. per l.), NH_4Cl or $(NH_4)_2SO_4$ (15 g. per l.) to increase the conductivity of the bath, $NiCl_2.6H_2O$ (15 g. per l.) to assist anode corrosion, and H_3BO_3 (15 g. per l.) which acts as a buffer to maintain the pH of the solution at 5.3. In the case of a tin bath, the tin salt would be furnished by Na_2SnO_3, the conducting salt by NaOH, which also assists anode corrosion, the addition agent to effect the deposit being glucose or other organic material.

X-ray examination shows that all electrodeposited metals have crystalline structures, the differences in their physical properties and appearances being caused by differences in the size and shape of the crystals. Plating baths are operated within definite current-density ranges beyond which poor types of deposits result. In general, for the production of fine grained or so-called amorphous deposits, which are readily polishable, plating baths show low ionic metallic concentration produced not by dilute solutions but either by the use of salts showing low ionization or salts whose ionization is depressed by the addition of another salt having a common ion, or by the employment of compounds producing the metallic ion not by primary but by secondary ionization, or by reduction at the cathode. This explains the widespread use of double salts such as $NiSO_4.(NH_4)_2SO_4.6H_2O$, complex compounds such as the double cyanides, and, in the specific case of chromium, chromic acid. Deposition potentials in general are lower than the decomposition voltage of water, but in the case of chromium and alloys, such as brass, the deposition potential is above that of the decomposition of water, and hydrogen is evolved at the cathode. This latter condition is essential for the deposition of

Table 19. Electroplating Baths (*Mantell*)

Metal	Bath type	Anode	Cathode c.d. amp. per sq. ft.	Temp., °C.
Copper	Acid sulfate	Cu	15–40	25–50
	Alkaline cyanide	Cu	3–14	35–40
Nickel	Single sulfate	Ni	5–20	20–30
	Double sulfate	Ni	3–6	20–30
	Sulfate-chloride	Ni	14–50	50–60
Iron	Chloride	Fe	100–180	90–110
	Double sulfate	Fe	20–30	20–30
Cobalt	Sulfate	Co	30–165	20–30
Chromium	Chromic acid	Pb	100–300	40–50
Zinc	Sulfate	Zn	12–30	20–30
	Sulfate (hot)	Zn	80–100	50–60
	Chloride	Zn	40–100	20–40
	Alkaline cyanide	Zn	8–20	40–50
Cadmium	Alkaline cyanide	Cd	10–50	20–30
Lead	Fluoborate	Pb	5–20	20–30
	Perchlorate	Pb	20–30	20–30
Tin	Alkaline stannite	Sn	10	50
	Alkaline stannate	Sn	5–15	60
Silver	Cyanide	Ag	3–8	15–25
Gold	Cyanide	Au	1–5	60–80
Platinum	Phosphate	Pt	1	70
Brass	Cyanide	Cu-Zn alloy	2–3	32–45

chromium, and about 70 per cent of the current is consumed in hydrogen evolution. In general, however, current efficiencies of plating baths are high, usually being above 90 per cent. A large number of metals are commercially plated. The general characteristics of the baths employed are given in Table 19. After plating and drying, the objects may be polished, usually with mild abrasives such as tripoli and rouge, on high-speed wheels.

Mechanical equipment for plating assumes many different forms. Large quantities of small objects are plated in barrels which rotate in the plating solutions. Larger objects are generally wired or hung on rods which in turn connect with cathode bars. In recent years there has been a tremendous development of automatic plating, in which the successive operations involved in the entire production flow sheet are mechanized by having the objects hung on rods suspended from conveyors with automatic mechanisms causing the object to pass through different baths such as cleaners, dips, plating solution, rising out of the tank at one end and dipping into the next one by a tripping-and-raising action connected to the conveyor. The plan view of a machine of the type widely employed in the automotive industries for plating, where the objects receive successive finishes of copper, nickel, and chromium, is shown in Fig. 22.

Commercial Thickness of Electroplates

Ni, rarely over 0.001 in.; usually 0.0005 in. or under.
Cr, generally 0.00002–0.00004 in.
Zn, usually over 0.0005 in.; frequently around 0.001 in.
Cd, usually 0.0002–0.0003 in.; rarely over 0.0005 in.
Au, usually less than 0.00005 in.

Composite Plates:

Cu – Ni – Cr, 0.0003, 0.0005, 0.00002 in.
Ni – Cu – Ni – Cr, 0.0002, 0.0003, 0.0003, 0.00002 in.

Conversion Factors for Electroplaters

Amp. per sq. dm. × 9.3 = amp. per sq. ft.
Amp. per sq. ft. × 0.108 = amp. per sq. dm.
G. per l. × 0.134 = oz. per gal.
Oz. per gal. × 7.5 = g. per l.

For thickness of 0.001 in. for any metal,

$$\text{Weight plate in oz. per sq. ft.} = \frac{\text{sp. gr. metal}}{12}$$

Typical Plating-bath Formulas

(Concentrations in avoirdupois ounces per gallon)

Chromium

Chromic acid, CrO_3	33
Sulphuric acid, H_2SO_4	0.34
Temperature, °F	113
Cathode current density, amp. per sq. ft	150–200

Copper

A. Cyanide

$CuCN$	3
$NaCN$	4
Na_2CO_3	2
$Na_2S_2O_3.5H_2O$	⅛ (optional for brightener)
Temperature, °C	40–50
Cathode current density, amp. per sq. ft.	3–6

B. Acid

$CuSO_4.5H_2O$	24
H_2SO_4	6
Glue	⅛ (optional for brightener)
Temperature	room
Cathode current density, amp. per sq. ft.	25–50

Nickel

$NiSO_4.7H_2O$	32
$NiCl_2.6H_2O$	2
H_3BO_3	4
pH	5.8–6.2
Temperature, °F	120–130
Cathode current density, amp. per sq. ft.	20

Brass

CuCN	2
$Zn(CN)_2$	1
NaCN	5
Na_2CO_3	1
Temperature, °C	40–50
Cathode current density, amp. per sq. ft.	2–4

Zinc

A. Cyanide

(1)		(2)	
$Zn(CN)_2$	8	ZnO	6
NaCN	3	NaCN	10
NaOH	7	NaOH	5
Temperature			room
Cathode current density, amp. per sq. ft.			10–20

B. Acid Sulphate

$ZnSO_4.7H_2O$	48
NH_4Cl	4
$NaC_2H_3O_2.3H_2O$	2
H_2SO_4	0.4
Temperature	room
Cathode current density, amp. per sq. ft.	15–30

Cadmium

	(1)	(2)*
CdO	3	6
NaCN	7	1.6
NaOH	2	4
$NiSO_4.7H_2O$	.....	0.13
Goulac	.....	1.5
Temperature	Room	Room
Cathode current density, amp. per sq. ft.	10	25

* U.S. patent 1,681,509.

Lead

Basic lead carbonate	20
Hydrofluoric acid, 50 %	32
Boric acid	14
Glue (optional)	0.025
Temperature	room
Cathode current density, amp. per sq. ft.	10–20

Tin

Sodium stannate	12
Sodium hydroxide	2
Sodium acetate crystals	2
Temperature, °C	60–80
Cathode current density, amp. per sq. ft	10–30

Gold

Gold fulminate, dwt	0.3 or 5
NaCN	2
$Na_2HPO_4.12H_2O$	0.5
Temperature, °C	60–80
Cathode current density, amp. per sq. ft	1

Silver

	Strike	Plate
NaCN	16	5
AgCN	¾	4
Temperature	Room	Room
Cathode current density, amp. per sq. ft.	5+	3–5

The relation of time of plating, thickness of deposit, and current in commercial baths for nickel is given in Fig. 23, for copper in Fig. 24, for zinc in Fig. 25, and for cadmium in Fig. 26.

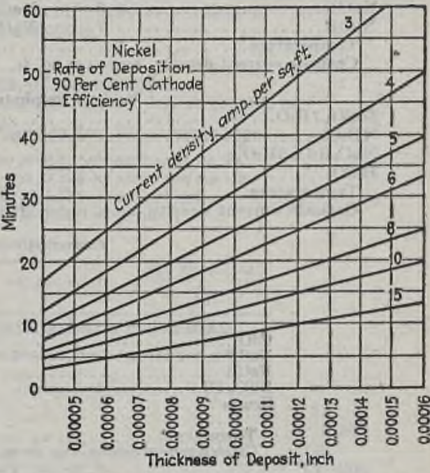


FIG. 23.—Plating characteristics of nickel bath.

Electroplating on Aluminum. In any application of electroplating on aluminum, the type of plate should be selected for the particular job. It is possible to apply a flash coating of zinc to aluminum and then plate with nickel, copper, or other metals. Unfortunately the corrosion resistance of such a plate is poor and suited only for indoor service. The zinc alone does not suffer this disadvantage and is useful for special purposes. Chromium may be applied directly to aluminum, but it has not yet been possible to obtain quite as satisfactory luster as is possible when chromium is applied over nickel. There is a possibility, however, that such chromium plates will have application for special corrosion jobs, particularly against mild alkalies. For general use it is advised that nickel be first applied to the aluminum after a surface roughening, and then the desired metal should be applied to the nickel. The aluminum alloys are divided into three classes, each requiring a somewhat different plating procedure, as follows:

Plating 2S Aluminum.

1. Remove grease.
2. Rinse in clear cold water.
3. Dip for from 10 to 15 sec. in 1 part of 48 to 52 per cent hydrofluoric acid to 9 parts of water.
4. Rinse in clear cold water.

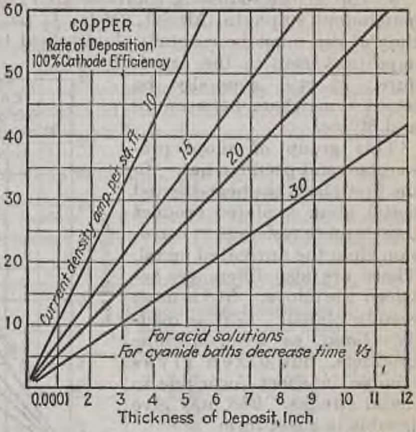


Fig. 24.—Plating characteristics of copper bath.

5. Dip in the following solution: nickel chloride 37 oz., HCl (sp. gr. 1.18) 0.2 gal., water 1.0 gal., temperature 75° to 80°F. The time of dip must be carefully determined as this is the important step in the procedure. This is accomplished by plating a series of specimens after different times in the dip and then bending or breaking them. A 35-sec. dip usually gives good results, although such factors as temperature, purity of the chemicals in the dip, and the temper and polish of the metal plated on affect the time required. The speed of the action of the dip may be regulated by the amount of acid and the temperature. Generally a time of dip of 15 to 40 sec. is desirable in production work. The actual composition of the nickel dip is not so important as the resultant adhesion of the electroplate.

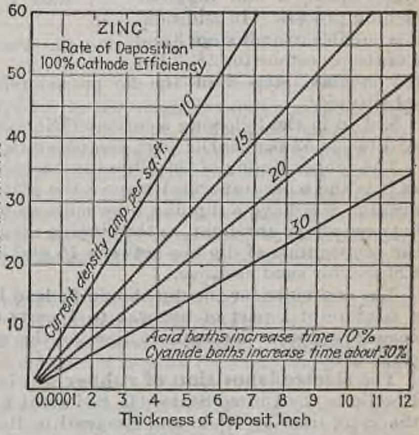


Fig. 25.—Plating characteristics of zinc bath.

6. Rinse in clear cold water.
7. Plate at 15 amp. per sq. ft. in the following nickel bath: single nickel salts 19 oz. per gal., magnesium sulphate 10 oz. per gal., ammonium chloride 2 oz. per gal., boric acid 2 oz. per gal. The time of plating should be determined to give the desired corrosion resistance. A time of plating of 1 hr. is suggested, but this can be reduced later if it is found that a thinner plate meets the service requirements.
8. Rinse in cold water and dry by means of a hot-water dip.
9. Polish nickel.
10. Clean and plate with any desired metal.

Plating 3*S* and Strong Alloys. For this group of alloys a different etching process is needed, otherwise the plating procedure is unchanged. Step 5 (as given above) will therefore be rewritten for these alloys for substitution in the above procedure.

5. Dip in the following solution: HCl (sp. gr. 1.18) $\frac{1}{2}$ gal., water $\frac{1}{2}$ gal., manganous sulphate ($MnSO_4.2H_2O$) $\frac{1}{2}$ oz., temperature 75° to 80°F. The time of dip must be carefully determined by experiment as this is the most important step in the procedure. It will generally be found somewhere between 10 and 60 sec.

This group of alloys presents several peculiarities. In the first place, the heat-treated metal gives a plated product that is more resistant to corrosion than the untreated metal. There are also differences between the alloys. 51*S* is most readily plated. 17*ST* is readily plated, as screw-machine products, but often gives trouble in sheet form due to metal streaks. 25*S* may give trouble in sheet form.

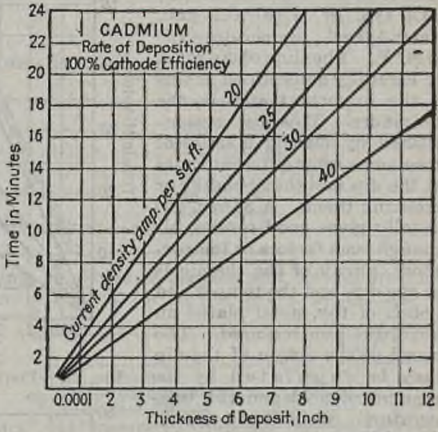


Fig. 26.—Plating characteristics of cadmium bath.

Plating Castings.* Castings require a still different etching process. In addition, it is possible to omit steps 3 and 4 of the procedure for 2*S*. For the etching, step 5 of the 2*S* procedure, the following method should be substituted:

5. Dip in the following solution: HNO_3 (sp. gr. 1.42) 3 parts, hydrofluoric acid (48 to 52 per cent) 1 part, temperature 75° to 80°F.

The proper timing of this dip must be carefully determined by experiment, as it is the most important step in the plating procedure. Each alloy composition will have a dipping time most suited to it, although the differences between alloys are slight, as the dipping range for each alloy is wide. Usually the proper time of dip lies between 15 and 30 sec. for die castings, and 60 to 120 sec. for sand castings.

The container for the dip should be lead lined. It should be painted with a mixture of 1 part of beeswax to 4 parts of paraffin. This is particularly necessary above the solution line, as the greatest attack of the lead takes place here.

The **electrodeposition of rubber** was invented in 1921 by Sheppard and Eberlin in the United States (U. S. Patent 1,476,374, Dec. 4, 1923) and about the same time by Klein and Szegvari in Budapest (British Patent 223,189). It has entered the technical field not as a competitor of mechanical methods but rather as a supplementary process and as a means of reducing costs.

---

* 195 alloy presents special difficulties. Even a 3-min. etch is not sufficient to secure adhesion, although the corrosion resistance appears fairly good. The procedure for the strong alloys may be applied to secure good adhesion, but it is sometimes difficult to secure uniform results.

Electrolytic deposits can be made from artificial rubber dispersions in aqueous solutions of slightly alkaline characteristics, as well as from natural latex preserved with ammonia. The rubber particles, existing as colloidal suspensions, are negatively charged and on the passage of current move toward the anode. If the current is reversed, the particles will also reverse their direction. With a steady flow of current in one direction, the rubber will deposit on the anode in a solid uniform layer. It has been found that in the Sheppard process sulphur, pigments, and accelerators can be mixed with the latex or rubber dispersion and a rubber compound deposited by electrolysis on metal or any other substance. This compound will possess approximately the same percentage composition as the total solids in the bath. It is washed and dried and can then be vulcanized.

In general a current density of 0.25 to 0.33 amp. per sq. in. proved satisfactory. The anode metal varies over a wide range, including lead, cadmium, zinc, tin, antimony, alloys of these metals, and iron and steel under certain conditions. Copper and copper alloys, however, proved objectionable. A typical plating bath follows:

Solids	Concentration, g. per 100 cc.	Percentage of total solids
Rubber from latex	8.00	53.3
Sulphur	0.30	2.0
Zinc oxide	1.50	10.0
Whiting	4.50	30.0
Carbon black	0.30	2.0
Paraffin wax	0.30	2.0
Tuads (accelerator)	0.03	0.2
Gum arabic	0.075	0.5
Total solids	15.00	100.0
Ammonia	0.16	

Ordinary fine sulphur, colloidal sulphur, or polysulphide sulphur may be used. Among the pigments and fillers, litharge, lithopone, white lead, magnesium carbonate, zinc oxide, titanox, clay, silica, certain carbon blacks, and a few others may be compounded without great difficulty. Softening agents and such materials as glue, dextrins, or gums for facilitating the dispersion of the added substances are readily incorporated. Accelerators insoluble in water may be employed, since they will be colloidally suspended in the solution. It has been found advantageous to homogenize the ingredients before mixing with the rubber emulsion.

Thickness of deposit is somewhat limited but not to so narrow a margin as might be expected from the nonconducting characteristics of rubber. Layers varying from 0.01 to 0.15 in. have been produced. The bath has sufficient throwing powers to effect a uniform coating on edges, corners, etc. Adequate stirring is of course important for smoothness of deposit.

Rubber and rubber compounds may be deposited not only as a coating on metals or other materials, but they may also be plated on a mold and later stripped off. Among the applications of the process described are the production of coated fabrics for automobiles; waterproofed textiles; coated metal wire for insulation and protective purposes; coated metal apparatus such as rubber-lined tanks, etc.; the direct production of rubber gloves, inner tubes for tires, and the like, as well as rubber sheets and rubber slabs for stock in the rubber industry.

The economic advantage of electrodeposition, as compared to mechanical milling and spreading, lies in the reduction of power and labor. Mixing and compounding operations are cheaper if done by high-speed mills. Furthermore, electrodeposited rubber will generally have a greater strength than the mechanically milled product, since it has not been degraded by friction. Rubber latex as ordinarily shipped has a solid rubber content of about 35 per cent. By means of concentration processes in the course of development, rubber shipments containing as high as 60 per cent have been made. [Sheppard and Eberlin, Electrodeposition of Rubber, *Ind. Eng. Chem.*, **17**, 711 (1925); Sheppard, Electrodeposition of Rubber, *Trans. Am. Electrochem., Soc.*, **52**, 47 (1927); U. S. Patents 1,476,374, 1,580,795, 1,583,704, 1,589,324-5-6-7-8-9, 1,589,330-2, 1,548,689.]

Electrotyping. Electrotyping has as its object the reproduction of the printer's type setup, of engravings and medals, while electroforming produces or reproduces an object by electrodeposition. The mold of the object to be reproduced is first made either in soft metal by pressure, or by impressing the object in wax. In the latter case the mold is a nonconductor, but its surface is made conducting by coating it with graphite. In other cases the surface may be made conducting by causing a thin metallic deposit to be formed on the object, for example by the use of a $CuSO_4$ solution over the object, out of which the copper may be precipitated by the use of finely divided iron filings. One reason for the use of the second method is that deposition does not take place at once all over the graphited surface, but the deposit "grows" from a point where contact is made. In other cases lead molds coated with graphite may be used. After forming, the electrotype is removed from the copper mold and backed, to give the piece rigidity and strength, with a low-melting-point metal such as those of the tin-lead alloy type.

For accurate reproduction the mold becomes a negative of the object to be produced. The most important example of the use of electrodeposited negatives is in the phonograph industry, where an original wax record is first made and then coated with graphite, upon which copper is deposited to form a *master matrix* which is a negative. From this plate one or more master records are made by electrodeposition to give positives which serve as the forms from which the final or pressing matrices or *working masters* are deposited. Best results are obtained by making the initial deposit of nickel, often coated with chromium to increase resistance to abrasion. The accuracy of reproduction is evidenced by the fact that several "generations" of plates can be obtained with satisfactory results in the shape of the final pressed phonograph record made from the working masters.

In ordinary electrotyping work the electrolytes used are 8 to 10 per cent H_2SO_4 solutions saturated with $CuSO_4$ at room temperature. Current densities are relatively low.

Electrolytic Cleaning and Pickling. The electric current may often be advantageously employed in speeding up cleaning and pickling as well as in improving the quality of these operations. Grease, oil, dirt, and lacquers are removed by some form of alkaline cleaner. This may be operated alone or with the aid of the electric current, in which case the object is suspended in the cleaning solution from a rod and is made cathode or in other cases alternately anode and cathode in the circuit. In electrolytic cleaning, solutions containing a fairly high content of alkali are employed as electrolyte. Alkaline salts such as sodium phosphate (Na_3PO_4) or sodium carbonate (Na_2CO_3) are added to control the alkalinity of the solution and aid in emulsifying "dirt." Iron anodes or, more frequently, the iron tanks are used as anodes, the object to be cleaned being hung on the cathode bar. Sufficiently high e.m.f. is applied so that a current density of at least 10 amp. per sq. ft. (1 amp. per sq. dm.) is obtained on the cathode, at which a vigorous evolution of gas occurs. Cleaning takes place rapidly, partly as the result of the formation of free alkali at the cathode, but also due to the mechanical action of the hydrogen which is evolved on the metal surface and tends to lift off films of oil, grease, paint, and dirt, and to assist in their emulsification. The gas evolution causes agitation of the solution and hence constantly brings fresh portions of the bath to the cathode surface. When zinc, tin, and lead, or their alloys, are cleaned electrolytically, there is a tendency for them to dissolve in the alkaline solution and be redeposited in small amounts in thin films. These films may prevent adherence of electroplatings. Under such conditions the redeposited metal is removed by causing the piece to function as anode for a short time. Electrolytic cleaning has been applied industrially in operations not connected with electroplating, such as the removal of paint from paint cans so that they may be reused, cleaning of steel barrels, drums, and shipping containers.

It is commercially desirable to get the most effective use of the electrical energy employed. The cleaning bath should be of such composition and concentration as to have high conductivity and resultant low bath resistance and low voltage drop across the terminals of the tank. The current supplied to the bath should be such as to give the most effective gas-generation rate at the anode and cathode. A current density of at least 10 amp. per sq. ft. is necessary; but the current density may be pushed to such a high figure that the gas bubbles evolved at the electrodes may blanket the electrode surfaces and cause high voltage drops through the gas films formed adjacent to the electrode surface. Too high a current density will increase the voltage drop across the bath, with resultant increase of electrical energy consumption without increased efficiency. Actually, too high a current density and too rapid a generation of gas bubbles on the object being cleaned may result in decreased efficiency of cleaning and higher power costs than necessary.

Electrolytic pickling has recently found commercial employment, particularly for the removal of oxides on machined parts which have been heat-treated. The operation is analogous to electrolytic cleaning in that the object to be pickled is hung on the cathode rod and suspended in an acid electrolyte containing chiefly H_2SO_4. Operating temperatures vary from room temperatures to those near the boiling point, depending upon the object treated. Sufficiently high current density is applied so that hydrogen is copiously evolved at the cathode. The hydrogen in expanding pries off the scale, bit by bit, and bubbles to the surface. Lead anodes are commonly used. If the electrolyte contains chlorides as the result of addition of either HCl or NaCl, the anode will be attacked and a thin layer of lead deposited on the object being pickled. Experimental tests do not indicate any striking advantage of the electrolytic over chemical pickling methods as far as the amount of acid used is concerned, but it is possible to employ lower temperatures and obtain better surfaces with electrolytic pickling than by chemical methods. In the case of steel, the amount of iron dissolved by the two methods varies considerably with different kinds of scale. Electrolytic pickling in general is more rapid than chemical pickling.

Scale-removing methods have been devised which consist first of electrolytic pickling followed by electrolytic cleaning. In one method the object is hung as cathode in a lead- or rubber-lined tank with anodes of metals which form protective films on the object, such as lead which is commonly used, or tin or zinc. Electrolytic pickling is done in an electrolyte consisting of dilute H_2SO_4 and HCl and some NaCl at 65°C. Because of the presence of chlorides in the electrolyte and resultant anode corrosion, thin adherent metal films of lead are deposited on the object being pickled. This action continues until all the scale has been removed and all the surface coated. At this time the object is removed from the acid bath and placed as anode in an electrolytic cleaner containing soda ash, caustic soda, and a small quantity of Na_3PO_4 at 90° to 100°C. In this bath the lead coating deposited in the acid bath is removed, after which the object is withdrawn, washed, and dried. The process is stated to be free from the difficulties of pickling and hydrogen embrittlement. It need not be very carefully controlled, inasmuch as in the acid bath, after the first layer has been deposited, further action results in the deposition of porous spongy deposits without detriment to the object.

In the Bullard Dunn process (U. S. Patent 1,775,671, Sept. 16, 1930) applied to many objects such as heat-treated castings, forgings, etc., in the descaling bath lead or tin is used as anode, the electrolyte being (with Pb) 2.2 oz. salt, 11.0 fluid oz. 66°Bé. H_2SO_4, 3.5 fluid oz. 20°Bé. HCl, and 113.5 fluid oz. of H_2O. Of the anode surface 75 per cent is duriron and 25 per cent is lead. Current density is 60 amp. per sq. ft. and anode-cathode spacing is 8 to 10 in. at 6 volts across the bath. Deleading is done in an alkaline cleaner. In descaling, the object under treatment is the cathode; in deleading, it is the anode.

Electrolytic Polishing. Strip steel, rods and sheet, particularly of the nickel chromium or "stainless" alloys, are polished by anodic treatment in phosphoric acid solutions as a competitive method to mechanical surfacing.

Anodic oxidation of aluminum has received considerable study and attention. Films of oxide are produced on aluminum and its alloys for protection against corrosion, for electrical insulation, for decorative effects, and for other uses. Protective adherent oxide films can be produced anodically on aluminum and its alloys. These can be dyed, painted, oiled, or otherwise treated. A number of different methods have been devised. Chubb (U. S. Patents 999,749, 1,068,410, and 1,068,411) uses sodium silicate, ammonium borate, or other electrolytes; Mershon (U. S. Patent 1,065,704), borax; Presser (U. S. Patent 1,117,240), sodium carbonate; Abernathy (U. S. Patent 1,323,236), potassium permanganate and H_2SO_4; Flick (U. S. Patent 1,526,127) ammonia and ammonium sulphide, the article after treatment being dyed by acid dyes; Kujirai (U. S. Patent 1,735,286), oxalic acid or oxalates; Bengough and Stuart (U. S. Patent 1,771,910), 3 per cent chromic acid solution. Other processes employing H_2SO_4 electrolytes have also been developed. Airplane and dirigible parts of aluminum alloys are often anodically treated in a 3 per cent chromic acid solution in which the voltage across the bath is raised from 0 to 40 volts in 15 min., kept at 40 volts for 35 min., held at 40 to 50 volts for 10 min., and at 50 volts for 5 min., after which they are washed, dried, and oiled or greased. The coatings are very resistant to corrosion.

Electrolytic Refining

Copper. The art of electrorefining of copper was invented by James Elkington, in 1865, shortly after the discovery of the dynamo. The invention of the dynamo made possible the electrolytic refining of copper, and the pure metal in turn opened up the great electrical industry of today, which is the largest customer of the copper plants. In electrolytic refining of metals the starting material is a highly concentrated alloy. The object of refining is to remove the last impurities, to recover not only the principal metal in very pure form but also the foreign constituents, especially the precious metals. The impure metal is made the anode at which it dissolves in the electrolyte, to be deposited at the cathode in pure form, inasmuch as the impurities either do not dissolve and fall to the bottom of the tank as slime or mud, or, if they are dissolved, are precipitated by constituents of the electrolyte, or else can remain in the electrolyte without being deposited on the cathode. In general, in the case of copper refining, the anode material is comparatively pure, running 98 per cent copper or better and in most cases well above 99. During the smelting of the copper and lead ores, the base metals act as carriers for the precious metals so that the latter are found in the crude copper or lead. About 75 per cent of the total silver production of the world is recovered as a by-product of copper, lead, zinc, and nickel refining. In addition a very large portion of the platinum produced is a by-product of nickel refining, while considerable palladium is obtained from the same source as well as from copper slimes. Selenium and tellurium are produced almost entirely as by-products of copper refining.

The crude copper from the smelter, termed blister, comes to the refinery in the form of slabs which are sampled very carefully for their precious metal content and then cast into anodes for the refining cells. Cathodes are so-called starting sheets or thin sections of copper deposited on blanks employed as cathodes in starting-sheet cells, from which the deposited copper is stripped off as a sheet. Connectors are then attached and the starting sheet becomes the cathode in the refining cell. The cathode copper is exceedingly pure, usually running considerably better than 99.96 per cent copper in its commercial remelted forms. In order to have high electrical conductivity, the copper must be free of impurities, particularly the metalloids such as arsenic and antimony, and in order to avoid brittleness it must be free from tellurium and lead. The electric conductivity is a very delicate measure of copper purity. It is given according to the Matthiessen standard in which the resistance of 1 mg. pure soft copper at 0°C. is 0.14172 international ohm.

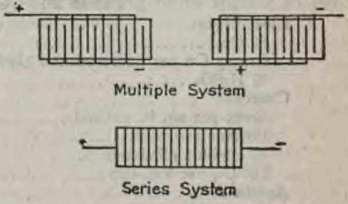


FIG. 27.—Arrangement of electrodes in copper refining.

The electrolyte universally used is a solution of $CuSO_4$ containing an appreciable amount of H_2SO_4, which is circulated through the tanks generally in cascade, *i.e.*, from one tank at a higher level to a succeeding one at a lower level. The use of addition agents such as glue and goulac to cause better cathode deposits, and reduce nodules and crystals, is general. Current densities in American refineries vary according to local conditions, power costs, and other items. Constant attention is directed toward reduction of voltage across the tanks to reduce energy cost. Every effort is made to eliminate contact resistances and unnecessary voltage drops through the tanks.

Soluble sulphates of impurities in the anode pass into the electrolyte, portions of which are purified at intervals. Nickel sulphate from such purification becomes a by-product of copper refining and finds extensive employment in nickel electroplating. In some plants copper builds up in the electrolyte to too great an extent, and portions of the electrolyte are run into depositing out tanks in which insoluble anodes are used, so that part of the copper of the electrolyte is precipitated.

Two different systems of electrode arrangement are in use. In the parallel, or multiple, system all cathodes are in parallel and all anodes are in parallel, the tanks being in series. In the series system, only the first and last electrodes are connected to the electric circuit, so that all of the electrodes in the circuit, with the exception of the first and the last, are bipolar, one side functioning as cathode and the other as anode. The arrangements of the two systems are illustrated diagrammatically in Fig. 27. The multiple system finds much wider application in that in the United States there are only one plant and part of another employing the series system.

In the multiple tank, close attention must be paid to the contacts. The series tank has relatively no contacts or conducting bars, and the electrodes are very close together, the anodes being thin, even plates. To produce such anodes they must either be rolled or specially hand cast, and the material used must be of good quality. The interest on the metal tied up in process and the investment in plant are less in the series system. The series system requires no starting sheets, but much closer supervision is needed to keep up the quality of the cathodes. As lead-lined tanks cannot be used in series work due to the relatively high voltages used, tank maintenance becomes an important item.

Average data for copper refining are shown in Table 20. For greater detail, see Mantell, "Industrial Electrochemistry," 2d ed, McGraw-Hill, 1940.

Table 20. Copper Refining

Electrolyte:	
% Cu	3-3.5
Grams Cu per liter	35–42
% H_2SO_4	15–18
Current:	
Amp. per sq. ft. cathode	15–20
Voltage per tank	0.2–0.4
% current efficiency	90–95
Lb. Cu per kw.-day	200–250
Anodes:	
Composition	99+ % Cu
Weight, lb	500–700
Cathodes:	
Original weight, lb	7–11
Finished weight, lb	145–300

Lead. The successful electrolytic lead-refining method, known as the Betts process, is used in the large refineries at Trail, B. C.; East Chicago, Ind.; Omaha, Neb.; and at an English plant in Newcastle-on-Tyne.

Anodes of lead bullion and cathodes of electrolytic lead in sheet form are connected in multiple. They are supported on copper bars across a tank containing lead fluosilicate ($PbSiF_6$) and free hydrofluosilicic acid (H_2SiF_6) as electrolyte. The anode mud clings to the uncorroded anode and contains practically all the impurities. It is collected from the anode and from the bottom of the tank and treated for the recovery of Sb, As, Bi, Cu, and Se. Any tellurium which may be present is discarded. The lead deposit on the cathode is melted together with the starting sheet and cast into bars.

During electrolysis, Pb, Sn, Zn, Fe, Ni, and Co go into solution while Cu, Sb, As, Bi, Cd, Ag, Au, Se, and Te remain at the anode. In the first group, tin will invariably deposit with the lead, since these two metals are so close together in the electrolytic series. It is therefore necessary to subject the lead bullion to a softening treatment before it is cast into anodes. The other soluble metals will not precipitate with the lead. Of the insoluble metals, small amounts of antimony may pass to the cathode, especially when a current density as high as 17 to 18 amp. per sq. ft. and a temperature of 37° to 38°C. are employed. This may be removed by poling the cathode lead with air in a kettle, which at the same time will separate any tin that may be present.

The electrolyte contains 7 to 10 per cent lead and 8 to 12 per cent total H_2SiF_6, the free H_2SiF_6 content being 3 to 5 per cent. Multiplying the lead value by 0.7 will give an approximate figure for the lead combined with H_2SiF_6. To ensure a solid cathode deposit, a hot strong solution of glue is added daily, to the amount of about 0.013 per cent of the weight of the electrolyte. In making the electrolyte at the plant, H_2SiF_6 is formed by the action of HF on SiO_2, then allowed to combine with lead, PbO, or white lead to yield $PbSiF_6$.

Tanks are arranged in double cascades. If impure anodes are used, the rate of circulation of the electrolyte is 3 to 4 gal. per min., while with pure anodes it becomes 7 gal. per min. The solution is raised from the sump by means of copper centrifugal pumps equipped with bronze shafts. An electrolyte loss of 5 to 10 lb. H_2SiF_6 per ton of normal lead bullion is attributed in part to the dissociation of the acid. If the anodes contain appreciable quantities of impurities, the loss will approach the higher figure.

Ordinarily, impurities in lead bullion do not total more than 2 per cent, 1 to 1.25 per cent being antimony. Anodes containing more than 2.25 per cent of foreign metals cause difficulties in electrolysis.

Table 21. Electrolytic Lead

Electrolyte:	
% Pb	7–10
% Total H_2SiF_6	12–15
Temperature, °C	35–40
Current:	
Current density, amp. per sq. ft	16–18
Voltage per tank	0.4–0.6
Anodes:	
% Pb	98+
Weight, lb	375–400
Cathodes:	
Original weight, lb	9–12
Finished weight, lb	130–150

Data on refining practice are given in Table 21. Cathodes are cast on sloping cast-iron tables fed by a trough at the upper end. The electrode spacing is 1¾ to 2½ in., the usual range being 2⅛ to 2¼ in. center to center and 1⅝ in. face to face. Rectangular, flat-bottomed tanks, similar to those used in copper refining, are employed in the electrolytic lead-refining plants. These tanks are coated with petroleum-residue asphalt not exceeding ¼ in. in thickness.

To ensure a full surface for slime particles, about 20 to 28 per cent of the anode is left unattacked, returning to the anode kettles as scrap. The anode mud which may drop to the bottom of the tank is likely not only to cause short circuits but to set up a chemical reaction between its components and the free acid of the electrolyte. A 10-day cleanup and renewal of anodes are the usual plant practice. The presence of bismuth, tellurium, and selenium is the determining factor in the treatment of the mud. If these metals be absent, the process is a simple one; but with bismuth to be recovered, the treatment becomes complicated.

Nickel. The largest share of the world nickel production is obtained from the Sudbury district of Ontario, Can. The ores are copper-nickel sulphides containing cobalt, iron, and precious metals. The ores are smelted to low-grade mattes which are then blown in basic converters for the removal of iron. The product of these converters, termed *bessemer matte*, is shipped to the nickel refineries where the process is a combination thermal and electrolytic one.

The process is based on the fact that in a molten system containing nickel sulphide, copper sulphide, and sodium sulphide, in general two liquid layers are formed, the upper carrying the bulk of the sodium and copper sulphides and the lower the bulk of the nickel sulphide. A separation is made of these two layers.

The bessemer matte, containing approximately 54 per cent Ni, 26 per cent Cu, 20 per cent S, and 0.30 per cent Fe, is smelted in a water-jacketed blast furnace with coke and soda flux. On solidification in pots a **top** and a **bottom** are formed. About 90 per cent of the total copper in the form of top is transferred to converters, where it is blown to blister copper ready for market. The separation between top and bottom is marked by a clean line of cleavage.

The bottom, containing the nickel, is again smelted with soda flux, poured, and allowed to solidify into two layers. The second bottom is nickel sulphide (Ni, 70 per cent; Cu, 0.90 per cent; Fe, 0.25 per cent), which is ground, washed free of soda, leached with dilute H_2SO_4 to remove iron, given a chloridizing roast, and leached to remove the copper; then follows a second roast with soda ash to remove all fractional remaining impurities. The resulting black oxide of nickel of the following analysis, Ni, 77.60 per cent; Cu, 0.10 per cent; Fe, 0.25 per cent; Si, 0.10 per cent; and S, 0.015 per cent, is ready for the market or reduction to metal in the open-hearth furnace. It is tapped from the furnace into pig nickel, nickel shot, and nickel anodes. The nickel anodes go to the electrolytic refinery where they are refined under the conditions tabulated.

The electrolyte, purified of iron and other metals, enters the catholyte box at an acidity equivalent to pH of 5.2. During refining, the electrolyte decreases in nickel content leaving the tank at a pH of 4.0. The electrolyte is purified of iron and precipitable metals by passage over finely divided nickel, using the cementation method. Further removal of iron is obtained by adjusting the pH of the solution to 5.2 and heating it to approximately 135°F., at which point ferric sulphates are precipitated. The electrolyte is then cooled down to 125°F. and returned to the cells through rubber-lined or hard-rubber piping. Every precaution is taken to keep the electrolyte out of contact with metals, the liquid coming in contact only with hard rubber, mastic, and, in a single case, with lead at the discharge-box end of the cell, where there is a lead apron.

Owing to the use of the closely woven special canvas diaphragms and the resultant voltage drop across them, the energy efficiency will be low in comparison with copper refining. The anodes are just slightly larger than the cathodes. A 3-day plating gives a 12-lb. starting sheet. Top buckle strips are made from 1½-day starting sheets being fastened together in the same manner as is common practice in copper plants.

Chlorides are not added for anode corrosion, but they are present from various sources to the extent of 75 mg. per l. The sulphur content of the electrolytically refined nickel is negligible.

The precious metal content of the original anodes is ¾ oz. per ton, being mostly platinum, palladium, and some rhodium. Slimes from primary anodes will run 12 oz. per ton. These slimes contain a large amount of nickel oxide, the nickel content being as high as 30 to 40 per cent. The slime goes back for reworking and remelting into so-called secondary anodes which have an original precious-metal concentration of 30 oz. per ton, yielding slimes running 500 oz. per ton. These slimes are concentrated by further operations to a product containing 5000 oz. per ton for shipment.

Table 22. Data on Electrolytic Refining of Nickel

Electrolyte:	
% Ni	40 g. per l.
% Boric acid	20 g. per l.
Temperature, °C	52–57
Circulation apparatus	hard-rubber pumps 10 rotaries, each 1100 kva.
Current:	
Amp. per sq. ft. cathode	11.5–12
Voltage per tank	2.4–2.5
% current efficiency	93–94
Kw-hr. per lb. Ni	1.1
Anodes:	
Composition	precious metals ¾ oz. per ton
Length, width, thickness, in	36 × 27 × 2
Weight, lb	415–425
Mode of suspension	cast lugs
Life, days	32–33
% Scrap	36–40
Cathodes:	
Starting sheet blanks	aluminum, Na_2S dipped, 30 lb. each
Size starting sheet, length by width, in	36 × 28
Weight, lb	11–12
Mode of suspension	nickel loops
Replaced after how many days	14
Weight, lb	125
Deposition vats, length by width by depth	16 ft. 9½ in. × 2 ft. 10½ in. × 5 ft. 2 in.
Number of anodes, cathodes	29, 30
Tank material	concrete, mastic, or gilsonite lined
Anode mud:	
Composition	
From primary anodes	precious metals 12 oz. per ton, Ni 30–40 %
From secondary anodes	precious metals 500 oz. per ton

Electrolytic nickel-dissolving cells are used for restoring the metal content of the electrolyte.

Cathode nickel is not remelted, but the cathodes are cut into squares either 2, 4, 6, 8, or 9 in. as specified by the customer. These are packed in barrels for shipment.

Tin. During the World War the electrolytic refining of tin was developed in the United States as a means of dealing with the metal obtained from the smelting of complex or impure Bolivian ores from which straight dry thermal methods of refining could produce only a poor grade of metal.

The bath employed commercially was 8 per cent H_2SO_4, 4 per cent cresol-phenol sulphonic acid, and 3 per cent Sn. Practically, lead is the only impurity that dissolves; consequently the electrolyte must contain a radical that will form an insoluble compound with lead, such as a sulphate, chromate, fluoride, etc. The other metals occurring as impurities in tin (As, Sb, Bi, Cu, etc.) are not dissolved and remain in the anode slimes.

The operating data are given in Table 23. The total acid in the electrolyte is calculated as H_2SO_4, although it was partly H_2SO_4 and partly cresol sulphonic acid. Glue-cresylic acid emulsions were used as addition agents. The addition agent consumed was of the order of ⅓ to 3 lb. glue and 8 to 16 lb. cresylic acid per ton of refined tin. Refinery starting sheets were made by pouring molten electrolytic tin over an inclined steel table of the size and shape of sheet desired, a method similar to that used for making starting sheets for the electrolytic refining of lead. The electrolytic tin analyzed better than 99.98 per cent Sn.

Table 23. General Data on Electrolytic Tin (*Mantell*)

Electrolyte:	
% Sn	3
% total acid (as H_2SO_4)	10.2
Temperature, °C	35
Circulation, gal. per min	5
Circulation apparatus	vertical centrifugal pumps
Current:	
Amp. per sq. ft. cathode	8–10
Voltage per tank	0.3–0.35
Current, kw. per generator	4500
% current efficiency	85
Kw.-hr. per lb. Sn	0.085
Anodes:	
% composition	Sn, 96.0; Bi, 1.0; Sb, 0.25; As, 0.15; Cu, 0.25; Pb, 1.0
Length, width, thickness, in	33 × 36 × 1¼
Weight, lb	350
Mode of suspension	cast lugs
Life, days	21
% scrap	25
Cathode:	
Size starting sheet, length by width by thickness, in	34 × 37 × 0.03
Weight, lb	8 to 10
Mode of suspension	wrapped
Replaced after how many days	7
Weight, lb	100
Deposition Tanks:	
Length, width, depth	12 ft. 11 in. × 3 ft. 5 in. × 3 ft. 6 in.
Number anodes, cathodes	26, 27
Electric connection	Walker
Material of construction	wood, lead lined
Anode mud:	
% anode	5
% composition	Pb, 20; Cu, 5; As, 3; Sb, 5; Sn, 30; Bi, 20

Silver. The raw material from which pure silver is produced is in the main concentrates from anode slimes from copper refining as well as those of other nonferrous metals such as lead, nickel, and zinc; silver concentrates resulting from the desilverization of lead by the Parkes process; and silver-gold bullion of various compositions including secondary metal and scrap. From copper-refining slimes the crude silver may contain 95 per cent Ag, 1 to 3 per cent Au, the remaining 2 per cent being Cu, Bi, Pb, Te, Fe, Ni, Pt, and small amounts of other metals; while from the desilverization of lead the crude silver may contain 98 per cent Ag; 0.5 per cent Au; 1.5 per cent being Cu, Bi, Pb, and Zn. Such material is cast into anodes and refined electrolytically in a $AgNO_3$-$Cu(NO_3)_2$ electrolyte.

In the electrolytic refining of copper, all the precious metals originally in the anodes drop to the bottom of the tanks during the process of electrolysis and are known as *slime* or *anode mud.* When the scrap anodes are lifted from the tanks after electrolysis, the slime is sluiced from the bottom of the tank and pumped to the silver refinery. On reaching the silver refinery, the slime is allowed to settle in large tanks, filtered, and given a light roast (300°C.). This converts all the copper into copper oxide, readily leachable with 10 per cent H_2SO_4. The leached slime, containing less than 1 per cent copper, is melted and refined in small oil-fired reverberatory *doré furnaces.* During the refining process, the lead forms a lead slag which is skimmed off and sent to the lead refinery for further treatment. Antimony is volatilized and recovered from the flue dust. Selenium and tellurium are partly volatilized but are mainly removed by the addition of alkaline fluxes. The doré silver is cast into anodes and electrolyzed for the parting of the silver from the gold, using both the Thum and the Moebius systems (Fig. 28*a* and Fig. 28*b*).

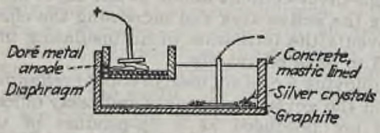


FIG. 28*a*.—Thum cell.

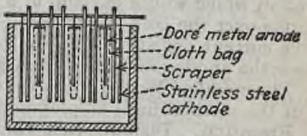


FIG. 28*b*.—Moebius cell.

In the Thum system the electrolytic cells consist of shallow, glazed, porcelain tanks, or mastic-lined concrete tanks, the bottom of which, lined with graphite slabs, forms the cathode. The doré silver anodes are arranged horizontally in wooden frames above the cathode. These frames are covered with canvas which acts as a diaphragm and prevents the gold slime from mixing with the cathode deposit. The silver is deposited in a loose crystalline form which is scraped from the bottom of the tank at regular intervals. In the Moebius system, the anodes and cathodes are arranged vertically in much the same manner as in copper refining, the anodes being enclosed in canvas bags in order to catch the gold slime. The cathodes are sheets of stainless steel. Mechanically operated wooden scrapers are used to remove the silver crystals from the cathodes into trays which are periodically withdrawn and emptied. The cathode deposit of silver crystals is very pure. After washing, it is melted in large graphite retorts into standard 1,000-oz. bars, assaying over 999 in fineness.

The gold slime resulting from the silver electrolysis contains all the gold, platinum, and palladium which were originally present in the blister copper. It is treated with H_2SO_4 to remove the excess of silver present and is then melted in graphite crucibles and cast into gold anodes to be electrolyzed using the Wohlwill process.

The silver electrolyte is silver and copper nitrates (60 g. Ag per liter, 40 g. Cu per liter) operated at a current density of 40 to 50 amp. per sq. ft.

Gold. The process employed for gold refining is due to Wohlwill. The method, which is used for the refining of gold in all of the United States mints, consists in electrolyzing gold anodes in a hot acid solution of 7 to 8 per cent gold chloride and 10 per cent HCl. In order to maintain low interest charges on the gold tie-up, a high current density of 110 amp. per sq. ft. of cathode surface is used. This necessitates a thorough stirring of the electrolyte, which is accomplished by means of a small air lift.

In gold electrolysis, the platinum and palladium are totally soluble and are allowed to accumulate in the electrolyte until present in sufficient quantity to recover. The amount of platinum and palladium recovered is small, averaging only about 1 oz. of the combined metals per 1,000,000 lb. copper refined.

The electrolysis cells are constructed of glazed porcelain and chemical stoneware. They are relatively small in size, in that the electrolyte is an expensive one. The cathodes, having the same area as the anodes, are of thin gold foil or thin rolled-gold sheet. In copper refineries the anodes are fairly thin. In governmental refineries or mints where interest on metal tied up is not a factor, the anodes may be fairly thick. At periodic intervals, due to the accumulation of impurities, part of the electrolyte is drawn off for purification and replaced by a strong $AuCl_3$ solution.

For the treatment of anodes of high silver content, Wohlwill modified his process by using an unsymmetrical a.c. in which an a.c. of low frequency and greater current density is superimposed on the d.c. The quantities of gold dissolved and deposited are functions only of the d.c. The a.c. opposes passivation which would develop in a high-silver-content anode as the AgCl layer forms over the gold anode, lessening the active area and increasing the effective current density. The a.c. prevents the formation of an insulating film over the anode and causes the AgCl to flake off readily.

The modified Wohlwill process employing a.c. is not used in copper refineries, but the method finds employment in the government mints and assay offices.

Bismuth. Bismuth-rich slimes are produced as a by-product of the Betts method of lead refining. The slimes are washed, dried, and smelted under oxidizing conditions with Na_2CO_3 and NaOH to produce alkaline slags which carry off the arsenic and antimony as sodium compounds and the copper as oxide. The residue is crude bismuth containing mainly silver and lead. This is refined electrolytically in an acid $BiCl_3$ solution containing somewhat more than 100 g. HCl per liter and 3 to 4 g. of bismuth in a Thum cell, the same type as is used for silver refining, at 50° to 60°C. The lead from the crude bismuth anode goes into solution as $PbCl_2$ but does not deposit with the bismuth. It concentrates in the electrolyte, portions of which are periodically removed and the $PbCl_2$ crystallized out.

Electrowinning

Copper. In several places in the world, copper-containing minerals are treated for the recovery of the copper by hydrometallurgical methods, in which the mineral is leached or dissolved by a solution which in turn becomes an electrolyte and is stripped of a portion of its metal values by electrolytic precipitation, the stripped or spent electrolyte being returned for leaching fresh ore in a cyclic process. The metallic values of ores treated by hydrometallurgical or electrowinning methods are usually very low, being of the order of 1.5 to 1.75 per cent copper (or less). At the plant of the Chile Copper Co. at Chuquicamata, Chile, the principal copper minerals are chalcanthite ($CuSO_4.5H_2O$), brochantite [$CuSO_4.3Cu(OH)_2$], and atacamite [$CuCl_2.3Cu(OH)_2$], and the leach contains H_2SO_4. In the electrolytic deposition insoluble copper silicide anodes are used. Power consumption per pound of copper is much higher than in copper refining because of the greater voltages across the cells.

In the plant of the Inspiration Consolidated Copper Co. at Inspiration, Ariz., the ore consisted of mixed oxides and sulphides in which the total copper of the ore was of the order of 1.1 to 1.2 per cent. The leach was a H_2SO_4-$Fe_2(SO_4)_3$ solution, and the insoluble anodes were 8 per cent antimonial lead.

In general, the ores were leached in large concrete tanks. At Chile the tanks are of reinforced concrete lined with mastic, while at Inspiration they were reinforced concrete lined with lead. In general, despite the fact that hydrometallurgical plants are operating with extremely low grade ores, the copper recovery is of the order of 80 to 90 per cent and the production costs are considerably lower than in pyrometallurgical plants operating smelters. These smelters, in addition, have the advantage of using high-copper-content ores.

Operating data for different plants vary widely depending upon ores treated, leaching solutions used, impurities encountered, and methods of purification. For further data see Mantell, "Industrial Electrochemistry," 2d ed., McGraw-Hill, New York, 1940. Cell electrolytes are 20 to 35 g. Cu per liter, 20 to 45 g. H_2SO_4 per liter. Current densities average 10 to 12, although as low as 5 amp. per sq. ft. are used. Energy consumptions are 1 to 1.5 kw.-hr. per lb. Cu; tank voltages are 1.8 to 2.5, and current efficiencies 65 to 90 per cent.

Zinc. Vast quantities of complex zinc ores are available which are not readily amenable to pyrometallurgical methods, but in the treatment of which leaching processes, coupled with electrolytic deposition, have particular advantages. In addition, the subsidiary metals can be completely recovered.

Electrolytic zinc plants are now operating on a very large scale in a number of places in the world. In general, all the processes employed involve H_2SO_4 and the production of $ZnSO_4$ electrolytes, which can be divided into those termed low-acid, low-current density, and the high-acid, high-current density, also known as the Tainton process.

The ores treated are of the complex zinc-lead-copper-iron sulphide type. These ores, averaged after roasting, contain 33 per cent or higher Zn, 4 to 4.5 per cent S, and a soluble zinc content of the order of 82 to 86 per cent, 10 per cent Pb, 21 to 22.5 per cent Fe, and 1.7 to 2 per cent Cu, with about 7 to 7.5 per cent of the ore being insoluble in acids. In the roasting of concentrate containing 5 per cent iron (or less), practically all the iron combines with zinc as $ZnO.Fe_2O_3$ (ferrite), which is nearly insoluble in dilute H_2SO_4. As the iron content increases above 5 per cent, the combination of zinc and iron oxides can be partly controlled by temperature regulation during roasting. The roasting furnaces are operated to convert the zinc sulphide in the ore to acid-soluble oxides and sulphates and, at the same time, to produce SO_2 which is used for the manufacture of H_2SO_4 employed in leaching.

In general, the procedure is cyclic. Electrolyte from the cells after the greater portion of the zinc content has been deposited is used to leach roasted ore, termed calcine, the solids and liquids being separated, the $ZnSO_4$ solution being purified by means of metallic zinc. Spent electrolyte from deposition tanks contains 10 to 11.5 per cent H_2SO_4 and about 2.5 per cent zinc. After the leaching treatment, the liquor contains about 0.5 per cent H_2SO_4 and 10 per cent zinc.

The high-acid, high-current-density method is employed in the electrolytic zinc plants erected to treat the zinc-bearing ores of the Coeur d'Alene district. These ores cannot be classed as particularly favorable for electrolytic treatment, inasmuch as they tend to form an unusually large amount of insoluble zinc ferrite in the roast and yield appreciable quantities of gelatinous silica in the leaching operation. Furthermore, they contain relatively large amounts of cobalt, which is one of the most troublesome impurities from the standpoint of electrolytic zinc treatment. In the Tainton high-acid process the return electrolyte used for leaching carries 28 to 30 per cent free acid, and the electrolysis is carried out at a current density of 100 amp. per sq. ft., both of these amounts being about three times as great as the corresponding figures in ordinary electrolytic zinc practice.

Table 24. Electrolytic Zinc

	Low acid, low c.d.	High acid, high c.d.
Electrolyte:		
Zn to cells, g. per l.	100–150	200–220
H_2SO_4, cell discharge, g. per l.	100–130	280
Current:		
Density, amp. per sq. ft.	30–40	100
Current efficiency, %	90–94	87–92
Voltage per tank	3.7	3.2–3.5
Anodes:		
Material	Lead	Lead alloy
Construction	Solid	40–50% perforated
Cathodes:		
Sheets	Aluminum	Aluminum
Zn weight on cathode, lb.	8–11	22–24
Tanks	Wood, lead lined; or lined concrete	Wood, lead lined
Tank mud	MnO_2	MnO_2

Table 25. Electrolytic Recovery of Cadmium

Raw material	Precipitate from Zn dust purification of Zn electrolyte
Electrolyte:	
Cd to cells, g. per l.	100–120
Zn to cells, g. per l.	80
Concentration H_2SO_4, cell discharge, g. per l.	70–80
Temperature, °C	35
Current per cell:	
Amp. per sq. ft.	4.25
Current efficiency, %	80–90
Kw.-hr. per lb.	0.82
Voltage per tank	2.6
Anodes:	
Composition	Lead
Anode-cathode spacing, in.	3½
Cathodes:	
Composition	Aluminum
Removed	24 hr.
Weight, lb.	100
Tanks:	
Number anodes and cathodes	27,26
Material of construction	Wood, lead lined

In the Tainton process, zinc ferrite is separated magnetically from the calcine and in the leaching process is first added to a strongly acid solution resulting from stripping the electrolyte in the deposition cells, the neutralization of the acid being then completed by the use of the ZnO portion of the calcine free from ferrite. With the allowable production of zinc ferrite in the calcine, less rigid control on roasting is necessary than is needed for the usual type of leaching of zinc ores.

Cadmium Recovery. Cadmium is found in very small quantities in practically all zinc ores. It can be profitably produced only as a by-product in the manufacture of some other metal. Important sources of raw material are bag-house condensation products from lead and copper furnaces, zinc ores, "blue" powder, and particularly cadmium residues from electrolytic zinc plants. These materials are leached, purified by chemical treatment and by the precipitation of heavy metals with zinc, to produce liquors containing only zinc, cadmium, and free H_2SO_4. These liquors are electrolyzed between insoluble lead anodes and aluminum cathodes. In earlier plants, rotating cathodes were used, but in present practice they are stationary. Operating data are given in Table 25. Cadmium has found widespread application in electroplating and rust-proofing. Demand for the metal has at times exceeded the supply and markedly increased the price.

Manganese is leached from reduced manganese ores by sulphate solutions which are electrolyzed in multicompartment cells to produce pure metal.

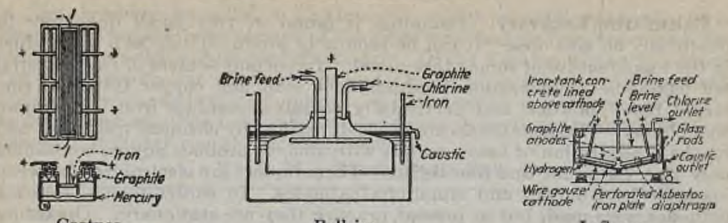
Alkaline Chloride Electrolysis

Aqueous Electrolytes—Chlorine and Caustic. Chlorine and NaOH are made by the electrolysis of a solution of an alkaline chloride, generally NaCl. The sodium ion travels to the cathode. The cathodic products are a solution of NaOH and hydrogen gas evolved at the cathode. The chlorine ion travels to the anode, gives up its charge, becomes a chlorine molecule, Cl_2, and to the greater extent is evolved as gaseous chlorine, although some dissolves in the solution. The anodes almost universally in use are those nonattackable by chlorine, consisting of graphite. In every commercial cell, precautions are taken to keep the anodic and cathodic products separate from each other so that they do not interact to form hypochlorites. The products of electrolysis are removed as rapidly as possible from the sphere of electrolytic action. Cells in commercial use fall, with only minor exceptions, into those in which the hydroxyl ions are not liberated inside the electrolysis chamber (as in the mercury cells) and diaphragm cells in which mixing of the anolyte and catholyte is prevented by a porous diaphragm. These may be subdivided into those rectangular in shape and those circular or cylindrical in form. The bell-jar cell is an example of the type which does not use a diaphragm, but in which mixing of the anodic and cathodic products is prevented by regulation of the electrolyte fed to the cell.

The fundamental principles of construction of the more important cells used commercially are shown in Fig. 29.

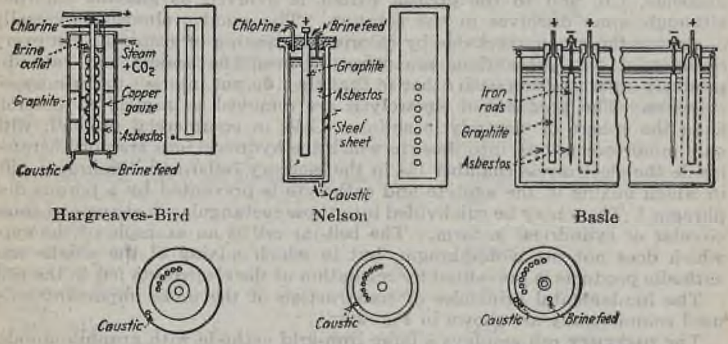
The **mercury** cell employs a large iron-grid cathode with graphite anodes and an intermediate electrode consisting of mercury. In one portion or chamber of the cell a solution of NaCl is electrolyzed to produce chlorine at the anode and sodium metal at the mercury cathode. The sodium metal alloys with the mercury to form an amalgam. Circulation of the mercury by mechanical means, such as pumps, screws, or rocking, causes the mercury from the NaCl compartment to pass into another compartment where it functions as an anode in a NaOH electrolyte, with the production of hydrogen at the iron cathode. The second section is often termed the denuder. The sodium from the amalgam is discharged with the formation of caustic solution. A number of modifications of a mechanical nature for the circulation of the mercury have been proposed. In general, mercury cells produce caustic liquors of much higher concentration than those of other types.

In the **Castner** cell the mercury is circulated from two outside electrolysis compartments to a central denuder section. In the **Sorensen** cell the

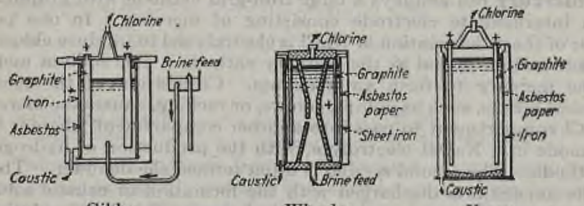


Castner Bell-jar LeSueur

Hooker Townsend Allen-Moore



Hargreaves-Bird Nelson Basle



Gibbs Wheeler Vorce

FIG. 29.—Typical cells for Cl_2 and NaOH.

mercury is mechanically circulated by means of a cup wheel which returns the metal from the lower level denuder to the higher level of the electrolysis chamber, from which the mercury circulates back to the denuder. In the **Krebs** cell, built in capacities from 4,000 up to 15,000 amp. and shown in Fig. 30, the mercury circulation is by means of an Archimedean screw or pump from the denuder chamber to the electrolysis section, from which higher level the mercury flows back to the denuder section. Typical operating data are given in Table 25.

In the **bell-jar** cell, used to some extent abroad but not at all in the United States, a bell jar consisting of a nonconducting and nonporous material is inverted in a vessel containing the NaCl electrolyte, while outside the bell there is a ring-shaped iron cathode. The graphite anode is inside the bell. Brine feed is through the top of the bell, past the anode. The bell also contains a chlorine outlet. As cathode alkali is formed, because of its specific gravity and mixing due to hydrogen evolution, it is distributed throughout the whole of the containing vessel. Because of hydroxyl-ion migration and diffusion, it tends to ascend the bell jar but is prevented by the downward flow of brine solution coming from the anode, so that a sharp alkali boundary and neutral layer are formed at a level between the bottom of the anode and the bottom of the bell jar. Commercially, the cells are small in size and cannot be depended upon to operate in a satisfactory manner, difficulty being experienced in the maintenance of the alkali-salt-solution boundary.

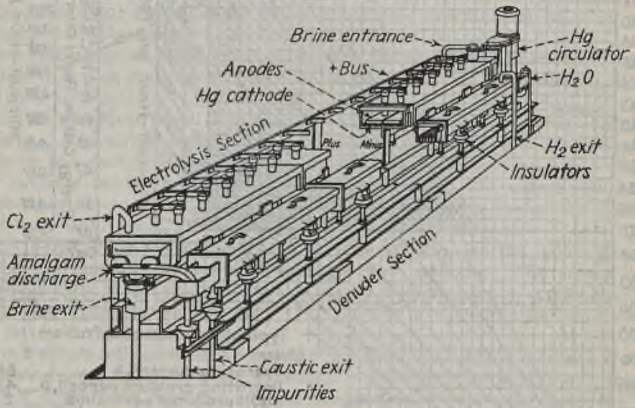
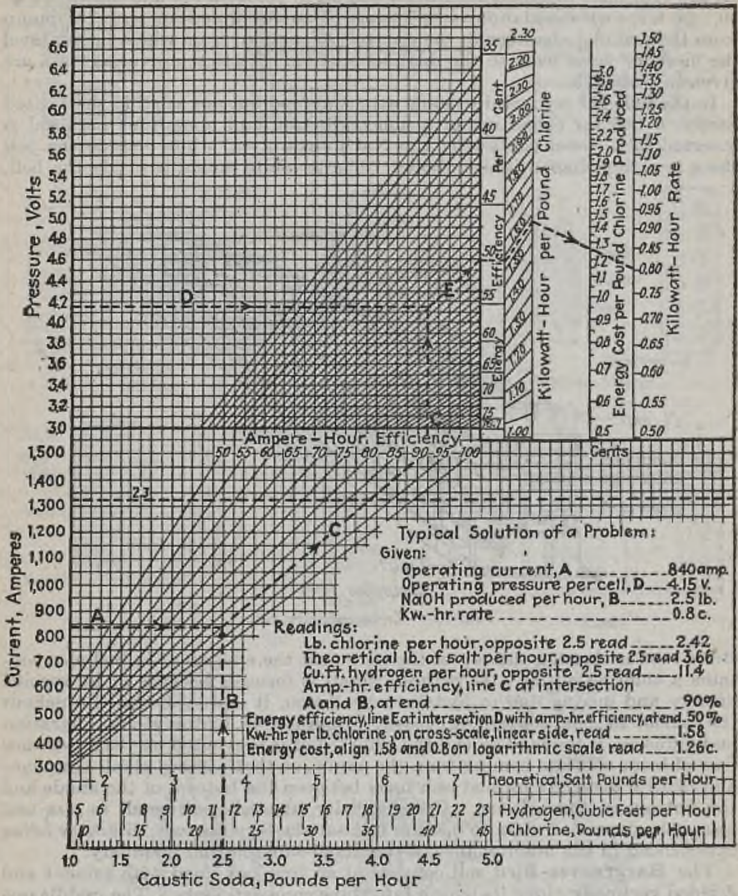


FIG. 30.—Krebs mercury cell.

The **Hargreaves-Bird** cell consists of an iron box lined with cement and divided vertically along its length into three compartments. The middle one contains the graphite anodes, brine feed entering the cell at the bottom of the middle compartment and overflowing at the top, a portion of it diffusing through the diaphragm and being electrolyzed. The separating partitions are asbestos or asbestos cement serving as diaphragms, while the cathodes are alongside of the diaphragms and consist of copper gauze. The electrolyte percolates through the diaphragm to the cathode and into the cathode compartment. Steam is introduced into the cathode compartment and tends to keep the diaphragms open. At the same time CO_2 is also introduced, uniting with the cathode product to form Na_2CO_3 so that the solution drawn off at the bottom of the cathode chambers is a mixture of NaCl and Na_2CO_3, which are separated by crystallization. The commercial cells show extremely long life. The cell is in use only in the paper industry. It has the disadvantage of producing Na_2CO_3 instead of the more valuable NaOH.



Typical Solution of a Problem:

Given:

Operating current, A .... 840 amp.
Operating pressure per cell, D ... 4.15 V.
NaOH produced per hour, B ...... 2.5 lb.
Kw.-hr. rate .... 0.8 c.

Readings:

Lb. chlorine per hour, opposite 2.5 read 2.42
Theoretical lb. of salt per hour, opposite 2.5 read 3.66
Cu. ft. hydrogen per hour, opposite 2.5 read 11.4
Amp.-hr. efficiency, line C at intersection
A and B, at end 90%
Energy efficiency, line E at intersection D with amp.-hr. efficiency, at end . 50%
Kw.-hr. per lb. chlorine, on cross-scale, linear side, read 1.58
Energy cost, align 1.58 and 0.8 on logarithmic scale read ..... 1.26 c.

FIG. 31.—Calculation chart for raw materials, products, and power in electrolytic chlorine cells. [*Horine, Chem. & Met. Eng.*, **21**, 71 (1919).]

The **Townsend** cell consists of a center compartment containing the graphite anode, the electrolyte, a brine feed and a chlorine outlet, and two side compartments containing kerosene separated from the center compartment by an asbestos diaphragm alongside of which are iron-grid cathodes. The side compartments have hydrogen exits and adjustable swan necks for drawing off the caustic liquors. Saturated salt electrolyte is circulated through the anode compartment and part of it diffuses through the diaphragm and is electrolyzed. The products of electrolysis in the form of drops of electrolyte are carried away from the cathode by the hydrogen gas, to become mixed with the kerosene oil through which they drop to the bottom of the compartment. The kerosene oil serves to equalize the hydrostatic pressure on both sides of the diaphragm. Commercial cells operate at voltages of 4 to 4.2 and have capacities of 4,000 amp. per cell. Current efficiencies are of the order of 90 to 95 per cent. The materials of construction for the cell are cement and steel. In general, the shape of the cell is rectangular.

The **Giordani Pomilio** cell is similar in construction, employing vertical submerged diaphragms, but kerosene is not used.

The **LeSueur** was the first commercial porous, submerged diaphragm cell employed in the manufacture of Cl_2 for paper work. It is shown diagrammatically in Fig. 29.

The **Allen-Moore** cell resembles the Townsend cell in certain structural features. The anode compartment consists of a shallow U-shaped reinforced-concrete pot carrying the graphite anodes and forming the anode chamber. The cathode compartments are bolted externally, the separation between the compartments being made by asbestos diaphragms backed by perforated sheet-iron cathodes. The cell is rectangular in shape.

The **Buck-McRae** cell is similar in construction to the Allen-Moore cell but in addition has a central diaphragm and cathode. The entire cell fits in an iron tank and is enclosed at the top by a cement block or blocks through which the graphite anodes rise. Simplicity of construction is a feature. The cell is rectangular in shape.

The **Nelson** cell consists of a rectangular steel tank which carries a U-shaped perforated-steel cathode plate to which is fastened the asbestos diaphragm. Brine is introduced into the central compartment which contains the anode and the chlorine outlet. Brine percolates through the diaphragm to the cathode chamber in which an atmosphere of steam is maintained.

The **Hooker** cell is a high amperage unit, almost cubical in shape, using completely submerged diaphragms, being the commercial successor of the Townsend cell.

The **Basle** cell is an unusual type, finding employment in Europe. The feature of the cell is the mechanism used to circulate the electrolyte from anode to cathode. Both the graphite anodes and the iron cathodes are surrounded by asbestos-bag diaphragms. Hydrogen generated at the cathode inside the bag carries the cathode products up through the short iron tubes into gutters on the top of the cell, the gutters being covered by a collecting hood. Chlorine is taken off at the top of each anode through nonconductive collecting bells. A constant-level brine feed is used. The largest unit takes about 7,500 amp. and produces 11 to 13 per cent NaOH and 98 per cent chlorine, at a current efficiency of 90 to 92 per cent at a voltage of 3.3 to 4.5 at 40° to 50°C.

The cylindrical cells are represented by the designs due to **Gibbs, Wheeler,** and **Vorce.** A central chamber contains the graphite anodes, surrounded by an asbestos diaphragm alongside of a perforated iron-sheet cathode, the entire cell being enclosed in a steel cylinder. The chlorine outlet is at the top in all cases, and in the Gibbs and Wheeler brine feed is at the bottom, in the Vorce at

Table 26. Electrolytic Chlorine and Caustic

	Kind of cell						
	Sorensen small mercury	Krebs, mercury type	Hargreaves-Bird carbonate cell	Rectangular free diaphragm cell	Square cell, Hooker Type "S"	Vertical rectangular submerged diaphragm cell	Cylindrical cell
Voltage across cell	4.3	3.85–4.0	4.2	3.6–3.8	3.35	3.5–4.2	3.5–3.6
Current per cell, amp.	1,300	3,250–15,000	3,000	1,000–1,500	6,000	3,000–3,200	800–1,000
Current density, anode, amp. per sq. in.	0.78	1.03–1.68	0.03	0.3–0.42	0.382*	0.45–0.52	0.14† 0.34*
Current density, cathode, amp. per sq. in.	0.78	0.97–1.61	0.21	...	0.323*	0.32–0.39	0.35† 0.5*
Current efficiency of cell, %	90	94–96	90	92–97	95	90–96	94–96
Energy efficiency of cell, %	50	50–60	50	58–64	65	52–58	61–62
Lb. NaOH per kw.-hr.	0.69	0.68–0.75	0.69	0.76–0.83	0.93	0.74–0.77	0.86
Lb. Cl_2 per kw.-hr.	0.62	0.6–0.67	0.62	0.67–0.74	0.83	0.66	0.76
Anode material	Graphite	Graphite	Graphite	Graphite	Graphite	Graphite	Graphite
Cathode material	Mercury	Mercury	Perforated steel	1/16 in. perforated steel plate	Steel and wire screen	Perforated iron sheet	Perforated steel
Diaphragm material	None	None	Composition	Asbestos paper	Deposited asbestos	Asbestos cloth and paper	Asbestos paper
Cell proper material	Concrete-lined cast-iron frame	Iron, ebonite	Cast-iron casing, acid-proof brick lining	Steel and concrete	Concrete and steel	Cement	Steel and cement composition
Anode life, days	480	...	750	300–450	450	300–500	600–900
Average operating period of cell, days.	28	...	360	120	Diaphragm run 150–250 days	90–300	150–240
Raw materials	NaCl	NaCl, 300–315 g. per l.	Saturated brine	NaCl	Brine	Saturated brine	Saturated brine
Concentration of cathode alkali.	20%	350–650 g. per l.	170 g. Na_2CO_3 per l.	110 g. per l.	135 g. per l.	120–180 g. per l.	100–120 g./l.
Salt concentration, cathode alkali.	Trace	Trace	...	150–170 g. per l.	14–15%	110–150 g. per l.	140–170 g./l.
Size of anode	10 anodes, 28½ in. × 6 in. × 2 in.	...	16½ in. × 9 in. × 2 in., 72 anode plates	Active surface 3,600 sq. in.	90 anodes 1¼ × 6¼ × 18 in.	...	24 anodes, 2 in. × 2 in. × 36 in.
Size of cathode	Surface of mercury approximately the same as anode	...	10 ft. × 5 ft.	...	Active surface 129 sq. ft.	...	22 in. diam. × 34 in
External measurements of cell	5 ft. × 6 ft. × 1 ft.	...	11 ft. 6 in. × 7 ft. × 1 ft. 6 in.	...	4 ft. 5⅝ in. × 4 ft. 11½ in. × 3 ft. 7⅝ in. high. Pedestal and insulator 10½ in. additional	...	26 in. diam. × 42 in.

* Calculated from "active" anode or cathode surface. † Calculated from entire anode or cathode surface.

the top. In the Gibbs and the Vorce cells, the anodes depend from a cover and are supported on the external cylinder, while in the Wheeler cell the weight of the anodes is carried by a central double-cone or cylindrical cement pedestal. Caustic flow in all cases is from the external cathode compartment at the bottom of the cell. The usual construction materials are graphite for anodes; asbestos for the diaphragm; iron, or one of its alloys in some cases, for the cathode; cement for the cell base and cover; and steel for the containing cylinder. The three cells differ in other more or less minor structural features.

Commercial operating data on the various types of cells finding employment in plants in the United States are given in Table 26. For greater detail, see Mantell, "Industrial Electrochemistry," 2d ed., McGraw-Hill, 1940. Figure 31 is a calculation chart useful in connection with commercial chlorine cells. It is the work of Horine [*Chem. & Met. Eng.*, **21**, 69 (1919)]. Power costs are very important in chlorine production. Different cells operate under relatively widely different current efficiencies and voltages, so that from 0.62 to 0.76 lb. chlorine per kilowatt-hour are produced. Figure 32 shows the relation between energy efficiency as expressed in pounds of chlorine per kilowatt-hour, cost of power in cents per kilowatt-hour, and the total power cost in dollars per ton of chlorine produced by the cell or plant in question.

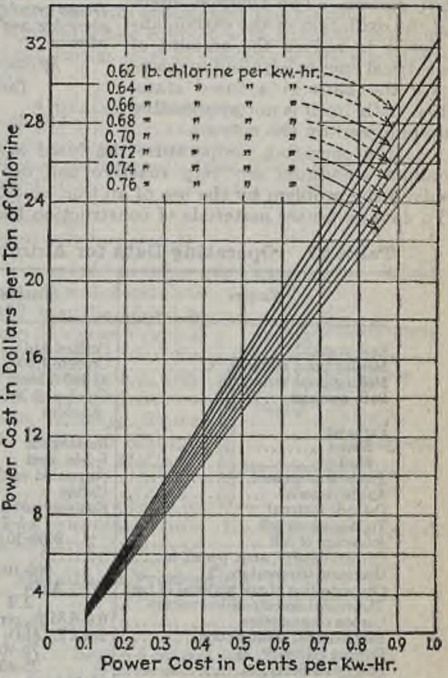


FIG. 32.—Power cost per ton of chlorine at varying power costs and chlorine yields.

Fused Electrolytes

Aluminum. The entire world production of aluminum is obtained by the electrolysis of a solution of alumina (Al_2O_3) in fused cryolite ($AlF_3.3NaF$). The process is based on the patents (1883–1889) of Hall in the United States and Heroult in France. Fundamentally, aluminum is produced by the electrolysis of alumina dissolved in a bath of aluminum fluoride and the fluoride of one or more metals more electropositive than aluminum, such as sodium, potassium, or calcium. Cryolite melts at about 1000°C. and at temperatures slightly above its melting point is able to dissolve as much as 10 to 20 per cent of its weight of alumina with resultant decrease in its melting point. As the result of electrolysis alumina is decomposed, aluminum metal being deposited at the cathode in a molten condition (melting point about 660°C.) and oxygen at the carbon anode with which the oxygen reacts to form the probable primary product of CO_2. This is believed to be subsequently reduced to CO by the hot carbon. The thermal effect of the oxidation of the carbon anodes is to reduce the amount of electrical energy required to maintain the bath in a fused state. The bath itself is not appreciably decomposed by the current.

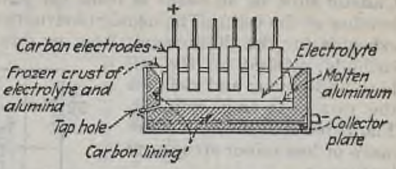


FIG. 33.—Electrolytic aluminum cell.

At the operating temperature the fused cryolite (or modified bath) and molten aluminum are very reactive and destructive of containers. Hall solved the problem by the use of an iron crucible or a box lined with carbon. To date no better materials of construction have been found.

Table 27. Operating Data for Aluminum and Magnesium

Factors	Aluminum	Magnesium
Raw material	Purified Al_2O_3	Anhydrous $MgCl_2$
Melting point material, °C	Cryolite 1,000	$MgCl_2$ 708
Melting point metal, °C	Al 660 (about)	Mg 651
Bath material	Cryolite + Na, Al, Ca fluorides	$MgCl_2$ + NaCl
Furnace:		
Shape	Rectangular	Rectangular
Shell	1–2-in. steel	Steel
Anode arrangement	Suspended vertically	Suspended vertically
Anode material	Carbon	Graphite
Cathode material	Rammed carbon lining	Steel pot
Voltage across cell	5.5–7	6–9
Amperage of cell	8000–30,000	
Current density, amp. per sq. in.	....	16–35
Operating temperature, °C	900–1000	670–730
Concentration of raw material in bath	2–5% Al_2O_3	
Theoretical decomposition voltage	2.0	
Carbon consumption	0.6–0.85 lb. per lb. Al	
Raw material consumption	2–2.2 lb. Al_2O_3 per lb. Al	4–5 lb. $MgCl_2$ per lb. Mg
Current efficiency, %	70–90	70–80
Energy efficiency, %	25–40	20–30
Energy consumption	10–12 kw.-hr. per lb. Al	8–13 kw.-hr. per lb. Mg
Specific gravity molten Al	2.29 (1000 °C.)	
Specific gravity molten cryolite	2.095 (1000 °C.)	

The reduction cell or "pot" is a strong steel box, usually rectangular in shape, provided with a carbon lining 6 to 10 in. or more in thickness. Figure 33 shows the essential features of such a cell. External and internal dimensions of the cell and lining vary considerably with current capacity. In commercial practice the smallest cells are 8,000 amp. and the largest of the order of 40,000 amp. Generally the more current that can be used in a cell, the lower will be the producing cost of 1 lb. aluminum. Aluminum production is approximately proportional to the current, while it takes just about as much labor to run a small cell as a large one. The limit of current capacity is set by increase in difficulties involved in changing anodes, breaking the frozen top crust, and operating the larger cells. For multiple-electrode pots the practical limit is reached at about 30,000 amp. Operating data are given in Table 27.

Aluminum Refining. Aluminum is refined electrolytically in an *inverted* cell in a fused electrolyte composed of cryolite, AlF_3, and BaF_2 nearly saturated with Al_2O_3 [Frary, Electrolytic Refining of Aluminum, *Trans. Am. Electrochem. Soc.*, **47**, 275 (1925)]. Heavier aluminum alloys at the bottom of the cell are in contact with the carbon anode, and the refined, purified metal rises through the electrolyte to make contact with graphite cathodes suspended vertically through a top frozen crust of electrolyte which contains a large concentration of alumina.

Table 28. Aluminum Refining

Bath	Cryolite, AlF_3, BaF_2 nearly saturated with Al_2O_3
Cathode	Purified molten metal, graphite
Anode	Impure molten metal, carbon (Cu-Al alloy)
Cell working temperature	900–1100°C.
Voltage	5–7
Current per cell	20,000 amp.
Purity of metal	99.80–99.90 per cent Al

Magnesium. Magnesium is prepared in large quantities by the chloride process, in which the raw material and electrolyte are $MgCl_2$ to which some NaCl may be added to lower the melting point. Figure 34 shows a typical cell. Operating data are given in Table 27. For manufacturing processes involving preparation, purification, and drying of the $MgCl_2$ and its electrolysis, see Gann, The Magnesium Industry, *Trans. Am. Inst. Chem. Engrs.*, **24**, 206 (1930); or Mantell, "Industrial Electrochemistry," 2d ed., McGraw-Hill, 1940.

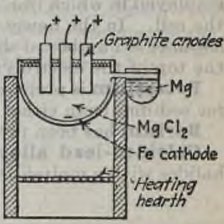


Fig. 34.—Magnesium cell.

Beryllium. Beryllium is produced from beryl by chemical concentration, producing chlorides or fluorides. Mixed fused salts are employed as electrolyte.

Table 29. Beryllium Production

Density Be metal	1.85
Melting point Be metal	1285°C.
Original raw material	Beryl, 4–5 per cent Be, 12–15 per cent BeO, 19–22 per cent Al_2O_3, 67 per cent SiO_2

Cell raw material	$BeCl_2$	BeF_2
Bath	$BeCl_2$-NaCl	BaF_2-BeF_2-NaF_2
Cathode	Iron pot	Iron
Anode	Graphite	Graphite
Bath temperature, °C	730–820	1300–1350
Cell current, amp	600	
Voltage	5–8	
Yield	10 parts $BeCl_2$ to 1 part Be	95%
Purity of metal	....	98%

Calcium. Calcium is made on a small scale by fused salt electrolysis according to the data shown in Table 30.

Cerium. Metallic cerium is produced from fused $CeCl_3$ plus alkali chlorides in cast-iron pot cells with graphite anodes, at 5.5 to 6.5 amp. per sq. in. at voltages of 10 to 15. The metal is used for pyrophoric alloys.

Lithium. Lithium is produced in small amounts by the electrolysis of fused halides of potassium and lithium which are produced from lithium minerals such as lepidolite, spodumene, and triphylite. The electrolytes used are either LiCl and KCl, or these with some LiBr. The metal is useful as an alloying agent for nonferrous metals such as copper. Its specific gravity is 0.534, melting point 186°C.

Table 30. Calcium Production

Raw material	Anhydrous $CaCl_2$
Melting point $CaCl_2$, °C	780
Melting point Ca, °C	800
Specific conductivity $CaCl_2$	1.9 at 800°C.
Type of cell	Contact electrode
Cell temperature, °C	780–800
Current density, amp. per sq. in	600–650
Energy consumption:	
Kw.-hr. per ton	50,000
Kw.-hr. per lb	22 to 24
Energy efficiency, %	10

Sodium. Sodium is produced on a large scale from the electrolysis of fused NaOH or NaCl. In the older method (NaOH) the Castner vertical cell is employed in which iron anodes and cathodes are used, with external heating of the cell. In the newer method (NaCl) graphite anodes are used and the cell assumes a number of different forms. In both cases sodium metal comes to the top of the electrolyte.

Potassium is prepared electrolytically in a manner similar to that employed for sodium, from either KOH or halide salts.

Barium has been prepared from a fused fluoride electrolyte.

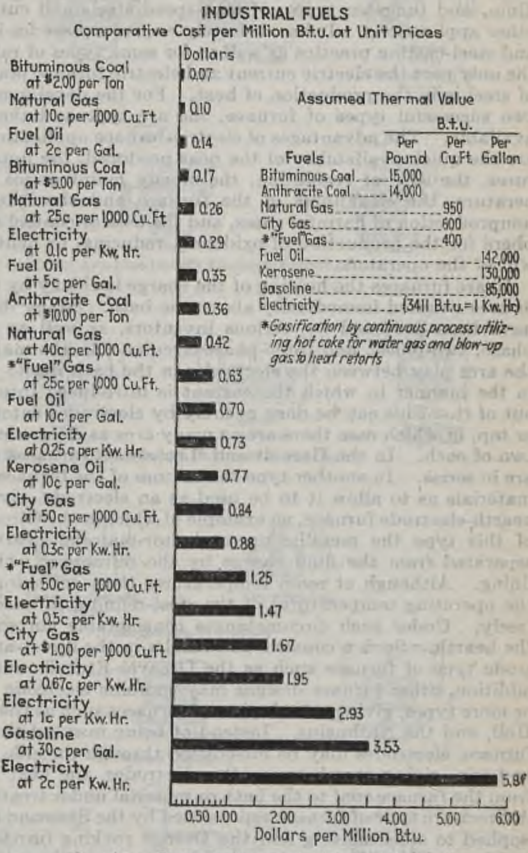
Calcium-lead alloys are prepared by the electrolysis of fused calcium halides with a molten-lead cathode in a furnace externally heated.

Electrothermics

The manufacture of many electrochemical and electrometallurgical products requires temperatures higher than those obtainable by ordinary combustion methods. In this particular field there is no competition with heating obtainable by the use of fuels. In general, electric heating shows greater flexibility of application than do competitive methods of gas or solid-fuel combustion. Electric heat can usually be developed at, or adjacent to, the point of use more rapidly than fuel heat. In electric heating there is a lower temperature gradient between the heat source and the point of use. In contradistinction, flame temperatures are often 1000° to 2000°F. higher than the working temperatures required. Electric heating shows higher relative efficiency, but unless power rates are very low it is more costly (see relations in Fig. 35).

The field of application of electric furnaces can be subdivided into (1) those employed for iron and steel; (2) those commonly employed in the production of ferroalloys; and (3) those especially adapted to the melting of nonferrous metals and alloys. To these must be added (4) the discussion of nonmetallic products of the electric furnace of the arc and resistance types, such as CaC_2, silicon carbide (SiC), and graphite, as well as distillation products such as CS_2.

The electric arc is the simplest and most practical of the general methods of electric heating. It has found the widest application in the steel industry. In general, the arc is formed between two carbon electrodes, but in order to strike the arc the electrodes are brought in contact, after which the gap may be gradually produced and the arc still maintained. If the gap become too wide, the arc will break. Hence, in arc furnaces some means of regulating the distance between the electrodes must be provided. Arc phenomena are explained on the assumption that some of the electrode material is vaporized by the heat of the arc, and that these vapors serve as the conductor. The arc gives us the highest temperatures yet obtained. The limit in commercial practice is set by the materials forming the electrodes and the materials of construction of the furnace body. The intensity of the heat of the arc may be appreciated from the fact that carbon vaporizes at 3500° to 3600°C.



Assumed Thermal Value

Fuels	B.t.u. Per Pound	B.t.u. Per Cu.Ft.	B.t.u. Per Gallon
Bituminous Coal	15,000		
Anthracite Coal	14,000		
Natural Gas		950	
City Gas		600	
*"Fuel" Gas		400	
Fuel Oil			142,000
Kerosene			130,000
Gasoline			85,000
Electricity	(3411 B.t.u.=1 Kw. Hr.)		

**Gasification by continuous process utilizing hot cake for water gas and blow-up gas to heat retorts*

Fig. 35.—Comparative cost of industrial fuels.

Electric Furnaces

Modern electric furnaces are somewhat complicated mechanisms. The most widely used electric furnace is of the three-phase arc type. The complete electrical equipment of a furnace installation includes:

1. A transformer, usually of the multiple type, to step down the voltage of the power supply system to the voltage or voltages needed for the furnace.
2. A secondary bus line and supporting structures between the transformer and the furnace.
3. Reactors, usually in the primary circuit of the transformer, to maintain arc stability and limit current fluctuations to the desired value.
4. Switching, instrument, and meter equipment.
5. Small d.c. electrode motors with automatic regulators.
6. Except in small hand-tilting furnaces, a tilting motor for the operation of the tilting mechanism.

It will thus be seen that the furnace proper, consisting of the shell, roof, electrodes, and tilting mechanism, is only a small part of the complete furnace setup.

Electric Steel Furnaces. The electric furnace has almost entirely replaced the crucible method for steel manufacture, and in the recent development of ferrous alloys, particularly of the stainless, nickel-chromium, vanadium, and tungsten types, of high-speed steels and cutting tools, has found other applications. It has definitely earned a place for itself in foundry work and steel-casting practice as well as for some types of rail steels. In general, the only part the electric current and electric furnace play in the manufacture of steel is in the production of heat. For the development of this heat only two successful types of furnace, the arc type and the induction type, are available. The advantages of electric-furnace application to steel metallurgy are the quick availability of the heat produced, the unusually high temperatures, the ease of regulation, the steady maintenance of any desired temperature, the cleanliness of the furnace and the method of heating, the nonproduction of harmful gases, and the ease of control of the furnace atmosphere for the production of oxidizing, reducing, or neutral conditions at the will of the operator.

In arc furnaces the heating of the charge is chiefly by radiation from an arc or arcs situated immediately above the bath. Many different arrangements have been proposed by various inventors, as well as utilization of single-phase, two-phase, and three-phase circuits. In the majority of arc furnaces the arcs play between the electrodes in the bath itself. Such furnaces differ in the manner in which the current is introduced into the furnace and let out of it. This can be done entirely by electrodes entering through the roof or top, in which case there are as many arcs as electrodes and never less than two of each. In the Heroult and 'Lectromelt furnaces as examples, the arcs are in series. In another type the bottom of the furnace may be built of such materials as to allow it to be used as an electrode, giving a direct-arc free-hearth-electrode furnace, an example of which is the Girod. In a modification of this type the metallic or conductor-material hearth electrode may be separated from the fluid charge by the refractory which forms the hearth lining. Although at room temperatures these refractories are insulators, at the operating temperatures of the steel-refining furnace they conduct fairly freely. Under such circumstances considerable currents can pass through the hearth. Such a construction results in the direct-arc buried-hearth-electrode type of furnace such as the Greaves-Etchells and Electrometals. In addition, other furnace designs may combine electrode arrangements of two or more types, giving mixed types of furnaces such as the Newkirk, the Booth-Hall, and the Nathusius. Instead of being inserted through the roof of the furnace, electrodes may be introduced through the side, or through the sides and top, giving arcs between the electrodes, the heat being reflected down from the furnace roof to the bath or material under treatment. These are the indirect-arc type of furnace represented by the Stassano and the Rennerfelt as applied to steel melting and the Detroit rocking furnace used in nonferrous melting. The different classes of furnaces are illustrated diagrammatically in Fig. 36.

The **Heroult** furnace heads the list of electric furnaces used in the manufacture of steel. It consists essentially of a shallow hearth lined with calcined dolomite or magnesite, roofed with silica brick. The hearth slopes up in front toward a pouring lip. The melting chamber is surrounded by poorly conducting material. The whole furnace is enclosed in a shell or casing of

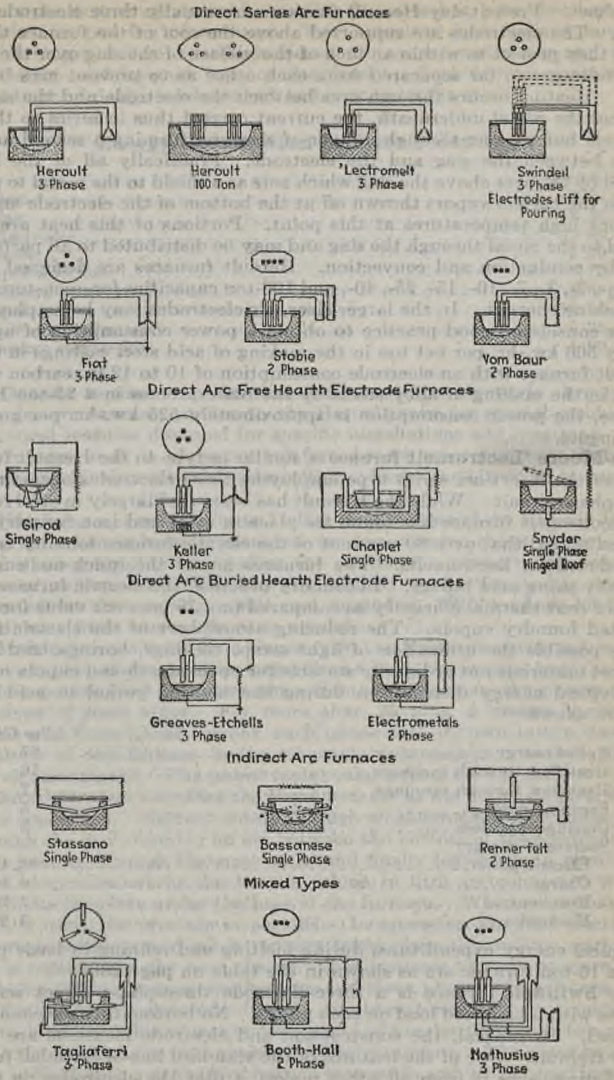


FIG. 36.—Types of arc furnaces.

steel sheet. Present-day Heroult furnaces are usually three electrode, three phase. The electrodes are supported above the roof of the furnace through which they project to within an inch of the surface of the slag over the metal. Electrodes are so far separated from each other as to prevent arcs between them. Heating occurs through arcs between the electrode and the slag, the slag and the metal underneath, the current carried thus in series to the next electrode but passing through a layer of slag and forming a second arc as it jumps between the slag and the electrode. Practically all of the heat is formed by the arcs above the slag which acts as a shield to the metal to protect it from the carbon vapors thrown off at the bottom of the electrode and from the very high temperatures at this point. Portions of this heat are transmitted to the metal through the slag and may be distributed to all parts of the bath by conduction and convection. Heroult furnaces are designed in ½-, 1-, 1½-, 2-, 3-, 7-, 10-, 15-, 25-, 40-, and 100-ton capacities for open-top, chute, or machine charges. In the larger sizes, six electrodes may be employed.

It is considered good practice to obtain a power consumption of approximately 500 kw.-hr. per net ton in the making of acid steel castings in a 3-ton Heroult furnace with an electrode consumption of 10 to 12 lb. carbon per net ton. In the making of alloy steels by the basic process in a 25-ton Heroult furnace, the power consumption is approximately 525 kw.-hr. per gross ton good ingots.

The **Moore 'Lectromelt** furnace is similar in type to the Heroult furnace, being of the direct-arc series type employing three electrodes operating on a three-phase circuit. While the Heroult has been used largely in steel refining, the 'Lectromelt furnace has found its place in steel and iron foundry work. It is estimated that over 80 per cent of the electric-furnace foundry capacity is produced in 'Lectromelts. The furnaces are of the quick-melting type, generally using acid linings. In foundry practice, the electric furnace shows a 75 per cent thermal efficiency as compared to a 35 per cent value for a well-operated foundry cupola. The reducing atmosphere of the electric furnace makes possible the utilization of light scrap, turnings, borings, and similar low-cost materials not ordinarily suitable for open-hearth and cupola melting.

A typical energy distribution during the melting period in acid hearth practice shows:

	Per Cent
Useful energy	67
Heat flow through enclosure	16
Heat flow through openings	7
Escaping gases	5
Cooling-water loss	5
Sources of energy:	
Electric power	93.4
Charge	0.46
Reactions	2.26
Electrodes	3.88

Typical energy expenditures during melting and refining in basic practice with a 15-ton furnace are as shown in the table on page 2805.

The **Swindell** furnace is a three-electrode three-phase direct series arc furnace with a balanced load on each phase. No bottom furnace connections are used. In general, the construction and electrode locations are similar to the Heroult. One of the features of the standard line of Swindell furnaces that distinguishes it from all other makes is that the electrodes do not tilt with the furnace when tapping but are raised in a vertical position until they clear the furnace, after which the ports are covered and the furnace tilted. In this furnace the electrode masts are separate from the furnace shell.

A multiple system of electrode melting furnaces has been developed for the continuous pouring of castings. Two complete furnaces are mounted on a revolving platform by which means the same transformer and set of electrodes are used for each furnace alternately. Each furnace contains the same features of design as described and is complete in itself.

Energy Distribution in Basic Practice

	Per Cent
Steel	45.80
Slag	8.50
Electrodes	10.60
Gases	3.20
Open-door loss	3.70
Cooling water	4.80
Roof surface loss	14.20
Side surface loss	5.20
Bottom surface loss	4.00

The **Greene** furnace may be classified as a three-electrode three-phase direct-series-arc furnace, similar to the Heroult save for special mechanical and electrical features designed for specific installations and locations. The Greene furnace shell is of heavy reinforced-steel construction mounted to roll over on the horizontal axis of the rolling cylinder shell in order to pour the metal.

The **Fiat** furnace is essentially an Italian modification of the Heroult design. The hearth is hemispherical, mounted on two circular shoes of cast steel. Electrodes are graphite, arranged at the points of a triangle and provided with special economizers. The hearth is slightly conductive and in practice is connected on the secondary side and grounded. When phase loads are balanced, very little current flows to ground.

The **Stobie** furnace is fed by a single-, two-, or three-phase current, whichever is available. For capacities larger than 6 tons, the furnace is of the four-electrode type supplied by a two-phase Scott transformer, the phases being independent of each other. For more than 25 tons, a design having six electrodes and three-phase current, each phase with its own return, is used. The feature of the furnace is the electrode economizers attached to the furnace electrode gear. The power factor ranges from 0.85 to 0.93.

The **Girod** furnace resembles the Heroult as far as the major constructional parts are concerned. Current enters through an electrode or electrodes passing through the roof, forming an arc between the bottom of the electrode and the slag, passing through the steel bath, and finally leaving by a number of soft-steel electrodes set in the hearth, which in turn are connected with a copper conductor plate under the base of the furnace. When more than one electrode is used, the arcs are in parallel. In operation the steel electrodes melt down somewhat, and when tapping takes place this molten part runs out but is replaced by the next charge. The furnaces use one or more electrodes, being single phase in smaller sizes and three phase in larger ones.

The **Keller** hearth-electrode furnace is characterized by its composite hearth made of a number of iron rods or a grid of iron bars fastened to a metal base plate from which the bars project vertically upward. The space between the bars is filled with a mixture of burned magnesite and tar rammed in place. The hearth is water jacketed and connected to the source of current. Under furnace-operating conditions the refractory in the spaces between the hearth bars becomes conductive and carries some of the current.

The **Chaplet** furnace is a special modification of the hearth-electrode type which avoids water cooling by the use of a hearth channel which leaves the hearth sideways and connects with a block or blocks of iron in subsidiary chambers at the side of the furnace. Transformer connections are made to these vertical blocks. The steel in the channel of the hearth electrode partially melts during operation.

The **Snyder** furnace uses a hearth electrode in order that high-voltage currents may be employed as far as possible. Commercial refractories at high temperatures become good conductors and thus limit the voltage values possible because of power loss by current leaks through the refractory. In the original type of Snyder furnace the shell was of steel with a thick refractory lining. It tilted for pouring. A single small electrode entered through the roof. From this the current passed in a long arc to the slag and left by a single water-cooled steel conductor set in the hearth, connecting with the metal bath in the furnace. The special feature of this furnace is the hinged roof which permits charging through the top rather than through the door.

The **Greaves-Etchells** furnace consists of a rectangular steel shell with a low-domed, silica-brick-lined roof. The walls are of chrome brick and the hearth a dolomite-magnesite mixture tamped in place with a pitch or tar binder on to a copper plate in the bottom of the shell. Three-phase currents are used, two phases being connected to vertical electrodes entering through the roof and the third phase to ground, to which the hearth is also connected.

When the hearth is hot, its resistance is less than that of the arcs. Special transformer connections are supplied to permit of operation of the furnace in a number of different ways. Power can be put through the top electrodes only and the furnace employed as a direct-series-arc type. By proper switching, power can be introduced through the top electrodes and caused to pass to the hearth, as in the Girod furnace, with the exception that in the Greaves-Etchells furnace the hearth is of the buried type instead of being free.

The **Electrometals** furnace has been developed chiefly in Great Britain. In the United States it is known as the **Grönwall-Dixon** furnace. It uses a three-phase, high-voltage current from Scott transformers so that two phases are employed on the furnace. There are two roof electrodes and a buried hearth electrode which acts as a neutral return. The loads on the roof electrodes are balanced, and very little power goes through the hearth which has a negligible resistance. The silica-brick-lined roof of the furnace is detachable. The hearth electrode is formed of steel conductors embedded in the lowest part, contact with the dolomite hearth being made by a graphite-rich bonding material.

In the **Stassano** furnaces the electrodes enter the furnace shell through the side, there being two electrodes for single-phase furnaces and three or six for three-phase units. The usual type of Stassano furnace is circular in plan, rotating slowly about an axis inclined from the vertical. The roof is domed and lined with magnesite brick, while the melting chamber is lined with dolomite, the brick being insulated from the steel furnace shell. The electrodes are of small diameter and symmetrically arranged around the furnace, three-phase transformer connections being delta.

The **Bassanese** furnace is a modification of the Stassano, having its electrodes mounted on mechanisms which make possible the use of direct as well as indirect action of the arc on the bath. The electrodes are inclined to a variable angle so that the arc may be made free burning or directly projected on the metal.

In the **Angelini** modification of the Stassano, the electrodes are arranged so that arcs may be formed between movable electrodes above the metal surface as well as between each electrode and the metal. This arrangement permits the use of single-phase currents at higher voltages than those employed in other designs.

The **Rennerfelt** furnace is of the indirect-arc type but differs from the Stassano in that it employs two-phase current obtained from a three-phase supply by a Scott transformer as well as using three electrodes, two entering through the side of the furnace and one through the roof. The roof electrode operates as a common or neutral return. Arcs are sprung between the electrodes above the surface of the furnace charge, but the unit may be operated with arcs in contact with the charge for short periods, to increase the activity and fluidity of slags. In the operation of the furnace the arcs burn regardless of whether the furnace is charged or not, so that the hearth may be sintered by electric heat and kept hot for any length of time.

The **Tagliaferri** furnace combines the indirect-arc heating of the Stassano with the direct-series-arc principle of the Heroult. Two- or three-phase, low-voltage current, generally the latter, is used. The three-phase unit has three roof electrodes and during the refining period operates as a direct-series-arc furnace. Three auxiliary electrodes enter through the side of the furnace. During the melting period, arcs are sprung between the roof and corresponding side electrodes.

In the **Booth-Hall** furnace there are one, two, or three roof electrodes for single-, two-, or three-phase circuits, as well as hearth electrodes composed of conductors buried in the hearth. With single- and two-phase power the current passes through the hearth electrodes during the refining period, while with three-phase supply the hearth acts as a neutral point. There is an individual hearth electrode for each roof electrode but located on the opposite side of the furnace instead of being directly underneath the bottom of the roof electrode. An auxiliary roof electrode is provided, connected with the return to the transformer in parallel with the hearth electrode.

The **Nathusius** furnace combines the principles of the direct-series arc, the buried-hearth type, and the resistance idea. As in the Greaves-Etchells, the furnace charge is heated from above by arcs and from below by currents through the hearth electrodes. In the Nathusius furnace the latter type of heating can be independently regulated through star-transformer connections to the hearth electrodes independent of those through the roof. The hearth as well as the whole of the charge may be included in the circuit as an ohmic instead of an inductive resistance. Three-phase currents are used, the transformer secondaries being star connected to the roof electrodes, and their separate returns to the corresponding ones in the hearth. The metal bath in the furnace thus becomes the neutral point of the transformer. A feature of the furnace is its great flexibility.

Induction Furnaces for Iron and Steel. In the United States the use of the induction furnace for ferrous metallurgy has been very limited, finding almost no application; but in the form of the **Ajax-Wyatt** and **Ajax-Northrup** induction furnaces, it has been extensively employed in nonferrous metal melting. In recent years the Ajax-Northrup high-frequency furnace has found greater application in steel and ferrous alloy manufacture, particularly on a small scale and for special materials.

An induction furnace may be described as a step-down transformer in which the metal under treatment is the short-circuited secondary coil or coils. The induction furnace is subject to the magnetic and electrical losses due to hysteresis in the iron core in the same manner as transformers; but on account of the design of the furnace, these losses are usually greater than those in transformers, with resulting lower power factor. In the induction steel furnace there is only one turn in the secondary winding, consisting of the steel to be refined in a ring-shaped channel through the center of which the primary coil and the core pass.

Of the large number of arrangements of primary and secondary which have been proposed, only two are used.

The **Kjellin** furnace is shown diagrammatically in Fig. 37. The core is built up of laminated sheets of soft iron and, like an ordinary transformer core, forms a closed circuit. The primary is made of insulated copper wire and air cooled, or in other designs made of water-cooled copper tube. The secondary is the ring-shaped refractory-lined channel for the steel, insulated by brickwork. The entire furnace is enclosed in a steel shell. Charging and tapping doors are provided in both the fixed and the tilting types. Some of the molten material must always be left in the furnace so that currents can pass on recharging. Furnaces have been built up to 8½-ton capacity, operating on low frequencies of the order of five cycles. Operating data for a 2-ton furnace show a consumption of about 170 kw. at 3,000 volts in the primary and 30,000 amp. in the secondary. Power consumptions are of the order of 750 to 850 kw.-hr. per ton of steel with cold charges of pig iron or pig iron and scrap.

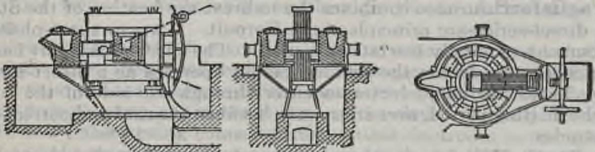


FIG. 37.—Diagrammatic sketch of the Kjellin induction furnace.

The **Röchling-Rodenhauser** furnace uses a combination induction-and-resistance principle. The unit consists of a soft-iron core with two primary windings, while the secondary has the shape of an 8, the middle part of which is a comparatively broad hearth to allow working of the bath and handling of the slags. Auxiliary secondary windings consisting of a few turns of strip copper are provided to heat the central hearth. These are separated from the primaries by a small air gap and connected with hearth electrodes or pole plates at opposite sides of the central hearth, through which currents may pass from the pole plates. These pole plates are made of corrugated cast steel embedded in the furnace wall behind a layer of refractory which at furnace-operating temperatures becomes conducting.

Single-phase furnaces have been built up to a rated capacity of 8½ tons, consuming 700 to 750 kw. with a primary voltage of 4,000 to 5,000. Three-ton furnaces show power factors of 0.7 to 0.8 at a frequency of 25 cycles. With molten charges, about 120 to 160 kw.-hr. per ton of steel is consumed when Bessemer steel is refined to open-hearth quality, while for reduction of the sulphur and phosphorus to 0.01 per cent, 200 to 300 kw.-hr. are required. On cold scrap, the power consumption is of the order of 900 kw.-hr. per ton.

Ferroalloys. In the development of the steel industry the production of alloy steels brought forth a demand for alloying agents in a form readily usable, as well as refining materials. These are the substances in general termed the ferroalloys, including ferrosilicon, ferromanganese, ferrochromium, ferromolybdenum, ferrotungsten, ferrovanadium, and silicomanganese. A large part of these are made in the electric furnace, generally of the arc type.

Ferrosilicon is made either in the blast furnace, in which case the percentage Si seldom exceeds 15 per cent, or in the electric furnace, where higher percentage Si alloys can be made. The raw materials are silica, carbon, and iron, the process being essentially a carbon-reduction one taking place at high temperatures. All ferrosilicon furnaces are of the mixed arc-resistance type.

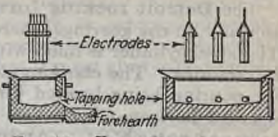


FIG. 38.—Ferrosilicon furnace.

Ferromanganese containing up to about 20 per cent Mn is made in the blast furnace almost exclusively, but there has been considerable electric-furnace manufacture of higher percentage Mn alloys. The furnaces used are low shafts working with open tops, in some cases having a hearth electrode but more usually with a neutral electrode in three-phase furnaces.

Ferrochromium, largely employed for chromium steels, armor plate, projectiles, cutting tools, as well as stainless steels, is made entirely in the electric furnace from chromite, carbon, and slag-forming materials. In the case of the low-carbon-content ferrochromium, silicon may be the reducing agent. The reduction of carbon content of high-carbon ferrochromium is accomplished in furnaces of the same type as used for steel refining, in which the material is remelted by the use of fluxes such as lime and fluorspar, with the addition of chromite.

Ferrotungsten is produced by electric-furnace smelting of a tungsten mineral [wolframite ($FeMnWO_4$), ferberite ($FeWO_4$), or scheelite ($CaWO_4$)], a carbonaceous fuel, and slagging materials, with the addition of iron ore or scrap iron if the iron content of the original tungsten mineral be insufficient.

Ferrovanadium is prepared from vanadium ores which are smelted with iron and fluxes, with reducing agents such as silicon 90 per cent or ferrosilicon, 50 per cent Si, or coke.

Ferrosilicon-titanium was produced in considerable quantities as a by-product of the electric-furnace refining of bauxite, in which the iron, silicon, and titanium content of the bauxite is reduced by carbon to form the ferroalloy. This settles to the bottom of the furnace and is tapped out, while the alumina of the bauxite becomes the slag which is drawn off as a purified alumina, granulated, and worked up for use in the manufacture of aluminum. Ferrosilicon-titanium finds use as a steel deoxidizer.

In general, furnaces for **ferromolybdenum** and **ferrotungsten** are small, of 200- to 750-kw. capacity, and are single phase. Ferromolybdenum meets competition from calcium molybdate. Ferrochromium and ferrovanadium furnaces are 2,000 to 5,000 kw., usually three phase, and like all the other ferroalloy furnaces, are of the arc-resistance type. The ferrosilicon, ferromanganese, and silicomanganese furnaces are larger, being 3,500 to 12,000 kw., three phase. Electrode current densities are 30 to 60 amp. per sq. in. cross section. Power consumptions are of the order of 1.2 to 3 kw.-hr. per lb. ferroalloy, being highest for ferromolybdenum.

Electric Furnaces for Nonferrous Metals. For nonferrous melting, three furnaces are of industrial importance in the United States and find wide application. The first to be considered is the *Detroit rocking furnace* in which it is estimated that about 85 per cent of the foundry output of the United States is melted and 20 to 25 per cent of the rolling-mill tonnage of brass and Cu-Zn alloys. It is estimated that 75 to 80 per cent of the rolling-mill production of brass in the United States is melted in *Ajax-Wyatt induction furnaces* which also find some application in foundries. The third furnace is the *Ajax-Northrup high-frequency induction unit.*

The **Detroit rocking furnace** consists of a steel cylinder mounted horizontally on cog gearings whereby it can be rocked by means of a small motor. The steel cylinder is lined with insulating brick which are in turn covered by refractories. The charging door is on the upper side of the drum, the spout for pouring being located directly below it. Graphite electrodes are introduced horizontally through centers at the ends of the drum. They can be adjusted for length of arc and power input, usually by hand. Single-phase a.c. is employed. Heating of the metal is not by direct contact with the arc but by reflected or radiated heat. During charging, the electrodes can be withdrawn until their ends are flush with the refractory lining of the furnace to avoid breakage. During operation, as soon as superficial melting of the charge is started, rocking of the furnace is caused to take place. The furnace has the advantages of rapid melting, a totally enclosed body so that volatilization losses, particularly of zinc, are cut down, as well as control of the furnace atmosphere which, due to the presence of the electrodes, is ordinarily reducing. Inasmuch as each charge can be completely poured from the furnace, it shows great flexibility when different alloys have to be melted. The furnace has been applied to nonferrous alloys, particularly brass and bronze, and to a smaller extent to grey iron, special iron-alloy work, and the duplexing of cupola-melted iron. On brass it shows a power consumption of 250 to 300 kw.-hr. per ton; 540 kw.-hr. per ton on melting cast-iron borings; 600 kw.-hr. per ton on synthetic grey iron, and 100 kw.-hr. per ton on duplexing cupola iron.

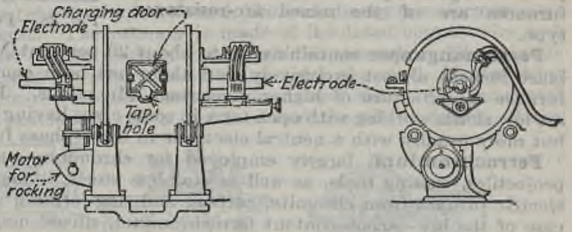


FIG. 39.—Detroit rocking furnace.

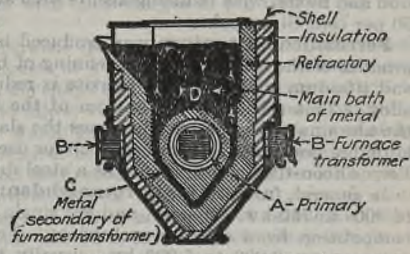


FIG. 40.—Ajax-Wyatt induction furnace.

The **Ajax-Wyatt** induction furnace is illustrated in Fig. 40. In operation, a single-phase alternating current of 220, 440, or 550 volts is fed to the primary coil *A* in the center of the furnace, which in turn energizes the furnace transformer *B*. This in turn induces a voltage in the V-shaped channel *C* which is always filled with molten metal or alloy and acts as a secondary of the furnace transformer. A current of relatively high value flows in this V-shaped channel and, owing to the resistance of the molten metal, the channel is heated at a constant rate. At the same time the magnetic fields about this channel set up electromagnetic forces which eject the hot metal out of the channel, so that it is in turn replaced by colder metal from the main bath above the melting channel. The circulation follows the path shown in the figure. Thus with a furnace stationary throughout the melting cycle, the metal bath is thoroughly mixed by an internal automatic stirring action, while the electric energy is converted to heat directly within the metal. The furnace has the disadvantage that it must always be partially filled in order to operate and cannot stop with cold charges. It cannot, therefore, be used with charges of a varying nature without some trouble. It has the advantages, however, of a simple mechanical design, great steadiness in working, small metal loss, small wear and tear on the lining, and in continuous operation a low energy consumption. This is of the order of 250 to 270 kw.-hr. per ton of red brass (85 per cent Cu, 5 per cent Pb, 5 per cent Zn, 5 per cent Sn); 200 kw.-hr. per ton of yellow brass (75 per cent Cu, 2 per cent Sn, 3 per cent Pb, 20 per cent Zn) for plumbing fixtures; 275 kw.-hr. per ton on nickel-silicon; 285 kw.-hr. per ton of 4 to 5 per cent tin bronzes, 310 kw.-hr. per ton of copper, and 90 kw.-hr. per ton of zinc.

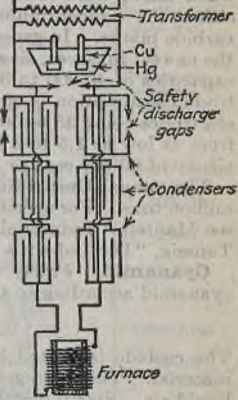


FIG. 41.—Ajax-Northrup high-frequency induction furnace (small sizes).

In the high-frequency **Ajax-Northrup** induction furnace, the material to be heated is not in the shape of a ring as in the ordinary induction furnace but is held in a crucible placed in the field of a high-frequency coil. The heating is produced by eddy currents generated in the material to be melted. If this is a nonconductor, a conducting crucible is used. Since the eddy currents increase as the square of the frequency, the reason for the use of high frequency is evident. The frequencies are of the order of 10,000 to 12,000 cycles per second. It has been found most convenient and satisfactory to use the oscillatory discharge of a bank of condensers as a source of high-frequency current. The desired frequency is obtained by the proper proportioning of the capacity and inductances of the oscillatory circuit. Figure 41 shows the circuits of a single-phase Ajax-Northrup furnace. The spark gap consists of water-cooled copper over mercury in a hydrogen atmosphere. With the two gaps, in whichever direction current from the secondary of the transformer is flowing, positive current must always leave a mercury surface. When the voltage between the mercury and the graphite reaches a certain minimum value, the mercury opens the circuit completely with great suddenness, causing very rapid and regular oscillations. The power input is controlled by regulation of the spark gaps. The induction coil around the furnace is a water-cooled flattened copper tube. The furnaces have been applied for nickel-silver where the energy consumption is 0.17 kw.-hr. per lb.; special steels produced at 660 kw.-hr. per short ton; silver and silver alloys, precious metals, special alloys, stainless steels, and a variety of other products.

Electric Furnace Products

Calcium carbide is made in the electric furnace according to the reaction

$$CaO + 3C = CaC_2 + CO - 125{,}000 \text{ cal.}$$

It does not seem probable that temperatures of 2000°C. are exceeded in the furnaces. The commercial material is dark colored and crystalline, but if pure it is colorless and transparent. The 80 per cent pure product melts in the neighborhood of 1800°C. The raw materials are carbon in the form of charcoal, low-ash anthracite, low-ash coke (or sometimes petroleum coke), and lime.

The furnaces in use are very large, being of the vertical-arc type. Ingot furnaces, in which the lower electrode is a small car which can be removed when filled, were formerly used, but hearth-electrode tapping furnaces of either the noncontinuous or continuous type are generally found in all carbide plants. In general, the furnaces are three electrode three phase. In the newest furnaces, single electrodes of sectional construction are found with capacities of 40,000 to 300,000 amp. The furnaces are single phase. Electrode consumption is of the order of 2 to 4 per cent, current density on the electrodes being 35 to 50 amp. per sq. in., while energy consumptions may be from as low as 1.5 to 3 kw.-hr. per lb., with perhaps an average energy efficiency of 60 per cent or better.

The annual United States production of CaC_2 is of the order of a quarter million tons. For specific data on furnaces, capacities, operation, and design, see Mantell, "Industrial Electrochemistry," 2d ed., McGraw-Hill, 1940; and Taussig, "Die Industrie des Kalziumkarbides," Knapp, Halle, 1930.

Cyanamid. Finely divided CaC_2 absorbs nitrogen at 1000°C., giving cyanamid according to the equation

$$CaC_2 + N_2 = CaCN_2 + C$$

The carbide is heated in furnaces, and nitrogen is passed over the heated material. The nitrogen is generally made by fractional distillation of liquid air. Starting with a carbide containing 75 to 80 per cent CaC_2, 80 to 90 per cent of the theoretical amount of nitrogen will be absorbed, resulting in a product containing about 20 to 22 per cent nitrogen with lime and carbide as impurities. If cyanamid be treated with superheated steam, ammonia is evolved according to the equation

$$CaCN_2 + 3H_2O = CaCO_3 + 2NH_3$$

Cyanamid may be converted into cyanides by the fusion of a mixture of salt, cyanamid, and carbide, and a rapid chilling of the melt, the equation being

$$CaCN_2 + C + 2NaCl = CaCl_2 + 2NaCN$$

Silicon carbide (trade names *Carborundum*, *Crystolon*, etc.) is made by heating a suitable charge of carbon and silica sand in a horizontal resistance furnace. The carbon used is either high-grade anthracite or good-quality coke such as petroleum coke. The proportions roughly correspond to the equation

$$SiO_2 + 3C = SiC + 2CO$$

but a slight excess of coke is commonly used, with the addition of some sawdust and sometimes of salt. The sawdust is added to increase the porosity of the charge, while the salt at the furnace temperatures distills and attacks some of the metallic impurities to form volatile chlorides, which are thus removed from the reaction zone. Newer practice omits the salt.

The furnace construction is simple. The two ends or headers are permanent stout brick walls through which the furnace electrodes pass. The side walls of the furnace are of either brick or other forms such as refractory-faced castings to hold the charge in place. The bottom of the furnace is usually made of insulating firebrick or gannister. Through the center of the furnace a heating core connects the electrodes, the charge surrounding the heating core (see Fig. 42). Operating data are given in Table 31.

Table 31. Products of Resistance Furnaces

Factors	Graphite	Silicon carbide	Fused alumina
Raw materials	Low-ash anthracite or petroleum coke	Coke, 98% silica sand	Bauxite, coke, scrap iron
Additions	...	Sawdust and salt	
Furnace:			
Type	Resistance	Resistance	Vertical arc resistance
Size	1,000 h.p.	2,000 h.p	550 kw.
Length, ft	30	30	
Cross section, ft	...	10 × 10	
Cross section of charge, ft. diam	2	3	
Walls	Refractory brick or concrete blocks	Refractory brick, cast-iron or steel supports	Steel, water cooled
Initial voltage	200	230	100–110
Final voltage	80	75	
Initial current	...	6,000	2,500
Maximum current at 200 volts, amp	3,700		
Final current, amp	9,000	20,000	
Current density across furnace charge, amp. per sq. ft.	900–2,250	650–2,200	
Core temperature, °C	...	2,350	
Furnace temperature, °C	2,200	1,820–2,220	2,000–2,200
Length of run, hr	24	36	
% Conversion of material	90–100	70–80	95–100
Energy consumption, kw.-hr. per lb	1.5	3.2–3.85	1.0–1.5
Energy efficiency, %	25–30	55–70	

Boron carbide, B_4C, is made in a resistance furnace from B_2O_3 and coke [Ridgway, *Trans. Electrochem. Soc.*, **66**, 117 (1934)].

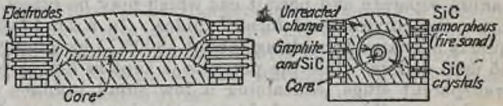


FIG. 42.—Silicon carbide furnace.

Graphite is made in a horizontal resistance furnace similar in construction to the SiC furnace. Graphite is produced either as a powder or as shaped articles such as electrodes, brushes, etc. When loose graphite or powder is made, low-ash anthracites or petroleum cokes are employed as raw materials. One theory of graphite formation is that the ash content of the raw material, containing iron, aluminum, and other oxides, functions as a catalyst for the conversion of amorphous carbon into graphite at the furnace temperature through the intermediary formation of carbides, which dissociate to give graphite. In the preparation of graphite articles the materials are formed and baked into an amorphous-carbon article, after which they are graphitized in the electric furnace. The ash content of graphite is very low, in that at the furnace temperature most of the metallic oxides are volatilized. A sketch of a typical furnace is given, as well as operating data. The ready

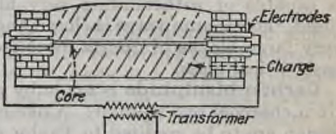


FIG. 43.—Graphite furnace.

machinability of graphite is well known, as well as its resistance to most forms of chemical attack. It is the standard anodic material for electrolysis of NaCl to produce chlorine and caustic.

Fused alumina is made in the electric furnace. The product is extensively used as an abrasive, particularly in grinding wheels, and as a refractory. The raw material is a red bauxite containing a few per cent of oxides other than alumina (iron and silicon) which give tougher products than those made from purer bauxite. The material is first calcined to remove water, and the fusion done in vertical arc furnaces with a bottom electrode. The ore is fed into the furnace, melted in the arc between the top and bottom electrodes until the furnace is filled, after which the entire shell of the furnace is removed, the block of fused material allowed to cool, taken away from the furnace bottom, and broken up into commercial products. Operating data are given in Table 31.

Silicon is made in three-phase arc furnaces with bottom electrodes by the reduction of silica with coke, a 95 per cent silicon being produced. It may also be made from cheap SiC and silica sand. The energy efficiency of the furnace is given at about 50 to 55 per cent. The material is used for high-grade silicon steels and as a reducing agent for the production of low-carbon ferrovanadium and ferrochromium, as well as silicon-aluminum and silicon-copper alloys.

Fused silica (vitreosil, etc.) is made from silica sand by passing a current through carbon rods or plates embedded in the material. When sufficient sand has been fused, the carbon resistor is withdrawn. By utilizing the hole left in the interior of the melt, the latter is blown and molded to the desired form. The crude articles thus obtained are trimmed and polished by the use of the oxyhydrogen flame, sandblast, and abrasives.

Fused Quartz. In recent years clear fused quartz has been made under pressure in electric furnaces. The clean natural crystals of small size are packed as densely as possible in a graphite or carbon crucible, so that in the inevitable cracking of the crystals as the temperature is raised the parts cannot separate and permit gas which may be present in small quantities to enter the crevices and form bubbles. In a modified vacuum furnace, the quartz is heated to melting (about 1800°C.) as quickly as possible, usually in 45 min. or less, while the pressure is kept as low as possible. The resultant transparent slugs, containing a few small bubbles, are placed in another graphite crucible suspended in a vertical carbon-tube furnace, with a graphite piston closely fitting the crucible and weighted. The slugs are heated to fusion, the bubbles mostly collapsed by the weight which also extrudes the quartz through the bottom of the crucible in rods, tubes, and other desired forms. When large blocks are to be made, a vacuum furnace is used which is capable of withstanding very high pressure. As soon as the quartz is fused, the vacuum valve is closed and the pressure raised. Thus are produced very large blocks of quartz more free from bubbles than many kinds of the best optical glass.

Carbon bisulphide is made by one company in an arc resistance furnace from charcoal and sulphur. Coke cannot be used. The process and furnace design are to be credited to Taylor (*Trans. Am. Electrochem. Soc.*, vols. 1 and 2). The CS_2 distills out of the furnace and is condensed in external condensers. Data are given in Table 32.

Phosphorus is electrothermally produced from phosphate rock and reducing materials such as carbon, with sand as a slag-forming product, in arc furnaces. The phosphorus distills out of the furnace and is collected.

Phosphoric acid is also produced in a somewhat similar manner, but as the phosphorus distills from the furnace it is allowed to oxidize to form phosphoric acid. Data are given in Table 32.

Table 32. Distillation Products of Electric Furnaces

Factors	Phosphorus	CS_2
Raw materials	Bone ash or phosphate rock, coal, and SiO_2	Charcoal, sulfur
Furnace:		
Type	Arc, carbon lining, vertical electrode	Resistance, two phase, a.-c., shaft feed
Capacity, kw		240–330
Current, amp		4,000–6,000
Voltage		60
Temperature, °C	1,150–1,450	
Production per 24 hr., lb		14,000
Energy consumption, kw.-hr. per lb	4.0–5.5	0.4–0.5
% thermal efficiency		30–45
% P recovered	80–90	

Zinc is produced by electrothermal processes whereby zinc ore is reduced to metal, volatilized out of the furnace, and condensed.

Gaseous Electrothermics

Nitrogen Fixation. In the United States there has been very little application of the fixation of nitrogen from the air in electric furnaces either of the arc or of the spark type. Atmospheric air is a mixture of 80 per cent by volume of nitrogen and 20 per cent of oxygen. The simplest fixation process would be one that would combine these, converting them into nitrogen oxides which in turn could be made into HNO_3 and nitrates. This can be done by the processes of Bradley and Lovejoy, Birkeland and Eyde, Schoenherr, and Pauling. All of these employ electric discharges through air to produce a very high temperature. The spark or arc is merely a heating means to reach the high temperature necessary for the combination of the nitrogen and oxygen. The mixture of air and oxides of nitrogen thus produced is treated with water or with an alkaline solution to give HNO_3 or nitrates, or a mixture of nitrates and nitrites.

The higher the temperature the more rapidly does the conversion of nitrogen and oxygen into nitrogen oxides take place. But the decomposition reaction proceeds in the same order. It is therefore necessary to have a very high temperature for efficient reaction, with a quick removal of the reaction products from the temperature zone so that they may be cooled down as rapidly as possible to prevent decomposition.

In the **Bradley-Lovejoy** process, now no longer in operation, a wheel carrying a set of electrodes was rotated so that the electrodes passed opposite and by a stationary set of electrodes so as to make and break continuous sparks, at the rate of about 6,900 sparks per second, in the space through which the air was passed. The units were limited in size and the desire to have larger ones led to the construction of arc furnaces.

In the **Birkeland-Eyde** process, employed in Norway where cheap power is available, the arc is deviated magnetically by means of a single-phase magnet field until the arc breaks, then a new arc is formed and the cyclic process continued. The high-voltage flame or arc is formed between two water-cooled copper electrodes by the use of a 50-cycle current at 500 volts.

26

Table 33. Materials of Construction

1. Electrolytic cells—aqueous electrolytes
   - *a.* Anodes
   - *b.* Cathodes
   - *c.* Diaphragms
   - *d.* Tank materials
   - *e.* Pipe lines, circulation systems
   - *f.* Floors and buildings
   - *g.* Busbars and power lines
2. Electrolytic cells—fused salt electrolytes
   - *a.* Anodes
   - *b.* Cathodes
   - *c.* Diaphragms
   - *d.* Container or tank materials
   - *e.* Circulation systems, stirrers
   - *f.* Floors and buildings
   - *g.* Busbars and power lines
3. Electric furnaces
   - *a.* Electrodes and their operation
   - *b.* Furnace linings
   - *c.* Charging mechanisms
   - *d.* Gas and dust-collecting systems
   - *e.* Furnace body and shell
   - *f.* Auxiliary equipment
   - *g.* Busbar and power lines

Material	Use	Characteristics	Remarks
		1*a.* Aqueous Electrolytes—Anodes	
Carbon, amorphous...........	In some early forms of cells	Appreciably attacked by chlorine	No longer used save in unusual cases, in NaCl electrolysis
Chilex, mainly copper silicide...	Copper electrowinning	Does not introduce Fe into solution. Films formed on surface catalyze anodic reactions	Used at Chile Copper Company, Chuquicamata, Chile
Graphite, artificial............	Chlorine and chlorate cells	Only slightly attacked by Cl_2; long life which can be lengthened by impregnation with Co salts, linseed oil, and synthetic resins, etc.	Standard for NaCl electrolysis. Can be machined to almost any shape. Not useful in cells containing H_2SO_4 or acid solutions; disintegrates in these
Iron........................	Electrolytic H_2-O_2 cells	Not attacked in alkaline solution. Can be passivated in certain alkaline solutions. Attacked by solutions containing chlorides, nitrates, sulfates, or acid solutions. Iron salts in solution are undesirable, causing lowered energy efficiency	Standard anode for electrolytic H_2 and O_2—generally coated with Ni to reduce O_2 overvoltage. Cheapest material of construction, universally available

Iron-silicon alloys, duriron	Formerly in copper electrowinning	Attacked by chlorides and nitrates in sulfate solutions. Introduced Fe into solution	Used only in isolated cases
Lead (lead-antimony alloy, chemical lead).	Electrodeposition and electrorefining of metals; electrowinning	Not attacked by sulfates in absence of chlorides. Often becomes coated, and then acts like a lead peroxide electrode	One of the standard insoluble or non-attackable electrodes
Lead-silver alloys	Zn electrodeposition	See Pb	Have been proposed as anode in certain per cent. for NaCl electrolysis. Form protective films resistant to anodic corrosion. Films have low O_2 overvoltage
Magnetite, fused	Early chlorine cells; copper electrowinning	Used as hollow castings. Good resistance to attack but introduces Fe into solutions	Castings were fragile
Platinum	Electrochemical analysis; persalts, peracids	Excellent insoluble electrode. Low O_2 overvoltage. Resistant to most forms of attack but affected by chlorides. Generally used in form to give greatest surface per unit of weight, as screen, gauze, thin sheet, or even foil or plating over other metals	High cost is disadvantage. Used only where other materials fail and Pt is needed. Platinized Pt electrodes give lower O_2 overvoltages

1*b*. Aqueous Electrolytes—Cathodes

Aluminum	Zn electrowinning; Ni refining	Used to form starting sheets	High overvoltage of Al allows deposition of metal. In Ni refining, Al blanks given a Na_2S dip so that starting sheet is easily pried off
Copper	Starting sheet blanks for Cu refining	Not soluble in electrolyte, when used as cathode with current on cell	Competes with Pb blanks
Iron	Chlorine and caustic		Universally used
Lead and lead alloys	Starting blanks for Cu refining and electrowinning	Not attacked in sulfate solutions	Usually oiled or coated to allow easy removal of formed starting sheet

1*c*. Aqueous Electrolytes—Diaphragms

Alundum	Electrolytic organic preparations	Ordinarily high voltage drop	
Asbestos	Chlorine and caustic cells	Resistant to almost all forms of attack. Can be woven so as to maintain porous structure	Almost universal. Can be built to show very low voltage drop or pressure loss through diaphragm
Asbestos-$BaSO_4$ and/or iron oxides, and plastics.	Special Cl_2-NaOH cells, Townsend		Offers mechanical difficulty in some cases. Is the so-called "Baekeland diaphragm"
Clay ware	Laboratory operations	Ordinarily high voltage drop	
Metal screens	Filter press and bipolar H_2-O_2 generators	Act as moderately effective diaphragms	Are mechanically strong but costly
Silica, "filtros"	Electrolytic organic preparations	Ordinarily high voltage drop	
Textiles (duck, linen, muslin)	Cathode boxes in Ni refining, Ag refining	Acts largely as filter or separator	Keeps anode slimes away from cathode deposit. Low voltage drop

Table 33. Materials of Construction—(*Concluded*)

Material	Use	Characteristics	Remarks
1*d*. Aqueous Electrolytes—Tank Materials			
Carbon-sulfur-sand mixtures...	General use	Properly made mixes are very acid resistant	Easily fabricated, cast, or built up
Ceramics or clay ware........	Silver refining		
Concrete, lined...............	Electrorefining and electrowinning; chlorine cells	Linings of mastic, asphalt, bitumen, or pitch determine character of tank	Concrete should be of slow setting type
Plastics (bakelite, celluloid, etc.)	Electroplating	In plating barrels, etc., is satisfactory	
Soapstone.....................	Nelson chlorine cell	Shows appreciable resistance to salt corrosion	Easily fabricated
Steel.........................	Chlorine cells, H_2-O_2 cells, chlorate cells	Steels and irons are alkali resistant	Standard satisfactory tank material
Steel, rubber lined............	Electroplating and similar operations	Character depends upon type of rubber and kind of construction used for tank	Steel tanks welded; several rubber-lining processes. Tanks stand temperatures to boiling point of H_2O
Wood (cypress, red wood, etc.).	Electroplating, electrorefining, electrowinning	Kind of wood selected depends upon electrolyte, concentration, constitution, temperature, etc.	Wood protected by acid-resistant paints of bitumen or asphalt type
Wood, lead lined..............	Electrorefining (Cu, etc.), electrowinning	Used where solutions are free of chlorides	Chemical lead, 6–8% Sb, used for stiffness. Sheets "burned" together
2*a*. Fused Salt Electrolytes—Anodes			
Carbon, amorphous...........	Aluminum, cerium, etc.	Oxidizes around 500°–550° C. characteristics in Table 34	Cheap; easily made. See standard sizes, Tables 35, 36, 37
Graphite......................	Na, Mg, Be electrolysis, Pb alloys	Resists oxidation better than amorphous carbon. Better anode for Cl_2 liberation	Often too expensive. See standard sizes, Table 38
Iron..........................	Castner sodium cell	Sometimes iron alloys or Ni-Fe used	Anode is in form of casting, is cheap, and has relatively long life
2*b*. Fused Salt Electrolytes—Cathodes			
Carbon.......................	Aluminum	Carbon in plastic mass form, rammed into place in steel box or pot and baked *in situ*	Raw materials are ground coke, petroleum coke, tars, and pitches
Iron..........................	Na from NaCl, Na from NaOH, Be, lead alloys, Ca from $CaCl_2$	Is satisfactory cathode for Na deposition; does not alloy with Na	In Be electrolysis, cathode is special water-cooled casting. In Na from NaOH, cathode is cylindrical. In Pb alloy manufacture, iron pot holds molten Pb which is cathode
Steel.........................	Magnesium	Cathode is steel pot container	Cathode pot is externally heated
2*c*. Fused Salt Electrolytes—Diaphragms			

Iron or iron-alloy wire gauze	Na from NaOH, Na from NaCl	Diaphragm prevents passage of Na metal particles to anode. Diaphragm material resistant to electrolyte	Openings in mesh must not be so fine as to clog readily, but still small enough to serve as diaphragm
	2d. Fused Salt Electrolytes—Containers		
Steel or cast iron	Al, Mg, Ca, Na, Be, etc.	Common container is made of steel plate or cast in one or more pieces of ordinary or special cast iron	Frozen layers of electrolyte adjacent to walls often protect shells or containers if they are unlined
	3a. Electric Furnaces—Electrodes		
Carbon	General electric furnace work; iron and steel furnaces; carbide, etc.	Characteristics are a function of raw materials (coal, petroleum coke, etc.) and method of manufacture (extruded, molded, etc.). Can be turned and machined, but operations are more difficult than with graphite	For standard sizes, connecting pins, carrying capacities, etc., see Tables 35, 36, 37. Cheaper than graphite. Lower current density must be used than for graphite
Carbon, sectional, of prebaked shape. "Miguet."	Carbide and other furnaces	Allows continuous feed; blocks of electrodes joined to frame	For details, *see Trans. Am. Electrochem. Soc.*, **52**, 335 (1927)
Carbon, baked in place, "Söderberg."	Iron and steel furnaces, carbide	Carbonaceous paste is fed to electrode shell which burns off as electrode is consumed. Electrode baked in place. Electrode is continuous	
Graphite	General electric furnace work; Detroit rocking furnace, non-ferrous melting, cyanamid	See Table 34 for electrical and physical values. Can be readily machined, joined, and cut to unusual shapes	For standard sizes, connecting pins, carrying capacities, see Tables 38, 39, 40. More expensive than carbon
	3b. Electric Furnaces—Linings		
Carbon, preformed block	Ferrovanadium and other furnaces	Useful only in reducing atmosphere	Can stand higher temperatures than any common refractory
Chromite	Basic electric furnace practice	Has limits of use and special applications	Expensive; has high melting point
Clay brick	CS_2, phosphorus, etc.	Fire brick is suitable	Brick serves as structural material as well as to resist action of furnace charges and gases
Dolomite	Basic electric furnace practice	Has limits of use and special application	Is cheap; widely used
Ground brick	Base of resistance furnaces, as SiC and graphite	Electrical insulator	Cheap
Layer of material under treatment.	Fused alumina	Unfused layer of material adjacent to shell protects it	
Magnesite	Basic electric furnace practice	Has limits of use and special application	Must be carefully burned to prevent shrinkage
Silica-type refractory	Acid electric furnace practice		

NOTE. For specific data and information on electric furnace refractories, see chapter by A. V. Bleininger in Liddell, "Handbook of Chemical Engineering," McGraw-Hill, New York, or Norton, "Refractories." McGraw-Hill, New York, 1931.

Table 34. Comparison of Amorphous and Graphitic Carbon Electrodes

	Acheson graphite electrodes	Amorphous carbon electrodes*	Gas-baked amorphous carbon electrodes†	Large electrothermal electrodes	Copper	Aluminum	Iron
Specific resistance, ohms per in. cube	0.00032	0.00124	0.00161	0.00220	0.00000065	0.00000120	0.00000380
Specific resistance, ohms per cm. cube	0.000813	0.00325	0.00400	0.00550			
Comparative section area for same voltage drop	1.	3.8	4.4	6.8			
Weight, lb. per cu. in.	0.0574	0.0564	0.0560	0.058	0.320	0.090	0.280
Weight, lb. per cu. ft.	99.0	97.5	97.0	100.0	554	155	484
Apparent density, g. per cc.	1.585	1.558	1.55	1.60			
Tensile strength, lb. per sq. in.							
Lengthwise	800–1,000	1,000–1,500	1,000–1,500	....	20,000–30,000	24,000–30,000	30,000–50,000
Crosswise	500–600	600–900	600–900				
Temperature of oxidation in air, °C.	640	500	500	500			

* Very good electric baked electrode (small).

† Typical small electrode.

In ordinary operation of the furnace, a flame is formed at each reversal of the current every 0.02 sec. The furnace consists of a narrow iron chamber lined with firebrick. Air enters through the walls and leaves the furnace at a temperature of 1000°C., containing 1 per cent NO. The gases pass through the steam boiler in which they are cooled to 200°C., then through a cooling apparatus where the temperature is reduced to 50°C., and then into an acid-proof, brick-lined oxidation chamber where the reaction $NO + \frac{1}{2}O_2 = NO_2$ is completed. The NO_2 is then absorbed in water to form HNO_3 which is concentrated. The net yield is stated to be 62.5 g. pure HNO_3 per kilowatt-hour.

The **Schoenherr** furnace has as its characteristic feature the use of a very long a.c. arc around which the air moves in a helical path. The process was used successfully for a time by the Badische Co. in Germany and has been operated on a commercial scale in Norway. In the latter plant the arc was 5 m. long in a 447-kw. furnace and 7 m. long in a 746-kw. furnace. It is estimated that 3 per cent of the power is used in the formation of the NO, 40 per cent is recovered in the form of hot water, 17 per cent is lost by radiation, 30 per cent is used in the steam boiler in which the gases are cooled, and 10 per cent is removed by water cooling after the gases have passed through the steam boiler.

The **Pauling** process uses a fan-shaped arc or an electric discharge quite similar to that obtained in a horn lightning arrester. The arc is lighted where the electrodes are nearest together and is blown upward by the hot air rising between the electrodes. The arc is broken every half period of the a.c. In a 400-kw. furnace, the arc length is about 1 m. at 4,000 volts. The NO concentration is about 1.5 per cent and the yield 60 g. HNO_3 per kilowatt-hour. The process is in use in Tyrol and in France where very cheap power is available.

Ozone may be formed in various ways, but its commercial production has been only by the electrostatic method. It is probable that ionization of air takes place with consequent dissociation of the oxygen, which recombines to form ozone. Within working limits in commercial ozone generators, the production is roughly proportional to the electrostatic intensity, and with a.c. to the frequency. Ozone generators have been made in various forms, but in general two or more discharging surfaces are placed in juxtaposition so as to form a condenser with an air gap which may or may not be furnished with a dielectric element. Most successful ozone generators have smooth electrodes and dielectrics and assume either the cylindrical or the plate form. They are operated on voltages ranging from 5,000 to 25,000 and frequencies from 50 to 500 cycles. An average yield is given as about 50 g. per kw.-hr., although much higher yields have been claimed. Concentrations of 1 to 3 g. per cu. m. air (although in some special cases the figure may reach 5 g.) are produced commercially. It is essential to refrigerate the air to dry it.

The tubular ozonizers are more compact and operate at lower voltages than the plate forms.

Electrostatic Precipitation. For descriptive matter, operation, and theory, reference should be made to Section 15, pp. 1867–1878.

Materials of Construction for Electrochemical Processes

Materials of construction for electrochemical apparatus may in general be subdivided into three classes: (1) those employed in electrolytic cells in which aqueous electrolytes are used, (2) those finding application in fused electrolyte work, and (3) those in electrothermic operations. These may be further subdivided as in Table 33.

26

In fused salt electrolysis, the containers are almost universally steel, either bare or lined with carbon. When the containers are unlined, an effective lining is usually formed by a fused layer of the electrolyte or the material being melted. Such is the case in fused alumina furnaces where the furnace consists of a steel shell which is protected by a layer of the alumina adjacent to the steel. In electric-furnace work the container of the furnace is always sheet steel or cast iron refractory lined, the type of refractory used depending upon the furnace operations. The refractory may be acid, in which case it is silica in the form of silica brick or ground material mixed with a binder and tamped into place; or basic, when it is either magnesite, dolomite, or in some cases chromite. In addition various other refractories may be used, such as those of the clay type or the aluminous refractories and in special cases silicon carbide materials. For preventing large heat losses, insulating refractories may be used, but they are never exposed to the furnace bath. They may be fire clay, diatomaceous earth, kieselguhr, magnesia, or asbestos compositions, as well as other special products. In some unusual cases where other materials fail as the result of high temperatures, the temperatures exceeding the point where the refractory gives satisfactory load service or else the melting point of the refractory being exceeded and reducing conditions maintained in the furnace, carbon refractories find application. They are used in the form of preshaped blocks or they may be rammed in place. For the base of resistance furnaces, particularly of the graphite, silicon carbide, and electrode baking furnaces, ground brick grog or gannister is employed. For the side walls and heads of such furnaces, high-quality clay refractory brick serves. For insulating materials in such furnaces the usual substances cannot be employed, in that they would contaminate the charge of the fur-

Table 35. Standard Amorphous-carbon Electrode Data (*National Carbon Co., Inc.*)

Diam., in.	Area, sq. in.	Weight per 60 in. length threaded, lb.	General limits current-carrying capacity, amp.	Current density, amp. per sq. in.
		Round Electrodes		
6	28	92	1,200– 1,700	40–60
8	50	158	2,000– 3,000	40–60
10	79	250	3,000– 4,800	40–60
12	113	360	4,500– 6,800	40–60
14	154	510	5,400– 8,500	35–55
16	201	590	7,000–11,000	35–55
17	227	730	7,900–12,500	35–55
20	314	971	11,000–17,300	35–55
24	452	1437	15,000–25,000	35–55
		Square Electrodes		
6	36	125	1,500– 2,200	40–60
8	64	224	2,500– 3,800	40–60
10	100	333	4,000– 6,000	40–60
12	144	500	5,500– 8,500	40–60
14	196	647	6,800–10,800	35–55
16	256	864	9,000–14,000	35–55
20	400	1320	14,000–20,000	35–55

nace. Insulation is therefore obtained by the use of relatively thick layers of finely ground calcined carbonaceous materials such as those made from petroleum coke or low-ash anthracite.

Table 36. Amorphous Carbon—Weights of Unthreaded and Threaded Electrodes (*National Carbon Co., Inc.*)

Size, in.	Unthreaded	Threaded
4 × 40	30	
5⅛ × 48	63	
6 × 6 × 48	100	
6 × 48	80	77
6 × 60	95	92
7 × 48	105	98
7 × 60	133	126
8 × 48	134	127
8 × 60	165	158
10 × 28 × 28	492	
10 × 48	222	209
10 × 60	263	250
12 × 48	292	270
12 × 12 × 60	500	
12 × 60	382	360
14 × 60 extruded	525	493
14 × 60 molded	542	510
14 × 72 extruded	622	590
14 × 72 molded	642	610
16 × 60	647	590
16 × 72	781	724
16 × 16 × 80	1152	
16 × 16 × 84	1210	
16 × 16 × 110	1584	
17 × 60 extruded	743	686
17 × 60 molded	787	730
17 × 72 extruded	871	814
17 × 72 molded	949	892
18 × 72	1027	970
20 × 72 extruded	1223	1130
20 × 72 molded	1285	1192
20 × 20 × 80	1760	
20 × 20 × 84	1848	
20 × 84 extruded	1403	1310
20 × 84 molded	1515	1422
20 × 20 × 90	1980	
20 × 90 molded	1638	1545
20 × 20 × 100	2200	
20 × 20 × 120	2640	
20 × 180	3033	2940
24 × 72 extruded	1760	1630
24 × 72 molded	1890	1760
24 × 84 extruded	2130	2000
24 × 84 molded	2226	2096
24 × 110 extruded	2750	2620

Carbon electrodes for use in electric furnaces are made from either calcined petroleum coke or low-ash anthracite coal, or mixtures of these. Materials before molding into the electrode are carefully ground and screened and a composite aggregate made of such screen sizes as to give mechanically strong electrodes. The aggregate is mixed with a binder, molded under relatively high pressures, baked, and machined for continuous feeding of the electrodes.

Furnace manufacturers have eliminated electrode designs which necessitate shutdowns for electrode changes. Continuous feed of electrode has become standard practice. The manner of joining electrodes has been reduced to two methods. The first method employs electrodes turned down and threaded at one end, and drilled and tapped at the other end. These can be fitted together as male and female joints. The second method calls for both ends of the electrode to be drilled and tapped; a small threaded pin serves as the connecting medium.

Table 37. Amorphous Carbon—Standard Connecting-pin Data
(National Carbon Co., Inc.)

Size of pin (pitch diam. by length of thread), in.	Diam. of electrode used for, in.	Max. diam. top of threads, in.	Max. length over all, in.	Average weight, lb. oz.	Pins per package
3 × 6	6	3.28	6⅝	2 3	24
3 × 6	7	3.28	6⅝	2 3	24
4 × 8	8	4.41	9⅛	6 0	12
5 × 10	10	5.41	11⅛	11 0	6
6 × 12	12	6.41	13⅛	19 0	3
7 × 14	14	7.59	14	28 0	Crate as desired
8½ × 17	16	9.13	17	51 0	Crate as desired
8½ × 17	17	9.13	17	51 0	Crate as desired
8½ × 17	18	9.13	17	51 0	Crate as desired
10 × 20	20	10.63	20	85 0	Crate as desired
10⅞ × 24	24	11.69	24	120 0	Crate as desired
12 × 24	24	12.81	24	150 0	Crate as desired

Table 38. Graphite Electrodes *(Acheson Graphite Corp.)*

Size, diam., in.	Approximate weight per piece, lb.	Size rectangular, in.	Approximate weight per piece, lb.
Cylindrical			
1/16 × 12	0.002	1¼ × 12 × 12	2.10
⅛ × 12	0.008	½ × 4 × 24	2.94
¼ × 12	0.035	½ × 6 × 24	4.50
⅜ × 24	0.15	½ × 12 × 12	4.05
7/16 × 24	0.21	¾ × 3½ × 19½	3.13
½ × 24	0.26	¾ × 5 × 18	4.21
⅝ × 24	0.40	¾ × 12 × 12	6.75
¾ × 24	0.60	⅞ × 2 × 21¼	2.20
⅞ × 24	0.82	1 × 4 × 30	7.50
1 × 24	1.00	1 × 6 × 30	11.30
1 × 48	2.00	1 × 12 × 12	8.35
1⅛ × 24	1.4	1¼ × 3 × 36	7.88
1¼ × 24	1.75	1¼ × 5 × 30	11.50
1½ × 24	2.60	2 × 4 × 30	14.50
Square			
2 × 2 × 30	7.50	2 × 4 × 40	19.00
2 × 2 × 36	8.90	2 × 6 × 28½	20.00
4 × 4 × 13	12.00	2 × 7 × 30	25.75
4 × 4 × 17	15.75	2 × 7 × 40	32.48
4 × 4 × 40	37.00	3 × 6 × 30	33.25
6 × 6 × 40	84.50	4 × 8¾ × 15	31.45
6 × 6 × 48	102.00	2 × 8 × 48	44.00
8 × 8 × 48	179.00	3 × 9 × 48	75.20
		3 × 11 × 48	92.00
Tubular			
2¼ O.D. × 1 5/16 I.D. × 24	4.6	4 × 10 × 48	112.00

Table 39. Standard Graphite Electric Furnace Electrodes (*Acheson Graphite Corp.*)

Round electrode, diameter by length, in.	Approximate weight per piece, lb.	Sectional area		Approximate carrying capacity, amp.
		Sq. in.	Sq. cm.	
2 × 24 (50 × 610 mm.)	4.50	3.1416	20.266	600– 800
2½ × 30 (63 × 760 mm.)	8.40	4.9087	31.661	800– 1,200
3 × 40 (75 × 1000 mm.)	16.60	7.0686	45.595	1,200– 1,800
4 × 40 (100 × 1000 mm.)	30.00	12.566	81.025	1,800– 2,300
5⅛ × 40 (130 × 1000 mm.)	48.00	20.629	133.084	2,300– 3,000
6 × 48 or 60 (150 × 1220 or 1500 mm.)	75.00 or 94.00	28.274	182.369	3,000– 4,000
7 × 48 or 60 (175 × 1220 or 1500 mm.)	107.00 or 134.00	38.485	248.234	4,000– 5,500
8 × 48 or 60 (200 × 1220 or 1500 mm.)	140.00 or 175.00	50.265	324.227	5,500– 6,500
9 × 48 or 60 (225 × 1220 or 1500 mm.)	176.00 or 220.00	63.617	410.348	6,500– 8,500
10 × 48 or 60 (250 × 1220 or 1500 mm.)	230.00 or 285.00	78.640	506.662	8,500–10,000
12 × 48 or 60 (300 × 1220 or 1500 mm.)	320.00 or 400.00	113.100	729.608	10,000–15,000
14 × 60 (350 × 1500 mm.)	500.00	153.94	993.169	15,000–20,000
16 × 60 (400 × 1500 mm.)	700.00	201.06	1297.239	20,000–26,000
18 × 60 (450 × 1500 mm.)	900.00	254.47	1641.839	26,000–30,000

Table 40. Graphite Electrode Current Density (*Acheson Graphite Corp.*)

Amp. per sq. in. cross section

Diam. of electrode	Cross section sq. in.	Circumference, in.	Amperes																	
			1000	1500	2000	2500	3000	3500	4000	4500	5000	5500	6000	7000	8000	9000	10,000	12,000	15,000	18,000	20,000
2	3.14	6.28	318.3	447.3																	
2½	4.90	7.85	203.6	305.4	401.2																
3	7.07	9.50	141.5	212.2	283	353	424														
4	12.6	12.6	80	120	160	200	240	280	320												
5⅛	20.6	16.1	48	73	97	121	146	169	194	218	242										
6	28.3	18.85	35	53	70	88	106	124	141	159	177	194	212	247							
7	38.5	22.0	26	39	52	65	78	91	104	118	130	143	156	182	208	234					
8	50.3	25.13	20	30	40	50	60	70	80	90	100	110	120	140	160	180	200	240	300		
9	63.6	28.27	16	24	31	39	47	55	63	71	79	86	94	110	126	141	187	188	236	283	
10	78.6	31.42	13	19	25	32	38	44	51	57	64	70	76	89	102	114	127	153	191	230	254
12	113.1	37.70	9	13	18	22	26	31	35	40	44	48	53	62	70	79	88	106	133	159	177
14	153.9	43.98	7	10	13	16	20	23	26	29	33	36	39	46	52	59	65	78	97	117	130
17	227.0	53.41	4	7	9	11	13	15	18	20	22	24	26	31	35	40	44	53	66	80	88
24	452.4	75.40	2	3	4	6	7	8	9	10	11	12	13	15	18	20	22	26	33	40	44

Present-day dowel pins are turned out of stock just large enough to ensure good threads. Electrodes are now drilled and threaded in special self-centering lathes. Mechanical details on the dowel pins such as an end boss, champering the first thread, and reduction of thread tolerances have done away with electrode overhangs. Joint trouble has been reduced to a minimum.

Desire for a continuous feed of electrode as it is consumed has brought the development of the Söderberg electrode. In this method, electrode paste (the "green" mix) is tamped into a shell which is a vertical continuation of the electrode holder. As the electrode is consumed, the tamped mixture passes through zones of increasing temperature up to a heating ring where baking takes place. The electrode is continuous in that the form is kept filled up with electrode paste. It is jointless, baked, and renewed right in the furnace.

Average practice allows a current density on the electrodes of 20 to 40 amp. per sq. in. cross section.

Furnace electrodes made of Acheson graphite are employed when very high currents are necessary, and a balancing of all factors shows that graphite electrodes, although costing more per unit, would allow more economical operation.

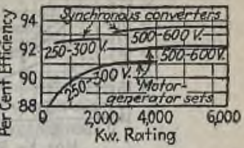


FIG. 44.—Efficiency of synchronous converters and motor-generator sets.

Converter efficiency includes losses of transformer and induction regulator or booster and 0.3 per cent alternating-current lead loss. Motor-generator set efficiency is based on 100 per cent power factor for motors. All losses are calculated or measured in accordance with the A.I.E.E. rules. The resulting efficiencies are close to the actual efficiencies.

Power for Electrochemical Processes

It is estimated that over 200,000 kw. capacity represents the annual purchase of d.c. machines for electrolytic service. Direct-current power can be obtained in several ways:

1. Purchased a.c. converted to d.c. by:
   a. Synchronous converters.
   b. Motor-generator sets.
   c. Rectifiers.
2. Generated power:
   a. Alternating-current generation and conversion as for purchased power.
   b. Direct-current generation:
      (1) Geared steam turbines.
      (2) Steam engines.
      (3) Diesel engines.

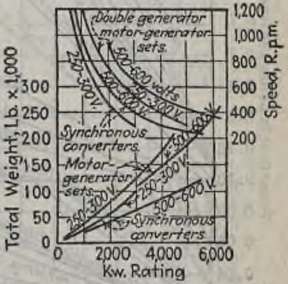


FIG. 45.—Speed of synchronous converters and motor-generator sets. (*Westinghouse Electric and Manufacturing Co.*)

The bulk of d.c. power is obtained from synchronous converters or motor-generator sets, while a few geared steam turbines are in operation. Recently the mercury-arc rectifier has found wide application.

At voltages above 600, the mercury-arc rectifier with no moving parts is favored over rotary converters or motor-generator sets for continuous electrochemical loads (for construction, etc., see Sec. 24).

The Cu-CuO or oxide type of rectifier has found application in low-voltage (6 to 15) practice such as electroplating, cleaning, etc.

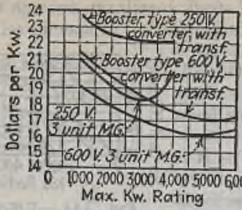


FIG. 46a.

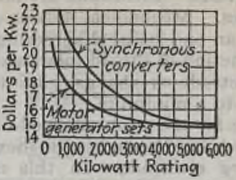


FIG. 46b.

FIG. 46a.—Prices of booster-type converters and motor generators for various ratings.
FIG. 46b.—Prices of synchronous converters and motor-generator sets. (*Westinghouse Electric and Manufacturing Co.*)

The converter prices include converter, transformer, and induction regulator or synchronous booster for a voltage range of plus and minus 5 to 10 per cent. The converters are rated at 40°C. rise and the transformers at 55°C. rise. Prices of the motor-generator sets include the complete set but not transformer prices. These machines are rated at 40°C. rise, with motors for 100 per cent power factor, three-phase, 60-cycles operation.

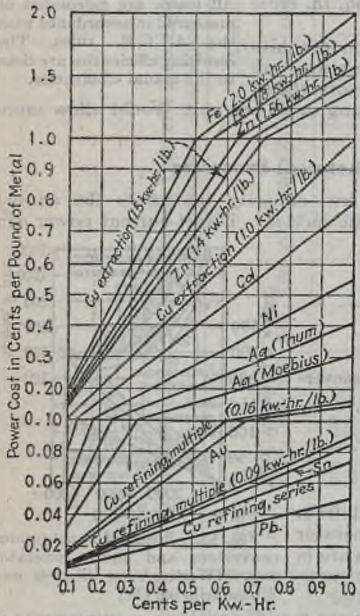


FIG. 47.—Power costs per pound of metal at varying power rates.

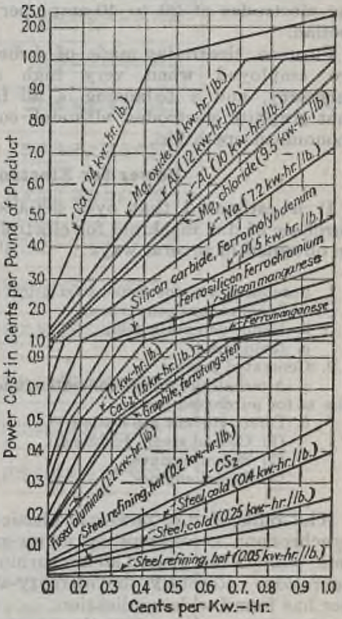


FIG. 48.—Power costs per pound of product from electric furnaces and fused electrolytes at varying power rates.

Table 41. Energy Consumption of Electrochemical Products
(*Mantell*)

Product	Kw.-hr. per lb.	Lb. per kw.-hr.	Voltage per tank, cell, or furnace
Alumina, fused	1 – 1.5	0.67– 1	100 –110
Aluminum	10 –12	0.08– 0.1	5.5 – 7
Cadmium	0.8	1.25	2.6
Calcium	22 –24	0.04	
Calcium carbide	1.3 – 1.4	0.71– 0.77	
Carbon bisulfide	0.4 – 0.5	2 – 2.5	60
Caustic, diaphragm cells	1.16 – 1.43	0.68– 0.86	3.4 – 4.2
Caustic, mercury cells	1.45	0.69	4.1 – 4.3
Chlorine, diaphragm cells	1.3 – 1.6	0.62– 0.76	3.4 – 4.2
Chlorine, mercury cells	1.6	0.62	4.1 – 4.3
Copper, electrorefining:			
Multiple system	0.09 – 0.16	6.3 –11	0.18– 0.4
Series system	0.074	13.5	16 – 18
Copper, electrowinning	1 – 1.5	0.67– 1	1.9 – 2.4
Ferrochromium, 70%	2 – 3	0.33– 0.5	90 –120
Ferromanganese, 80%	1.5 – 3	0.33– 0.67	90 –115
Ferromolybdenum, 50%	3 – 4	0.25– 0.33	50 –150
Ferrosilicon, 50%	2 – 3.5	0.28– 0.5	75 –150
Ferrotungsten, 70%	1.5 – 2	0.5 – 0.67	90 –120
Ferrovanadium	2 – 3.5	0.28– 0.5	150 –250
Gold	0.15	6.6	1.3 – 1.6
Graphite	1.5 – 2.0	0.67	80 –200
Iron	1.8 – 2.0	0.5 – 0.55	4 – 4.4
Lead	0.04 – 0.05	24	0.35– 0.6
Magnesium, chloride process	8 –13	0.08– 0.125	6 – 9
Magnesium, oxide process	14 –25	0.04– 0.07	9 – 16
Nickel	1.1	0.9	2.4 – 2.5
Phosphorus	4 – 5.5	0.18– 0.25	
Silico-manganese	2 – 3	0.33– 0.5	90 –120
Silicon carbide	3.2 – 3.85	0.26– 0.31	75 –230
Silver, Moebius	0.31	3.2	2.7
Silver, Thum	0.41	2.4	3 – 3.5
Sodium	7.1 – 7.3	0.13– 0.14	
Steel, cold charge	0.25 – 0.4	2.5 – 4	75 –150
Steel, hot charge	0.05 – 0.2	5 –20	
Tin	0.085	11.8	0.3 – 0.35
Zinc	1.4 – 1.56	0.64– 0.7	3.5 – 3.7

Table 42. Typical Power Costs

	Cts. per Kw.-hr.
New York Harbor (steam)	0.67–1.3
Norway	0.1 –1.5
Sweden	0.1 –1.5
Switzerland	0.3
French Alps	0.17–0.3
England	0.4 –0.5
Scotland	0.3 –0.5
Germany (brown coal)	0.38
Niagara Falls	0.3 up
Massena, N. Y.	0.36
Alabama (steam)	0.38–0.6
Tennessee (steam)	0.38–0.6
California (steam)	0.38–0.6
Ontario, Can.	0.15–0.4
Shawinigan, Que., Can.	0.15–0.2
Arvida, Que., Can.	0.1

Requirements of the electrolytic circuits are usually pretty definitely known. As a rule a voltage range of 10 per cent plus and minus will be ample to meet all operating requirements. This range is easily obtained on the ordinary self-excited generator.

A comparison of a.c. generation followed by conversion to direct current, and d.c. generation, shows that the latter has the advantages of lower first cost, slightly lower fuel cost, lower labor cost, with a possibility of lower maintenance cost, but the disadvantage of lack of flexibility when interconnected with other power systems, or in the conversion of the plant to other than electrolytic use, or when major changes are made in electrolytic circuits.

In general, synchronous converters show higher efficiencies than do motor-generator sets. This relation is shown in Fig. 44. Speeds of the different types of machines are given in Fig. 45 and prices in Fig. 46. In general, synchronous converters cost more than do motor-generator sets. For further details, reference should be made to Sec. 24, p. 2719.

Electric-furnace products and electric steel are made by the use of a.c. power. The furnaces involve the use ordinarily of multitap transformers with regulators, controls, and other incidentals. In general, the electric circuits show very high amperage values at moderate voltages. For steel furnaces transformers are usually special, while for other purposes, other than large sizes, they ordinarily involve no unusual features of design.

Power cost is an important factor in all electrolytic or electrothermal products. Figure 47 shows the power cost in cents per pound of metal at varying power costs in cents per kilowatt-hour for the electrolytic and hydrometallurgical industries, while Fig. 48 gives similar data for the electric-furnace products and the fused electrolytes. The curves are plotted from the power consumptions for these products given in Table 41.

Some typical power costs at various localities are given in Table 42. Unless otherwise stated the generation of power is in hydroelectric plants.

SECTION 26

ECONOMIC FACTORS IN CHEMICAL PLANT LOCATION

BY

Charles W. Cuno, Ph. D., Chemical Engineer and Director of Research, The Lehon Company. Formerly Consulting Engineer, St. Louis, Mo.; Vice President, Mississippi Valley Research Laboratories, Inc.; Consulting Chemical and Metallurgical Engineer, Industrial Bureau of Industrial Club of St. Louis. Member, American Chemical Society, American Institute of Chemical Engineers, American Institute of Mining and Metallurgical Engineers.

CONTENTS

ECONOMIC FACTORS IN CHEMICAL PLANT LOCATION

The plant engineer is often confronted with the problem of the location of branch plants or the relocation of existing units at more economic points of manufacture. It becomes necessary, therefore, that he know something of the factors governing plant location. In the discussion which follows, these factors, and the means of evaluating them, will be outlined.

The literature on the subject is meager. Weber ("Theory of the Location of Industry," trans. by Friedrich, University of Chicago Press), treating the subject purely from an academic standpoint, considers the three general factors: raw-material sources, in which he includes fuels and power; labor; and markets. Raw materials, including fuel, are evaluated according to price plus freight rates to points under consideration; labor according to comparative wage scales; and consuming markets according to magnitude plus freight rates.

By processes of triangulation and mathematical equilibriums, the logical point of production is theoretically computed. Weber's treatise has received a good deal of attention among economists. He has made no practical application of the theory and it is doubtful whether such application would be of value. Nevertheless, Weber's book probably is the classic among academic works on plant location, and therefore is worth reading.

Before discussing the factors of plant location, let us classify industries into (1) basic, (2) intermediate, and (3) tributary. Many writers have brought forth classifications of industries. No claim of originality is made for this one.

1. Basic Industries. Those industries are **basic** whose raw materials are for the most part unprocessed and whose products in the main are raw materials for other industries. Most basic industries, to succeed, must locate near cheap fuel or cheap power. Major raw materials must be obtainable advantageously as to quality and price, or at least on a competitive basis with the same industries in other localities.

2. Intermediate Industries in Various Stages. Those industries are **intermediate** whose major raw materials have already been processed and which are dependent to a degree upon procuring these processed materials from local producers, or at least in a competitive market from outside sources. An intermediate industry, to succeed, must have favorable connections with local producers or purchase in a highly competitive market and be able to sell to advantage in at least a portion of the national market.

3. Tributary Industries. Those industries are **tributary** whose raw materials may be either primary or secondary, but whose sources are not advantageous as compared with outside competition. A tributary industry, to succeed, should have an advantageous market locally due to freight differentials.

By way of illustration, the Missouri iron industry began about 1840, utilizing the ore bodies at Iron Mountain. It was a basic industry and flourished until the failure of the then known ore bodies, and until cheaper pig iron from competitive points entered the St. Louis market. In the meantime, intermediate industries in iron and steel fabrication were built up in the district. These intermediate industries continued to flourish even after the failure of their basic industry, owing to the highly competitive nature of the pig-iron market in the district. After the war, pig iron began again to be manufactured due to the Roberts process for coking Illinois coal. But pig-iron manufacture in the St. Louis district at this time should not be classed as a basic industry but as a *tributary* industry in that its sources of raw material are not advantageous as compared with outside competition. It succeeds because it has an advantageous market locally due to favorable freight rates, and a limited favorable outside area due to freight differentials. The situation is now again changing and the industry is becoming basic for the St. Louis district for two reasons; (1) the rediscovery of ore bodies in the Iron Mountain region, and (2) the establishment of advantageous barge service for Minnesota iron ore from St. Paul to St. Louis *via* the Mississippi River.

Again the district is lacking in salt deposits, but salt is sold on a highly competitive basis from Kansas, Michigan, Ohio, and Louisiana. To establish a basic industry in caustic and soda ash manufacture would be hardly feasible, but to establish it as a tributary industry is entirely feasible; in fact it has been done successfully.

Complementary Industries. In addition to the foregoing broad industrial classification, there are the **complementary** industries. An industry complements another when it serves that industry as a source of raw material, in the form of semifinished products. Examples of these are the coke industry which serves the pig-iron industry, and the sulfuric acid manufacture which serves the fertilizer industry.

Economic Factors of Plant Location. The economic factors of chemical-plant location can be grouped immediately into those concerned with efficient production, and those concerned with economic distribution.

The chief factors of production are five:

1. Raw Materials or Semifinished Products: Their quality, reserve, proximity to plant; sources of competitive material.
2. Labor: Supply and cost in kind, nationality, quantity, diversity, intelligence, wage scales, efficiency.
3. Power: Hydroelectric, public service, alternate sources.
4. Fuel: Kinds, thermal efficiency, reserve, alternate sources.
5. Water: Sources, mineral analysis, bacterial content, turbidity, quantity, seasonal temperatures, costs.

The chief factors of distribution are:

1. Transportation Facilities: Railroads, steamship lines, barge lines, terminals, wharves.
2. Freight Rates: Competitive points, differentials, favorable territory.
3. Markets: Local area, favorable area, competitive area, national area.
4. Competitive, Feeder, and Consumer Industries.

The factors that affect both production and distribution may be listed as:

1. Climate: Seasonal range, precipitation, humidity, wind, etc.
2. Taxes and Corporation Fees.
3. Municipal Restrictions: Nuisance laws relating to fumes, waste disposal.

Raw Material. Contrary to common belief, local sources of raw material may not be the chief attraction of a given locality. To this statement there are certain apparent exceptions, but usually some other factor of plant location is also a modifying factor.

In general, raw-material sources may be local, favorable or competitive and are favorable to a given plant location because of quality, quantity, reserve, and proximity. They become less valuable when similar conditions exist for sources of the same material in locations that are likely to become competitive points.

Raw materials are the most cheaply transported commodities and usually can be shipped much longer distances than finished products, because, in principle, American freight tariffs are based on bulk value, or, in other words, "what the traffic will bear." But this does not tell the whole story. Thus sulfur (low freight rate), to manufacture sulfuric acid, is transported long distances while sulfuric acid (higher freight rate) must be sold in a limited area for the added reason that 32 tons sulfur will make about 100 tons sulfuric acid. On the other hand, 1½ tons salt are needed to produce 1 ton caustic soda, while the freight rates on salt and caustic are approximately the same.

Labor. The supply and cost of labor involve such items as kind, nationality, quantity, diversity, intelligence, existing wage scales, and efficiency. In general, wages are lower in the country than in large cities. At present, wages are lower in the Southern states than in the North Atlantic and Mississippi Valley states. On the other hand, in quantity, diversity, intelligence, and efficiency, the labor in the cities will be found better than in the country. Labor of certain nationalities is more dependable, intelligent, and efficient than that of others.

In the chemical industry, labor, its supply and cost, hardly ever is a major factor of plant location, and with modern equipment and labor-saving devices it is doubtful whether it should be a major reason for the location of any industry. In the chemical industry, proportionally few common laborers are employed. Often the pay roll for research and technical men exceeds that for common labor. The pay for research and technical services is high, but this type of service can be obtained from any common source and its consideration should have no controlling effect on plant location.

Fuel. The third factor in plant location is fuel, its kind, thermal efficiency, reserve, etc. The kinds of fuel are coal, coke, oil, and gas. Certain of these are more flexible than others. When they are in competition with each other, lower prices with higher thermal efficiency are the usual result.

The tendency in chemical manufacture is to the use of the more flexible fuels, gas and oil. Often the advantage of precise thermal control offered by these overbalances the extra cost over coal. As with electric power, alternate sources of fuel are criteria in the selection of plant sites.

Power. Power may usually be classed as hydroelectric or steam-generated power. The plant site with cheap hydroelectric power has an appealing talking point for industrial development, but the fact is, there is little very cheap hydroelectric power, *i.e.*, in locations otherwise attractive. Steam-power plants are fast equaling or bettering all but the most exceptional hydroelectric performance, and Diesel engine installations, when fuel oil sells at less than 3½ cts. per gal., are furnishing power at less than 6 mills per kw.-hr.

For large users of power, alternate sources are too often neglected. It is all very well to tie up to a single source of hydroelectric power in the hope that for the life of the industry no failures may occur or no rise in rates prevail. There is, nevertheless, an historic background which cannot be overlooked. First, that hydroelectric power rates have been slowly increasing; second, that the costs of new installations are becoming more and more prohibitive. With this should be coupled the fact that the cost of powdered-coal generated power has been steadily decreasing in the past two decades. We find, however, that chemical industries, such as those that manufacture alkalies, carborundum, graphite, abrasives, or electrolytic metals, tend to centralize at sources of cheap hydroelectric power. Very few hydroelectric developments in the industrial world compare with Niagara Falls, which is therefore not to be regarded as a typical case.

Water. Water is an important factor in determining plant location. To more than nine-tenths of chemical industries water for evaporation and water for cooling purposes are very essential. River waters are usually lower in mineral analysis but higher in bacterial content and turbidity than well waters. Well waters are usually high in carbonate and bicarbonate. In many localities well waters are high in sulfate and often chalybeatic in character. They are ideal waters for cooling purposes as they usually come from the ground at a uniform and low temperature both winter and summer. Alternate sources are often important; a purified river water for boiler and steam purposes, and a well water for cooling.

We have discussed thus far the five principal factors concerned with efficient production. For specific cases, these five factors may be narrowed down to two or three. For the electrochemical plant the cost and dependability of power may be the paramount factor in determining plant location. If any fuel is used, it will probably be of minor consideration. Labor should be of minor importance. Hydroelectric stations usually also mean a good source of water. The major factors, therefore, narrow down to two; the cost of power and the cost and sources of raw material for such an installation.

In the location of a cement plant, suitable shales and limestone, coupled with cheap fuel, are perhaps the major factors. With comparative sources of shale and limestone, the decision between the two points of manufacture may narrow down to the types of fuel available; thus, natural gas at a low price, or fuel oil, can be used to better advantage in the cement kiln than powdered coal, and the availability of one or the other may be the deciding factor in the location of such a plant.

Having located our plant in respect to the five factors of production, our next task is to determine its efficiency from the standpoint of distribution. Successful merchandising does not end until the product is in the hands of the ultimate consumer.

Importance of Transportation Facilities. Such transportation facilities as railroads, steamship lines, barge lines, terminals, and wharves are the ways through which production flows to the consuming public. Where a city has more than one railroad, or better still, railroads and barge or steamship lines, the freight competition occasioned by these several transportation facilities makes for lower freight rates, higher efficiency, and better service.

The territory in which favorable freight rates apply, as compared to other competitive points, is an important consideration in plant location. The larger the population reached in this territory, the more important such a district becomes as an industrial center.

Markets are defined as local, favorable, competitive, and national. The local area is the population served by reason of plant location. The favorable area is that which an industry can serve by reason of lower cost production and freight rates. The competitive area is where the industry must compete on equal, or nearly equal, terms with like industries in other localities. In the national area the industry must be able to overcome freight-rate differentials completely as limiting sales factors.

Three factors, climate, taxes and corporation fees, and municipal restrictions, affect both production and distribution. The season temperature range, precipitation, humidity, wind, etc., which go to make the climate of a certain district, are factors that often enter directly into manufacture. Taxes and corporation fees are factors that increase or decrease overhead. High tax rates and high corporation fees often put industries out of certain states. Municipal restrictions are important factors in proper plant location. Many times industries find to their sorrow, after the expenditure of millions of dollars, that production is handicapped by the fact that certain municipal restrictions have not been considered. These restrictions may prohibit the use of certain materials or make necessary an expensive outlay to take care of certain obnoxious waste products, as for example stack fumes and effluents into streams.

Industrial Backgrounds. There still remain three important factors of plant location: competitive, feeder, and consumer industries. For location in a proposed district it is important, on the one hand, to know what competition the industry will have in its own territory. It is equally important to know the feeder and consumer industries at hand, the first to supply the needed raw materials and the second to supply an outlet for its finished product.

In an analysis of industries we find that the order of importance of these factors enumerated is often changed, so that in one case one factor may predominate; in another, another. It is to be noted, however, that fuel, power, and raw materials are being transported longer distances to plants satisfactorily located for distribution; that distribution, favorable freight rates, magnitude of markets, and labor conditions are important factors in the location of all industries; that basic industries, generally speaking, seek cheap fuel or power; that intermediate industries seek favorable location to sources of semifinished products; and that the factors of competitive, feeder, and consumer industries are often the most important factors in plant location.

Chemical Industries. In applying these several factors to the location of chemical industry, we find, as has been stated before, that the factors of competitive, feeder, and consumer industries bulk large among the others. The reason for this is that the products of chemical industries usually take much higher freight rates than the raw materials used in their manufacture.

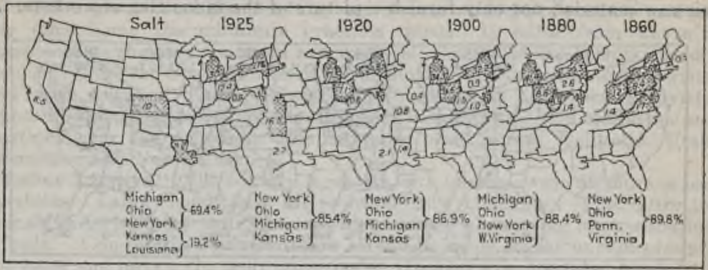
For intermediate chemical industries, for the same reason, it is almost imperative to locate near local sources of semifinished products. It would seem, therefore, that for basic chemical industries, and also for those intermediate industries for which chemicals are major raw materials, the most logical plant location would be at a point of exceptional distribution facilities, *i.e.*, where the local market is large and the favorable and competitive markets show a goodly number of feeder and consumer industries, with not too many competitors doing business in the same territory.

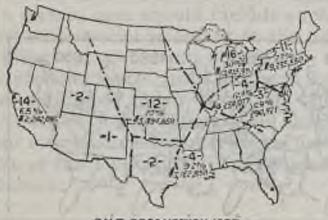
In basic chemical industries, the tendency is to ship raw materials farther and finished products less far. In intermediate industries, the tendency is to group around basic manufacture, when such basic manufacture is strategically located for distribution.

PRACTICAL AIDS FOR THE ENGINEER IN PLANT-LOCATION PROBLEMS

While each problem is a specific one there are, nevertheless, certain general directions that can be applied to every instance.

Charts and Diagrams. Valuable charts and diagrams can be prepared using an outline of the United States as a base map. They serve to visualize raw-material sources, location of competitive industries, competitive freight-



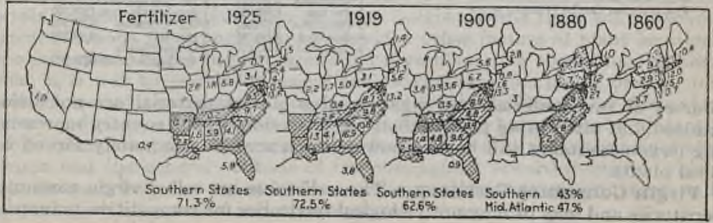


SALT PRODUCTION 1925
170 PLANTS, 7 STATES PRODUCED 96.3% VALUE $34,253,056
Michigan, Ohio & New York 69.4% Kansas & Louisiana 19.2%

FREIGHT TERRITORIES
ALKALI MANUFACTURE
☐ Favorable Territory ▨ Competitive Territory

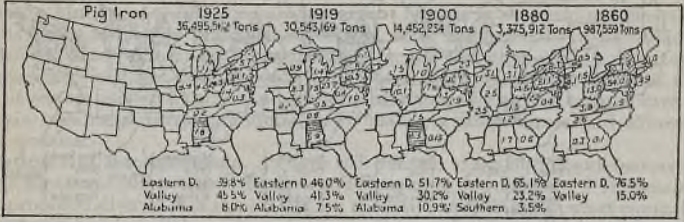
HOW ONE SOURCE OF IRON ORE DOMINATES THE
PIG IRON INDUSTRY BECAUSE OF LAKE TRANSPORTATION

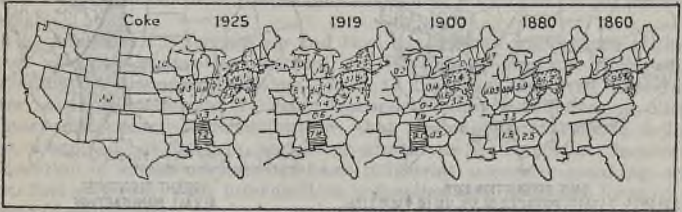
DISTRIBUTION OF
SULFURIC ACID PLANTS 1925
States for Which no Percentage is Shown Produced in all 14.4%



rate points, consumer territories, etc. A number of suggested bases for charts are herewith outlined.

Raw-material Surveys. Surveys of raw-material resources are well charted and, when combined with the location of the principal plants utilizing this raw material, not only furnish a picture of the industries as a whole, its sources of raw material, and what sources of raw material are not being utilized, but often bring to light districts or sections of the country representing large consuming and buying power which are not adequately served by local plants.





COAL TAR PRODUCTS 1925

Crudes	50	Plants
Intermediates	64	"
Dyes and other Coal Tar Products	74	"

SUGAR CANE REFINERIES

DISTRICT	POPULATION	DISTRICT	POPULATION
1-Boston Local	8,182,000	6-Pan-N.Y.-Phila.-Balt.	7,535,200
2-New York	8,406,000	7-Par-Balt.-Savannah	1,103,000
3-Philadelphia	6,300,000	8-Gulf Refineries	11,671,000
4-Baltimore	9,140,000	9-Pacific	9,191,000
5-Savannah	6,513,000	10-Competitive with Beet Sugar Refineries	48,381,000

Virgin Consumer Territory. These districts are called virgin consumer territories and may or may not be logical territories for competitive industries or branch plants, depending upon whether they answer favorably to the major factors of plant location.

When Consumer Territory Is Governing Factor. It has been said that in sulfuric acid manufacture it is necessary to establish only one fact, *i.e.:* Can a given consumer territory use 50 tons or more of sulfuric acid per day (this being the minimum practical unit)? If sulfur from Texas is considered as the only source of raw material, this statement is undoubtedly true; but there are other sources of sulfur dioxide, such as zinc works, pyrite, and other sulfides. Examples are the Atlantic ports using foreign pyrite, plants in and around St. Louis using sulfur dioxide from zinc refineries, and locations near the copper or zinc refineries of Montana, Colorado, Utah, Tennessee, and New Jersey.

Labor Surveys. The country is replete with labor surveys all more or less worthless. Labor conditions are continuously changing. The entry of a single large industry into a certain district may change the labor situation entirely. Comparative labor surveys should be made on the unit-of-production basis and not on the daily wage scale. Inquiry into the labor conditions of a certain area should include such items as labor supply; housing; nationality with special reference to the percentage of foreign born, the percentage of illiteracy, and negro population; sex; type, *i.e.*, whether the foreign born are predominantly from northern Europe or from the Mediterranean countries; and labor turnover.

As an example, a survey of the unit cost of production under identical labor conditions in branch plants manufacturing the same article was made in some of the Mississippi Valley states. The following are some of the figures obtained:

Item	Omaha	Kansas City	St. Louis	Indianapolis	Memphis	Chicago
1	$1.87	$ 1.71	$ 1.76	$2.05	$1.62	
2	.....	585.20	607.40	.....	.....	$644.20
3	.....	1.61	1.48	.....	.....	1.25
4	.....	21.13	22.85	.....	.....	22.85
5	.....	....	0.515	0.63		
6	.....	....	15.23	.....	.....	19.39

Freight Divider Lines. By plotting freight rates for a given raw material from points of origin to competing plants and, on the other hand, freight rates on finished products to competitive areas, we can determine lines where these rates are equal, or nearly equal, from the points of origin. These are called freight divider lines. In these lines lie the competitive points at which freight rates are equal. In shipments beyond these lines the industry has to absorb certain freight differentials, also on these freight divider lines will be found logical locations for branch plants, provided other factors of plant location, especially those of production, are found favorable. On these lines, or near them, it is well to search for exceptional advantages, for power or fuel, for local sources of raw material, for favorable labor conditions, for favorable water conditions, and for such special conditions as barge lines or shipping competition, low taxes and corporation fees. It is also well to consider the climate and the general attitude of the community toward industry, which includes such items as nuisance laws, police and fire protection, labor and compensation laws, union and open-shop conditions, etc.

Ocean, Lake, and River Transportation. The importance of ocean, lake, and river transportation is often not realized. For example, 56,000,000 tons of iron ore alone are transported by ship through the Great Lakes to Chicago, Cleveland, Detroit, and Pittsburgh. On a 100-mile stretch of the Monongahela and Allegheny rivers more than 33,000,000 tons of barge freight serve the iron and steel industry of the Pittsburgh district. Millions of tons of freight find their way by barge between St. Louis and New Orleans.

SUMMARY

It has been pointed out:

That industries migrate to strategic points where the economic factors that govern plant location indicate the convergence of minimum cost of production with minimum cost of distribution.

That this migration proceeds in spite of overdevelopment in other localities.

That it becomes necessary for the efficient plant manager to determine these strategic points by systematic survey in order to establish branch plants before competition shall have entered the field.

Twelve factors governing plant location have been described. The five concerned with production are sources of raw material; fuel; power; water; and labor. The four factors of distribution are transportation facilities; freight rates; markets; and consumer, feeder, and competitive industries. Those affecting both production and distribution are climate; taxes and corporation fees; and state and municipal restrictions.

For our purpose we have classified industries into basic, intermediate, and tributary. Other classifications have been mentioned and defined.

An outline survey has been presented for the benefit of the plant engineer who is called upon to locate a branch plant or relocate an existing one. It embodies such suggestions as the charting of raw-material sources, of competitive, feeder, and consumer industries, of freight divider lines, and of virgin consumer territory; surveys that include labor cost, alternate sources of fuel and power, water resources, climate, etc.; an investigation of the modification of freight territories by the utilization of ocean, lake, and river transportation; a rigid examination of the tax situation, corporation fees, state, county, and municipal restrictions; and, finally, such minor factors as police and fire protection, labor and compensation laws, unionized and open-shop districts, housing, amusement, and transportation for laborers, and all such minor details that if not taken care of are constant sources of petty irritation.

SECTION 27

ACCOUNTING AND COST FINDING

BY

George A. Prochazka, Jr., E. M., Consulting Engineer; formerly Chemical Economist, Ammonia Department, E. I. du Pont de Nemours & Co.; formerly Manager, Central Dyestuff & Chemical Company; Member, American Institute Cheminal Engineers, American Institute Mining and Metallurgical Engineers; Author, "Accounting and Cost Finding for the Chemical Industries."

CONTENTS

ACCOUNTING AND COST FINDING

REFERENCES: From the many books published on accounting and cost finding, the following have been selected as most fully meeting the requirements of the chemical engineer: "Accountants' Handbook," Ronald Press, New York, 1923; covering the entire field of accounting and cost finding from the viewpoint of the professional accountant. "Accountants' Encyclopedia," McGraw-Hill, New York, 1928; a four-volume reference work on systems, principles, auditing, and budgeting. Prochazka, "Accounting and Cost Finding for the Chemical Industries," McGraw-Hill, New York, 1928; giving details as to installing accounting and cost systems. Charlton, "American Mine Accounting," McGraw-Hill, New York, 1913; especially useful in finding forms to cover particular phases of cost and accounting work. Gilman, "Analyzing Financial Statements," Ronald Press, New York, 1925; an excellent book on analysis of financial statements. Esquerre, "Applied Theory of Accounts," Ronald Press, New York, 1921; a very popular book on general accounting theory. Kilduff, "Auditing and Accounting Handbook," McGraw-Hill, New York, 1927; a good reference manual. "Management Handbook," Ronald Press, New York, 1924; a reference book for various management problems, production control, etc. Hine, "Modern Organization," Engineering Magazine Co., 1916; an excellent short discussion of theory and practice of management. Young, "Statistics Applied in Business," Ronald Press, New York, 1925; general statistical methods.

GENERAL ACCOUNTING

Bookkeeping and Cost Work. To insure accuracy, cost calculations must be checked against general books. If the same accounts are used for both purposes, the costs can be journalized and can be made an integral part of the general-accounting system. The cost calculations are then checked automatically when the books are closed.

Debits and Credits. Every business transaction has two phases, *i.e.*, when goods are bought, money is exchanged for raw material. This gives rise to the debits and credits of bookkeeping. Some of the more important items are given in Table 1, from which data others can be worked out by analogy. By custom, the debits are placed on the left-hand side and the credits on the right-hand side of a page.

Table 1. Debits and Credits of Bookkeeping

Transaction	Account to	
	Debit	Credit
Raw material bought on credit	Raw Material	Accounts Payable
Goods sold on credit	Accounts Receivable Cost of Goods Sold	Sales Finished Stock
Purchase invoice paid	Accounts Payable	Cash
Customer pays for goods	Cash	Accounts Receivable
Raw material used in manufactures	Work In Process	Raw Material
Goods finished in factory	Finished Stock	Work in Process

Journals. The two phases, debit and credit, are noted in a journal and from these original entries postings are made to a ledger account. In modern bookkeeping, only unusual events are recorded in this way. The ordinary transactions are originally entered in a specially ruled journal, each column representing an account, and only column totals are posted to the ledger accounts. To facilitate this procedure, separate books are used for different kinds of transactions. The more important sections are given in Table 2.

Table 2. Bookkeeping Journals

Name of Section	Original Record of
General Journal	Unusual transactions
Purchase Invoice Book	Expenditures for which credit is extended
Sales Book	Goods sold on credit
Pay-roll Book	Wage payments
Cash Book	Money received and disbursed
Credit Book	Credits given or received

Ledgers. In the original scheme of bookkeeping all accounts were handled in the ledger. Today the main or general-ledger records essentially totals, whose details are to be found in one of the journals or in a subsidiary ledger such as the customers' ledger, the purchase ledger, and the various cost ledgers.

Controlling Accounts. The work in the subsidiary ledgers is always controlled in the general ledger by an account which independently summarizes the details and shows them to be in balance with the rest of the work. Such accounts are called **controlling accounts.**

Reserve Accounts. The true nature of reserve accounts is rarely understood. A portion of the earnings is set aside to provide for such things as:

1. Depreciation.
2. Obsolescence.
3. Bad debts.
4. Accrued taxes.
5. Sinking funds.
6. Contingencies (patent litigation, etc.).

Although money has been set aside on the books for the above-mentioned purposes, the money itself is kept with all the other money in one general fund which is used to run the business. Such money may become permanently tied up in a fixed investment. A reserve account, therefore, merely shows that a profit or other value has been reduced on the books—that the money has been set aside in a special bank account is not to be inferred. Reserve accounts appear as credit items on the books. In financial statements, the above-mentioned Items 1 to 3 are frequently set up as a deduction from the corresponding asset accounts.

Mixed Accounts. When dissimilar debit and credit entries, such as interest paid and interest received, are carried in one account, the account is said to be mixed. An expense account is said to be mixed when it carries an inventory balance, *e.g.*, unexpired insurance in an insurance account. Mixed accounts should be avoided because they are confusing.

Classification of Accounts. For practical bookkeeping purposes there are four kinds of accounts, which normally exist as follows:

Debits	Credits
1. Expense	2. Income
3. Asset	4. Liability

Selecting Ledger Accounts. The thoroughness with which information is obtained in accounting, or in cost work, depends largely on the care with which the general-ledger accounts have been selected. When an accounting classification is attempted many of the groups will be found disproportionate in size. Arbitrary decisions are often necessary to make a more effective grouping. These decisions must be indexed, as must also obscure items which are dealt with only occasionally. Such an index is helpful in charging bills to the proper accounts.

Trial Balances. After all entries for a period have been made, the mathematical accuracy of the work can be checked by taking a balance, *i.e.*,

Table 3. Bal

COMPARATIVE BALANCE SHEET AS AT DEC

ASSETS

CURRENT ASSETS:	1929	1928	Increase *Decrease*
Cash..............................	$ 2,269,917.62	$ 1,374,182.53	$ 895,735.09
Notes and trade acceptances receivable	238,924.74	46,099.97	192,824.77
Accounts receivable (less reserve)......	837,729.16	954,466.82	*116,737.66*
Inventories at cost....................	1,163,820.15	1,217,436.84	*53,616.69*
	$ 4,510,391.67	$ 3,592,186.16	$ 918,205.51
BALANCES RECEIVABLE FROM EMPLOYEES UNDER STOCK-SUBSCRIPTION CONTRACTS..........................	$ 63,518.00	$ 51,500.00	$ 12,018.00
INVESTMENTS........................	$ 1,174,859.78	$ 617,616.41	$ 557,243.37
PROPERTY ACCOUNT:			
Land, buildings, and equipment........	$21,887,388.21	$19,067,504.90	$2,819,883.31
Less: Reserve provided for depreciation and obsolescence of plant and equipment and depletion of brine wells....	$ 6,539,027.49	$ 6,105,733.59	$ 433,293.90
	$15,348,360.72	$12,961,771.31	$2,386,589.41
DEVELOPMENT EXPENSES FOR NEW PRODUCTS AND PROCESSES..........	$ 80,870.11	$ 436,705.25	$ *355,835.14*
DEFERRED CHARGES..................	$ 355,626.12	$ 262,864.02	$ 92,762.10
	$21,533,626.40	$17,922,643.15	$3,610,983.25

equating the open debits against the open credits. If the work has been properly done, the total open debits in the accounts will equal the total open credits in the remaining accounts.

Adjusting Entries. At the end of the year the sum of the 12 monthly cost entries, for depreciation, taxes, insurance, etc., must be reconciled with the exact amount of the annual charges. Special entries for bad debts and other contingencies must also be made before the books can be closed for an annual statement.

ance Sheet*

EMBER 31, 1929, AND DECEMBER 31, 1928

LIABILITIES

		1929	1928	Increase *Decrease*
CURRENT LIABILITIES:				
Accounts payable		$ 446,630.32	$ 429,244.78	$ 17,385.54
Taxes, insurance, etc., accrued		43,238.01	33,469.25	9,768.76
Federal taxes		287,838.00	285,479.89	2,358.11
Dividends payable		368,489.50	263,935.50	104,554.00
		$ 1,146,195.83	$ 1,012,129.42	$ 134,066.41
DEFERRED LIABILITIES:				
Containers charged to customers (returnable)		$ 221,858.84	$ 203,568.84	$ 18,290.00
RESERVES:				
Miscellaneous operating reserves		$ 134,019.50	$ 103,890.54	$ 30,128.96
CAPITAL STOCK:				
Preferred—authorized $3,500,000.00				
Issued (less amount held in sinking fund and treasury)		$ 2,475,000.00	$ 2,475,000.00	
Common—authorized 1,000,000 shares				
Issued	637,038 shares			
Allotted to employees under subscription contracts	13,318 shares			
Scrip outstanding	80 shares			
	650,436 shares	$10,041,424.50	$ 7,131,049.00	$2,910,375.50
		$12,516,424.50	$ 9,606,049.00	$2,910,375.50
SURPLUS:				
Free		$ 6,848,127.73	$ 6,380,005.35	$ 468,122.38
Appropriated for retirement of preferred stock		667,000.00	617,000.00	50,000.00
		$ 7,515,127.73	$ 6,997,005.35	$ 518,122.38
		$21,533,626.40	$17,922,643.15	$3,610,983.25

---

* Courtesy of E. M. Allen, President, The Mathieson Alkali Works (Inc.).

Financial Statements. Two statements are required to explain the affairs of a business. The balance sheet shows the values existing as of a particular moment. The profit and loss statement shows what brought about the present financial condition. Typical statements are shown in Tables 3 and 4.

Table 4. Profit and Loss Statement*

COMPARATIVE PROFIT AND LOSS ACCOUNT FOR THE YEARS ENDED DECEMBER 31, 1929, AND DECEMBER 31, 1928

	1929	1928	Increase *Decrease*
EARNINGS:			
After deducting manufacturing, selling, and general administrative expenses:			
Works operations................	$3,463,275.41	$3,211,726.37	$ 251,549.04
Other operations.................	117,654.84	107,521.96	10,132.88
Total earnings from operations.........	$3,580,930.25	$3,319,248.33	$ 261,681.92
Provision for depreciation, obsolescence, and depletion......................	1,026,720.51	912,752.39	113,968.12
Net earnings from operations........	$2,554,209.74	$2,406,495.94	$ 147,713.80
Income credits (net).................	57,904.70	*29,614.25*	87,518.95
Total income......................	$2,612,114.44	$2,376,881.69	$ 235,232.75
Federal income tax..................	287,838.00	285,479.89	2,358.11
NET INCOME FOR YEAR transferred to surplus account......................	$2,324,276.44	$2,091,401.80	$ 232,874.64

SURPLUS ACCOUNT

Free surplus at January 1, 1929..			$6,380,005.35
Add:			
Profit for year 1929..			2,324,276.44
			$8,704,281.79
Deduct:			
Appropriated for sinking fund for retirement of preferred stock....................................		$ 50,000.00	
Miscellaneous adjustments..........................		56,640.06	
Dividends paid:			
On preferred stock..............................	$ 173,250.00		
On common stock..............................	1,135,018.00	1,308,268.00	
Three shares of common stock for each share of common stock to holders of record April 15, 1929		441,246.00	1,856,154.06
Free surplus at December 31, 1929....................			$6,848,127.73

---

* Courtesy of E. M. Allen, President, The Mathieson Alkali Works (Inc.).

Table 5. Disposition and Source of Income*

The following summary shows the disposition of the aggregate amount of the income for the year and proceeds from the sale of common stock, as reflected by the balance sheet:

Net income from profit and loss statement		$2,324,276.44
Surplus adjustments		*56,640.06*†
Appropriated from income for depreciation, obsolescence, and depletion	$1,026,720.51	
Deduct depreciation and obsolescence on property dismantled and sold	593,426.61	433,293.90
Proceeds from sale of common stock		2,457,111.50
		$5,158,041.78
Disposition:		
Increase in property investment		$2,819,883.31
Increase in investments		557,243.37
Increase in net current assets		784,139.10
Dividends paid		1,308,268.00
Other miscellaneous changes in assets and liabilities (net)		*311,492.00*†
		$5,158,041.78

* Courtesy of E. M. Allen, President, The Mathieson Alkali Works (*Inc.*).
† Italics represent deductions.

ANALYZING FINANCIAL STATEMENTS

Purpose and Viewpoint. Financial statements do not reveal all the information that is contained in them and consequently they must be analyzed. The usual purpose is to develop the trend of a business for a series of statements and to uncover symptoms of the five common business ailments, namely:

Insufficient	Over-investment in
1. Capital.	1. Receivables.
2. Profits.	2. Inventories.
	3. Plant, etc.

These analytical investigations are made from a number of different viewpoints, such as management, credit extension, investment, etc. The last two, being made by outsiders, are usually conducted with meager information and, therefore, must be more searching. The method of analysis depends largely on the data available and the purpose of the investigation.

Principles. The analysis of financial statements is a matter of looking for exceptions. Every possible avenue of approach should be used to find conditions which suggest the need for a more extended investigation. Favorable findings must always be interpreted in a negative sense; *i.e.*, nothing bad has been uncovered. Extreme pessimism is justified because it is the best safeguard against having a few favorable indications lead to a hasty conclusion.

Balance-sheet Terms. Usage of financial terms is rather loose. Figure 1 shows some of the usual balance-sheet terms. The term liquid assets is often used to signify current assets. Working capital is the excess of current assets over current liabilities.

- Assets
  - Current assets
    - Quick assets
      - Cash accounts receivable
    - Working assets
      - Inventories
      - Prepaid charges
  - Fixed assets
    - Land
    - Buildings
    - Plant
      - Machinery
      - Equipment
  - Other assets
    - Deferred charges
    - Franchises
    - Patents
    - Good Will
    - Investments
- Liabilities
  - Current liabilities (obligations due within one year)
    - Accounts payable
    - Notes payable
    - Accrued liabilities
  - Long-term liabilities (obligations more than 1 year to run) also called "fixed, funded, or permanent liabilities"
    - Bonds
    - Mortgages
  - Net worth, owner's equity
    - Capital stock
      - Preferred
      - Common
    - Surplus
      - Appropriated (capital)
      - Undivided profits (earned)

Fig. 1.—Balance-sheet terms.

Balance-sheet Analysis. For an effective analysis a series of balance sheets are required. Considerable information, however, can be obtained from a single statement. Before attempting a mathematical analysis, the sheet should be scrutinized for such things as:

1. Solvency—excess of assets over liabilities.
2. Cash on hand—if relatively small.
3. Receivables—if large.
4. Inventories—if large.
5. Effect of deducting intangibles from surplus.
6. Depreciation—amount taken.
7. Tampering with assets through the use of appraisals or by marking up investments.
8. Improper combinations of balance-sheet items.

Percentage Method. A broad general survey of a balance sheet is obtainable by calculating each item as a per cent of the total. This is equivalent to finding out whether a man is tall or short, fat or thin. To average a number of statements and then find the relative proportions of the items is an important preliminary step for the trend method, discussed later.

Ratio Method. The analysis of a balance sheet by weighing one item in terms of another is adapted to the study of a single statement, and it is an additional safeguard to be used with the last statement of a series. The purpose of this method is to see whether the statement is properly proportioned from a financial viewpoint. There are nine ratios that are commonly used, but each one is not applicable to every type of business. Of the nine given below, the first four examine the resources used to establish a definite sale volume; the last five examine more particularly the credit situation. In the absence of knowledge as to the sales volume, estimates of the probable volume should be made.

1. Fixed assets per $100 of net sales.
2. Inventory per $100 of net sales. (This should also be worked out as the number of days of inventory on hand.)
3. Working capital per $100 of net sales.
4. Net worth per $100 of net sales.
5. Receivables per $100 of net sales. (This should also be worked out as the number of days credit extended.)
6. Fixed assets per $100 of net worth.
7. Liabilities per $100 of net worth.
8. Current ratio: Current assets divided by current liabilities. (This should be 2 or higher.)
9. Acid test: Quick assets divided by current liabilities. (This should be 1 or higher.)

Trend Method. The rate of increase or decrease of balance-sheet items over a period of years shows the direction in which a business is moving. To avoid studying the trend of inconsequential items a percentage analysis should be made first. After a proper grouping of items has been provided for, one year is taken as a reference basis. The grouped items are all given values of 100 and those of other years are reduced to an equivalent basis (by dividing the values of any one year by those of the reference year and multiplying the result by 100). This is the index-number method. Another way is to plot the actual quantities on semilogarithmic paper. The graphic method is simpler, but the placing of lines on a chart is somewhat difficult to control. After these data have been obtained it is a simple matter to see whether the business is moving ahead or falling behind by merely noting the trend of the figures.

Disposition and Source of Income. The true story of the activities of a management is best obtained by reviewing for a period of years the origin and disposition of all funds. These data are secured by calculating the increases and the decreases for all items on two consecutive balance sheets (see Table 3). The general balance-sheet equation, assets equal liabilities, can be rearranged so that the decreases of assets are added to the increases of liabilities to get the source of funds, and this must equal their disposition, the sum of the increases of assets and the decreases of liabilities (see Table 5).

Profit and Loss Analysis. The profit and loss account can be analyzed in a manner similar to that used for the balance sheet. A profit and loss analysis must also take up all debits and credits to the surplus account because extraordinary losses and capital profits are frequently charged direct to this account. The scope of the work depends on the data available and the purpose of the analysis. Some of the factors to be studied are:

1. Earnings as a per cent of net worth.
2. Earnings per share as a check on stock-market quotations.
3. Margin of safety—net income after paying bond interest divided by net income before paying bond interest.

4. Gross sales and returned goods.
5. Net sales.
6. Cost of goods sold.
7. Operating expenses.

Standard Ratios. Many people like to check statements by measuring the values in terms of standard ratios to which they have become accustomed. They believe that the inventories should bear a certain relationship to the investment in plant, the cash a definite relationship to the investment in plant, the cash a definite relationship to the inventories, etc. By these methods it is possible to get some general ideas as to the financial position, but such appraisals must not be given too much weight within close limits. It has been the experience of all investigators that conditions vary widely for every business, even when two firms are making the same products. Caution is necessary when such ratios are applied to cost estimates. If the machinery is of a large heavy type, erection, or maintenance labor will be a much smaller percentage of the total cost than when the units are small. The personal equation enters so largely into the management of a business that standard ratios cannot be used for other purposes than to make a few general approximations.

FIXED-PROPERTY ACCOUNTING

Fixed-property Charges. The carrying costs of fixed investments are usually a substantial item, often the largest single element. To understand the true nature of these costs, it is necessary to have a carefully devised system of fixed-property accounts.

Fixed-property Classifications. To furnish all the information required in business, the fixed-property costs must be available according to five separate classifications:

1. Major divisions for controlling accounts—four groups involved:
   a. Land.
   b. Buildings and structures.
   c. Machinery and equipment.
   d. Intangibles.
2. Construction elements: This is the original form in which fixed-property costs are obtained. These values should be arranged to meet the needs of the engineering department for future estimates.
3. Location of values: For fire insurance the value of fixed assets in each building must be known.
4. Departmental ownership: The plant values for each department are required for cost work.
5. Process values: When more than one process is performed in a department then the assets values must be subdivided to meet the cost requirements for each process.

Land. Under the caption land are placed the original acreage cost and all subsequent improvements thereto. Such things as pavements, wells, railroad sidings, etc., are logically included here because at a time of liquidation they cannot be separated from the land. Idle land should be kept apart from occupied land in the books of account, as should items which must be depreciated.

Buildings and Structures. The buildings and structures erected on a property are highly individual and present no accounting problem. They are usually numbered serially and are referred to by number in all the records.

Machinery. This covers the large fixed tools which are used by the production and the service departments. Little difficulty is experienced in developing an accounting method for these items.

Equipment. This covers the accessory devices, which are essential to the operation of machinery, and other auxiliary effects. Equipment accounting brings many problems. Distinctions between machinery and equipment must often be made on an arbitrary basis; usually equipment can advantageously be considered as an integral part of a machine.

Classes of Equipment. The more common classes of equipment are as follows:

1. Mechanical power-transmission equipment:
   - *a.* Shafting, hangers, collars, etc.
   - *b.* Pulleys.
   - *c.* Gears, speed-changing devices.
   - *d.* Belting.
2. Electrical equipment:
   - *a.* Wiring.
   - *b.* Panels, instruments.
   - *c.* Starting devices, circuit breakers, etc.
   - *d.* Motors.
   - *e.* Telephone lines, signaling devices.
3. Pumps:
   - *a.* Reciprocal.
   - *b.* Centrifugal.
4. Piping:
   - *a.* Steam.
   - *b.* Heating.
   - *c.* Air.
   - *d.* Vacuum.
   - *e.* Water.
   - *f.* Refrigeration.
   - *g.* Gas.
   - *h.* Underground.
   - *i.* Manufacturing.
5. Tools, instruments, etc.:
   - *a.* Thermometers.
   - *b.* Small tools.
   - *c.* Gages, etc.
6. Furniture and fixtures.
7. Miscellaneous effects.

Equipment Control. The size of the undertaking determines just how far it is feasible to go in setting up separate classes of equipment. The problem is one of not getting entangled with too many records and yet having enough data to distinguish between the merits of the various types of equipment. The following two cases illustrate the typical problems of equipment control.

Illustration 1. Piping maintenance expenses are not required in more detail than a class cost for a particular type of service. Costs for steam and kindred commodities must be figured at an average delivered value, because no particular department must be penalized for a more distant location. It is true that such a location costs more, but if such a location did not exist those nearer would cost more because the volume of production would be greatly reduced. The correct theoretical consideration is to figure the average cost of steam, water, etc., delivered at the consuming departments, and thus piping equipment need only be controlled by classes. Departmental piping maintenance should be charged to the processes or departments involved.

Illustration 2. Motors should be individually controlled because they are moved from one location to another. The usual practice is to remove a motor for repairs and to put a spare unit in its place, later making a spare out of the repaired motor. If a high depreciation rate is used, the repair costs can be charged against the Depreciation Account by offsetting with a Renewal Account. The exact procedure depends on the company accounting policy.

Fixed-property Codes. Many codes have been suggested for identifying supplies, materials, equipment, machinery, etc. Few, if any, ever break through their own intricacy sufficiently to be of great value to anybody beyond the few specialists who are in daily contact with them. For this reason, fixed property should be marked in a simple way. Using the machine names supplemented by a serial number to differentiate those of like kind is a good scheme. Nothing particularly complicated should be devised as it confuses the factory workers, and the code is not used enough to result in the economies achieved by a mnemonic classification of parts for machinery manufacture.

Intangibles. Patents, formulas, inventions, research work, good will, etc., are the usual intangibles. Essential data on these items should be reduced to writing and preserved in the company files. Writing out usually helps to clarify the concept.

Amortization. The extinguishment of a debt through the medium of a sinking fund is spoken of as amortization. The wiping out of a lease held through periodic expense charges is also spoken of as amortization. The word is often used as a synonym for depreciation.

Obsolescence. Improvements in machinery, resulting in lower operating costs, make existing machinery less valuable. The loss is known as obsolescence. The value may be completely or partially wiped out through such improvements. Generous provision for obsolescence should be made in all chemical manufacturing costs. An exact value for it cannot be determined because such a figure depends on future conditions which cannot be precisely appraised.

Depreciation. The continued use of machinery wears it out. The loss of value caused in this way is known as depreciation. Idle machinery also depreciates.

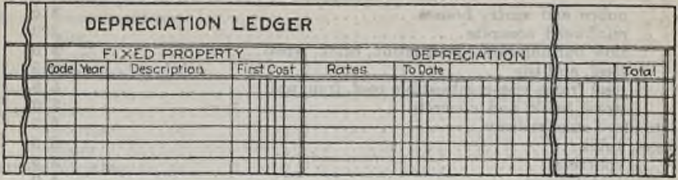
Depreciation Methods. Many depreciation methods have been proposed, but only three of them are of value in the manufacturing industries.

1. Straight-line depreciation in which a fixed percentage of the first cost is set up each year in a reserve fund.
2. Diminishing balances in which a fixed percentage of the depreciated first cost is set up each year in a reserve fund.
3. Useful life method (used mainly for dies and tools) in which the number of possible operations is estimated and the credits to a depreciation reserve are made in proportion to the productive operations.

In order not to mix the accounts, the charge for depreciation is made to the manufacturing operations and the corresponding amount is credited to a reserve account which is usually stated as an offset to the first cost. The original book value thus always appears intact on the books (see Reserve Accounts).

Depreciation Rates. A compilation of rates for various assets is to be found in Kilduff, "Auditing and Accounting Handbook," McGraw-Hill. These can be used to work out a proper average rate—commonly used rates for chemical plants are 5 per cent on buildings and 10 per cent on machinery. If renewals are to be charged against depreciation even higher rates are desirable. Table 6 indicates the average experience for different types of equipment as listed by Kilduff ("Auditing and Accounting Handbook," McGraw-Hill) (only those types more generally associated with chemical plants have been selected). In using these or other percentage figures it should be borne in mind that any value representing average conditions must be raised or lowered to compensate for the amount that actual operating practice departs from the normal plant experience. A pamphlet showing the depreciation rates for a wide range of assets as compiled by the Bureau of Internal Revenue can be obtained from the Government Printing Office at Washington, D. C.

Depreciation Ledger. Figure 2 illustrates a ledger sheet suitable for first-cost plant values and the depreciation charges for 10 years. A ledger like this can be used where each separate asset is depreciated at its own rate rather than by a flat value for the entire plant. A separate sheet should be used for each department.



DEPRECIATION LEDGER

FIXED PROPERTY				DEPRECIATION			
Code	Year	Description	First Cost	Rates	To Date		Total

Fig. 2.—Depreciation ledger.

New Construction. The charges originate through work done on contract or by the company itself. The complete costs are obtained from purchase invoices, from work orders, from supply requisitions, and from interdepartmental machinery transfers. A large amount of detail is involved. Accounts should meet the needs of the engineering, the cost, the production, and the accounting departments. New construction accounts should be closed promptly when the work is finished, and the values should be transferred to the fixed-property accounts.

Betterments. When machinery is moved to a new location to facilitate routing of materials this is known as a "betterment." The costs of such improvements to existing machinery cannot be charged to the Plant Account until the original costs have been analyzed so that all duplications can be eliminated.

Table 6. Depreciation Rates*

	Per Cent
Air compressing system	10.0
compressors	5.5
Autoclaves	8.5
Automobiles, passenger	27.5
trucks	25.0
Barges, wood	7.5
Belting, canvas transmission	35.0–25.0
leather transmission	20.0
rubber transmission	25.0
canvas conveyor	40.0
rubber conveyor	30.0
Blowers, cooling	6.0
drying	6.5
Boilers, heating	8.5
high pressure, fire tube	7.0
high pressure, water tube	5.0
yard	10.0
Bolt and nut machinery	7.0
Boring and drilling machines	10.0
Buildings, all wood, cheap construction	8.5
all wood, well built	4.0
brick and steel	2.5
brick and wood	3.0
concrete	2.0
concrete blocks, wood roof and floors	2.5

* Selected from Kilduff, "Auditing and Accounting Handbook," McGraw-Hill, New York.

Table 6. Depreciation Rates—(*Continued*)

	Per Cent
Buildings, corrugated iron, wood frame, concrete floor	5.0
wood floor	6.5
police and sentry houses	8.5
reinforced concrete	2.0
slow burning mill construction, brick, wood	3.0
steel, and tile	2.5
steel frame non-combustible roof, iron wall	5.5
stone, brick, and concrete	2.0
Cars, standard steel railway	5.5
Chimneys, brick	3.5
concrete	3.5
metal, lined	5.0
metal, unlined	8.5
Circuit breakers	8.5
Clutches, transmission, friction	8.5
Conveyors, ash	8.5
belt	12.5
chain	8.0
gravity rollers	6.0
pan	7.5
portable	12.5
screw	8.5
Countershafts, speed reducers, etc	8.5
Cranes, traveling	8.5
electric cab controlled	6.5
locomotive	7.5
Crushers	10.0
jaw	12.5
rock	15.0
Derricks, movable	12.5
stationary, steel	7.5
wood	10.0
Desks, office, metal	5.5
wood	7.5
factory, metal	10.0
wood	12.5
Dies, drop forging, stamping, etc	33.3
Docks	3.5
Drainage system	5.0
Drilling machinery	8.5
Dryers, direct steam	6.0
paddle	6.5
rotary	7.0
Ducts, mechanical draft	6.5
Dwellings, concrete rented to employees	4.0
frame rented to employees	5.0
Dynamos	6.5
Electric distributing system, inside	5.5
outside	6.5
Elevators, bucket	15.0
electric, freight	8.5
passenger	6.5
Engines, gas	7.5
oil	7.5
steam, Corliss	4.0
high speed	6.0
low speed	5.0
Exhaust fans	7.5

Table 6. Depreciation Rates—(*Continued*)

	Per Cent
Feedwater heaters	5.0
pumps	5.5
Fences, brick or concrete	4.0
wire	8.5
wood	10.0
Fuel-oil distributing system	7.5
Furnaces	12.5
Hoists, chain	10.0
electric	7.5
steam	7.5
Kettles, nitrating	20.0
soda	20.0
sulfonating	20.0
steam jacketed	12.5
Lathes	7.5
Locomotives, electric	4.0
gasoline	10.0
steam	5.5
steam or electric, narrow gage	6.5
Meters, flow	5.0
Milling machines	10.0
Motors, electric, alternating current	7.0
direct current	9.0
Ovens, annealing	10.0
brick	9.0
Packing machinery	8.5
Panels, switchboard	5.0
Pavements, asphalt	7.0
brick	5.0
concrete	5.5
granite	3.0
gravel	10.0
wood block	6.5
Pig-casting machines	20.0
Pipe-threading machines	5.5
Pipe systems, air	9.0
drainage	5.0
gas	4.0
steam, sprinkler	5.0
Pulleys, steel	5.0
wood	6.0
Pumps, acid	10.0
centrifugal	6.5
geared power	5.0
vacuum	8.0
Radiators, steam or water	5.0
Railways, narrow gage	8.0
standard gage	7.0
Rolls, crushing	7.0
Runways, crane, inside	6.0
outside	8.5
Saws, power, hack	8.5
Scales, automatic	10.0
industrial railway	7.5
platform	9.5
wagon	7.0
Screens, coal, sand, etc	10.0
heavy stone	12.5

Table 6. Depreciation Rates—(*Concluded*)

	Per Cent
Separators, centrifugal	10.0
Shafting, main line	5.0
Sidewalks, concrete	4.5
Starters for motors, electric	25.0
Stokers, mechanical	7.5
Tanks, storage for air, etc	4.0
manufacturing or storage	5.0–20.0
Towers, transmission lines	5.5
Transformers	7.0
Water mains	5.0
Wells, water	2.5

Replacements. In the literature of accounting, the term replacement is used synonymously with renewals and, to some extent, with repairs. It is good practice to assign different meanings to these expressions, since their separate services are needed to define different activities. "Replacements" is a convenient term for stating the cost of substituting a new unit for an old one, but it should not be held down to the narrow limits of having the two units identical. The technique of replacement accounting differs from new construction only in that an asset value must be extinguished in total on the fixed-property records before replacement costs can be entered.

Renewals. Putting new parts in an old machine is a renewal. As this increases the life of the machine, the cost of the parts and the labor of installation should not be charged against production but should be charged as an offset to depreciation. Present value of a machine is first cost, less depreciation, plus renewals. This procedure does not increase unit production costs but instead increases the time required to recover the first cost of the machine. Depreciation rates must be set high enough to cover the cost of renewals made within the life of the machine.

Repairs and Maintenance. The term repairs, popularly linked with maintenance, does not convey a precise meaning, and it is difficult to make the term less obscure. It is often used to refer to major upkeep, maintenance usually referring to minor upkeep. It is hard to draw a fine line between upkeep and renewals. Any attempt to demarcate the two by setting an arbitrary value at which a maintenance item becomes a renewal is not very

Table 7. Loss Calculation for a Scrapped-plant Unit

First cost of unit		$10,987.58
Depreciation reserves, 4½ years at 15 per cent to be deducted		7,416.62
Depreciated value		3,570.96
Renewals to be added		563.08
Present book value		4,134.04
Value to be realized:		
Salvage	$ 350.00	
Fixed property for sale	1,925.00	
Total to be deducted		2,275.00
Value to be scrapped		1,859.04
Liquidation expense		750.00
Loss on scrapped-plant unit		$ 2,609.04

satisfactory. Such a procedure easily might become a case of chopping off the dog's tail by inches, for, should we desire to do renewals in a piecemeal fashion, they would be charged off as maintenance. The answer lies in being lenient as to the interpretation of renewals and liberal as to depreciation rates. This practice insures more uniform costs which, after all, is one of the primary necessities of cost accounting.

Idle Fixed Property. The charges for carrying idle property should be segregated on the books so that the loss incurred is known.

Scrapped Plant. Table 7 shows how the loss incurred through the scrapping of a plant unit is calculated. Salvage refers to material and machines sold before and at the time the unit was actually scrapped. Fixed property for sale refers to unsold machines. Liquidation expense refers to the cost of dismantling.

Fire Loss. Calculations for loss incurred through plants destroyed by fire are similar to those for a scrapped plant.

COST ACCOUNTING

Cost Systems. Various classifications, such as special order, product, class, operation, process, etc., have been proposed. These distinctions in cost systems are somewhat confusing because they lay undue stress on a minor detail of technique. There are only two possible ways of figuring costs, namely:

1. **Standard basis** in which predetermined standards are calculated and are from time to time modified to conform to a more detailed experience. No other unit cost calculations are made for book entry, though the actual unit costs are usually checked. These standards are used for sales and other work. The cost-accounting system is carefully designed so that all departures from these predetermined standards can be effectively analyzed. These differences are accepted as profits or losses. The standard-cost method is simple, accurate, economical, is founded on correct principles, and makes results of operations available at the earliest possible moment. Further information is given under Relation of Output to Costs, and under Building Space.

2. **Actual basis,** in which the attempt is made to find out what things really cost by dividing expenses by actual output. This method is complicated, inaccurate, unsound in principle, and does not make results of operations available promptly. Costs cannot be figured for a short term without injecting numerous errors. Actual costs have no particular value if they are not economically correct—they cannot be passed on to customers and they are not in a form to suggest ways of reducing high costs. The actual-cost method has been popular with accountants, but the trend is toward the standard method, and combined actual and standard cost methods are being favored.

Cost-accounting Technique. In its simplest terms a cost system has to deal with many forms and figures and, therefore, considerable attention should be given to technique. The cost of operating a system depends to no small degree on the care with which the detailed technique has been worked out.

Cost Distributions. One of the difficult problems in cost work has been to properly charge expenses to finished goods. Some items can be easily charged to manufactured goods, but others appear to bear little relation to the finished articles. This has led to various methods of distributing indirect expenses: in proportion to the hours of direct labor, in proportion to the time of machinery; etc. This practice is dangerous because many substantial cost items are not so related to the finished product. Process steam rarely has any connection with the hours of direct labor expended on production. The only satisfactory solution to this problem is to departmentalize a business and to charge all non-material costs to specific departments. The cost of operating each department can then be determined and contrasted with its productive activity.

Departmental Organization. Functional activity is the guiding principle in departmental organization. When a function is not present in a sufficiently large volume to stand alone it must be merged with some other function to which it is closely related. (The usual procedure is to pass the work on to someone who has spare time.) Combinations of functions should not be made unless the total cost can be reduced to a rational unit basis.

Departments. The departments of a business may be exhaustively divided into three classes. These groups and their characteristic subdivisions are shown in Fig. 3.

<table>
<tr><th colspan="3">DEPARTMENTS</th></tr>
<tr><th>COMMERCE</th><th>SERVICE</th><th>PRODUCTION</th></tr>
<tr><td>Administrative
Sales
Office
Warehouse</td><td>Transportation
Mechanical
Technical
Labor maintenance
Power - Steam
Power - Electrical etc.
Real estate
Supervision</td><td>Departments
organized to get
the most effective
and economical
cost control, using
buildings,
processes,
machines etc.,
to set lines</td></tr>
<tr><td rowspan="3">GENERAL OVERHEAD</td><td>DISTRIBUTED</td><td rowspan="2">PROCESS COSTS</td></tr>
<tr><td>ADDED FACILITIES</td></tr>
<tr><td colspan="2">WASTED EFFORT</td></tr>
<tr><td colspan="3">RESULTS</td></tr>
</table>

FIG. 3.—Departmental organization.

Departmental Output. Figure 3 also shows the results obtained through the operations of the three groups of departments. The effect of their efforts will be expressed as:

1. General overhead, which is not reducible to specific products.
2. Wasted effort, or cost variation which can be either a gain or a loss.
3. New facilities, in the form of construction, research developments, etc.
4. Process changes, which are instrumental in converting raw materials into finished products.

Production Control. The management of a business can be simplified if a suitable technique for the control of production is adopted. Figure 4 shows the graphic method. Various boards are also in use. The idea is to follow the units in process by cards, metal disks, pins, etc. The particular method found to be most effective depends largely on the kind of data which must be correlated. Activities should be controlled in both the service and production departments.

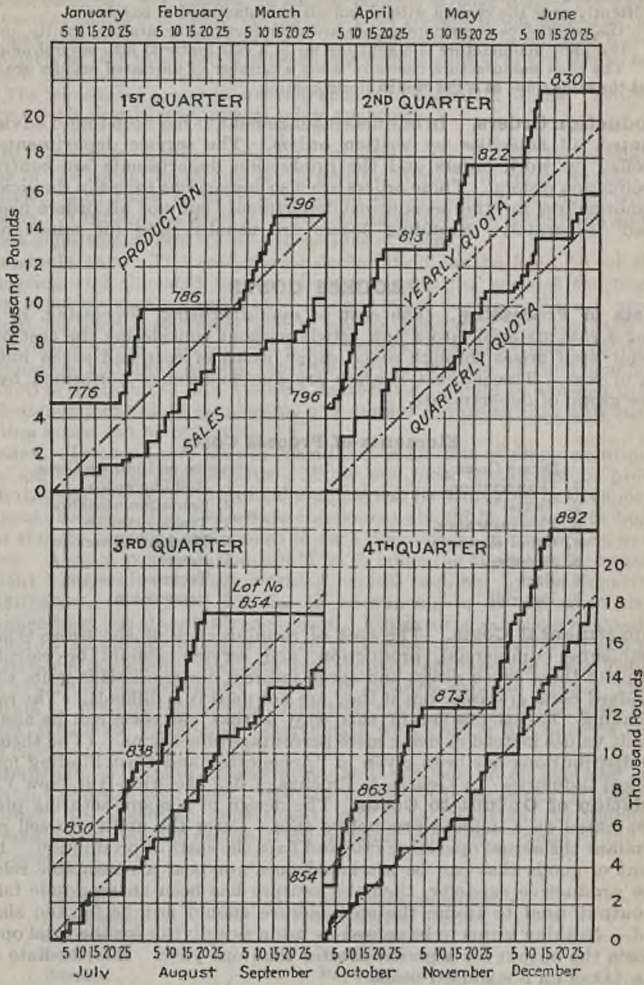


FIG. 4.—Production control.

Productive Operations. There are three separate points to be noted in the functioning of productive operations.

1. Goods are made for stock or for special order. In some trades they speak of the goods made in anticipation of orders as catalogue items and of goods made to order only as jobbing work.

2. Machinery is operated continuously as in the manufacture of sulfuric acid, or intermittently as is the custom with chemicals manufactured in batches.

3. In the productive operations, goods are manufactured separately, jointly, or as a class. The joint manufacture is a splitting up of a raw material into several new products. The class manufacture comes in when a number of unrelated articles are processed at the same time, as is the case in electroplating.

Production Orders. In business management, it has been found advisable to control all activities by written orders. The service departments are controlled by work orders and the production departments are controlled by production orders. These orders are also convenient to use in duplicate as a memorandum for the accounting department, but not all orders need be "costed" separately. Standing orders are usually used for minor repair work.

PROCESS COSTS

Costs of Processing. The cost of manufacturing, processing, or converting a raw material into a new article without considering the cost of the material itself arises through seven direct items of cost and seven indirect items of cost. The indirect charges are due to services performed by the service group of departments.

Elements of Process Costs

Direct Costs	Service or Indirect Costs
1. Depreciation.	1. Building space.
2. Taxes.	2. Labor maintenance.
3. Fire insurance.	3. Transportation.
4. Other expenses.	4. Mechanical service.
5. Supplies.	5. Power.
6. Fuel.	6. Technical service.
7. Pay roll.	7. Supervision.

Departmental Costs. The cost of running each of the three types of departments—commercial, production, and service—should be calculated in terms of these seven direct and seven indirect costs. This is quite simple if standard costs are used, but if they are not used it is difficult. The reason for making the calculation in this way is that the costs can be assigned real unit values instead of using mere percentage allocations. (The theory of these distributions has been given in "Accounting and Cost Finding for the Chemical Industries" by George A. Prochazka, Jr., McGraw-Hill, New York.)

Relation of Output to Costs. The design of a manufacturing plant is always based on a definite productive rate. Later the ability to sell goods determines the actual operating rate and fixes the costs of production. If the amount of goods that can be marketed does not bear a reasonable relation to the productive capacity, then the venture has been an economic failure. The output used to frame the cost picture should not be for too short a period. Nothing seems to be gained by using as a divisor for the total operating costs the output for a period shorter than one year. Intermediate costs can be taken on a standard basis.

Processing Units. The reduction of operating expenditures to unit process values must be logically done to be of value in cost accounting. The principles involved are:

1. A normal operating rate must be set at some percentage of capacity, such as 65, 75, 85, etc. Departures from the rate scheduled for a given period are held to create profits or losses and are not interpreted as giving increased or decreased costs.

2. When more than one result is related to the operating expenditures, then compound units are required. For instance, in sending material through a dryer the significant result is the removal of moisture, yet the ratio of dry material to wet determines, in part, the productive capacity of the machine. The conditions must be analyzed to find out if a number of unit values are required to suit varying conditions and also to find out just how much weight to give each factor.

3. The immediate purpose in selecting units is to evaluate a process. Article costs can be quickly determined if process values are known.

Unit Process Costs. Standard process costs should be calculated in terms of the seven direct and seven indirect costs or in terms of such other subdivisions of these as have been found to be more in harmony with the problem. Unit process costs must be calculated for each production and each service department, and for all subsections of them.

Depreciation. The cost due to depreciation can be fixed by an annual calculation and allocated to each department on the basis of the first-cost values used by that department. A more exact way is to have a property account for each department and to use an individual rate for each asset. This method makes it possible to study renewal and repair costs jointly with depreciation and thus obtain the true cost of the use of each machine in productive operations. All building depreciation should be charged to a real estate department or to an account of a similar nature so that unit costs for building space can be obtained.

Taxes. Income taxes are not considered as an item of manufacturing cost (they are deducted from surplus). State and local taxes must be provided for in the cost sheets. The annual amount can be estimated in advance from previous charges. The real estate component should be charged to building space and the personal component to each department in proportion to plant values. A more exact solution would be to make the allocation in proportion to plant values plus average working capital required by the department.

Insurance. The best practice in accounting is to segregate the fire insurance from the other insurance costs. Fire insurance is most conveniently handled by an annual calculation similar to those for depreciation and taxes. Liability-insurance costs fluctuate with the pay rolls, and the other insurance costs are usually conveniently treated as separate expense items. The distribution of fire-insurance costs should be made in accordance with underwriters' schedules, penalizing departments with greater fire hazards for the extra risk.

Expenses. A careful classification and analysis of all expenses are an important phase of cost work. Usually many of the items can be charged *in toto* direct to a specific department, but it is important to understand thoroughly the detailed nature of all business expenses. Some of the more usual ones are:

General.
Advertising.
Mechanical.
Office.
Repairs.
Selling.
Technical.
Transportation outbound.
Miscellaneous insurance.
Employees' liability.
Public liability.
Elevator liability.
Pay-roll liability
Boiler liability.
Group—accident, health, life, etc.

Supplies. It is costly to place all supplies on a perpetual inventory basis (see Material Accounting). Usually it is best to get a close check on important items by putting them on a perpetual basis, the others being controlled by purchase analysis. Office supplies and many nondescript items can be treated as expense by writing them off at the time they are purchased. A Spare Parts account is convenient for recording the purchase of new parts for machinery. Usual subdivisions for supplies are:

1. Factory.
2. Mechanical.
3. Metered—electricity, gas, water—when purchased.
4. Office—stationery, etc.
5. Packing.
6. Technical—laboratory, engineering, etc.

Fuel. When fuel is used on a sufficiently large scale to warrant weighing equipment, fuel meters, etc., then the balancing of the fuel account is not a difficult matter. The smaller plants without such auxiliary devices can get a reasonably close check from an analysis of railroad weights and an accurate record of disposition of coal receipts as to stock piles, power-house bin, etc. A check over a 3-month period will give much detailed information as to consumption. Costs of freight unloading, handling, etc., must be added to the coal invoice to get the price per ton. After these values have been ascertained, a standard value for inventory, for costing, and for other purposes can be established.

Pay Roll and Salaries. The problem of establishing labor and salary costs has seven phases, namely:

1. **Wage payments** are made in a number of different ways. All wage-payment plans to be effective must be easily understood by the worker. The simplest method is to pay the worker a fixed amount by the week, or by the day, and have the foreman see that the time agreed upon is put to use. A more common method is to pay on an hourly basis, using a time clock to control the amount to be paid. Piece rate, *i.e.*, paying an agreed amount for each operation performed, is a method commonly used in the metal-working industries. The straight piece rate has been scientifically refined into an incentive plan. There are some 12 recognized methods, each one being designed to meet particular conditions in the metal-working trades. The periodic payments of bonuses arbitrarily determined or graduated according to the productive rate is the most commonly used incentive plan in the chemical industries.

2. **Time cards** usually serve to fix the number of hours a man has spent in the factory. Some plants also use these cards to charge work to specific accounts or to allocate overhead, but these two activities are not the proper functions of time cards. At the end of a month it is necessary to split the entries on a weekly card so that the monthly charges can be properly entered on the books. Cards are usually handled on a numerical basis, each worker being assigned a number which has been coded to designate the department in which he works.

3. **Pay-roll sheets** should be designed so as to permit making up weekly statements and monthly cost records. Only the total hours worked per week and the total weekly payment need be recorded on the sheet. Room must also be provided for deductions, transfers, bonuses, incentives, etc. The sheet must be arranged so that the cost chargeable to each department can be taken off monthly as a subtotal.

4. **Labor transfers** among departments should be avoided as the practice is confusing. If much of this is essential special transfer slips must be provided debiting one department and crediting another. Room must be provided for the signatures of the foremen of the departments.

5. **Daily labor reports** must be made out by each department giving the classified charges for each man's time. Each man's time must be checked against his time-card record, and the total time for all men in the department must be compared with the total of the classified work.

6. **Monthly distribution sheets** are required to summarize the information collected under Item 5. These values must foot up to the same amount as those given on the pay-roll sheets (Item 3) before the data can be used on the cost records.

7. **Salary distributions** are usually a simple matter; the payment to each man is charged to a department or is prorated over a number of departments.

Labor Maintenance. The amount of money which must be expended to hire and to maintain a worker at his task is substantial. These costs must, therefore, be collected so that they can be distributed to departments in proportion to the hours of labor employed, or the dollar of the pay roll. Labor maintenance can be completely distributed to the departments or a standard-rate per cent of the pay-roll can be used—the unabsorbed amount being charged off as a loss and the overabsorbed amount being added as a profit. Some of the principal costs of labor maintenance are:

1. Personnel department to hire workers.
2. Salaries of clerical workers to keep labor records.
3. Cost of office space and equipment for clerks.
4. Cost of supplies used by clerks.
5. Cost of lunch-room space for employees.
6. Cost of change-house space, water, heat, etc.
7. Cost of pay-roll insurance, etc.
8. Cost of employees' liability insurance.
9. Cost of group life, and other insurance.

Building Space. To determine the annual cost of providing 1 sq. ft. building space it is necessary to consider two problems:

1. The annual cost of providing the land (*i.e.*, real-estate factor). This cost is due to certain general services, such as those given below, which are essential to the proper maintenance of a group of buildings. These charges can be distributed to individual buildings on an area, first-cost, or cubic-volume basis, which may be modified to meet special service considerations.

a. Watchman day or night.
b. Sanitation—cleaning of pavements, sewers, etc.
c. Yard illumination.
d. Maintenance of fences, pavements, sewers, fire equipment, etc.
e. Water for cleaning and other general purposes.

2. Building space costs are due to the items given below, of which *h* is obtained by a distribution of the costs given in Problem 1.

a. Depreciation.
b. Taxes.
c. Fire insurance.
d. Expenses:
    Elevator insurance.
    Repairs.
    Painting.
    Mortgage interest.
e. Supplies:
    Janitor.
    Miscellaneous
f. Pay roll:
    Janitor.
g. Labor maintenance.
h. Land component or real-estate factor.
i. Power:
    Heat.
    Light.
    Water.
    Power for elevators.
j. Transportation:
    Refuse removal.
k. Mechanical service:
    Sundry maintenance and repairs.
l. Supervision.

Building-space costs can be allocated each month so as to dispose of all charges, or they can be allocated on a standard-cost basis. The second method is preferable as proper provision can then be made for sporadic and unusual expenses which are so characteristic of real-estate maintenance. Some items do not even appear each year but are encountered only after a period of years. Units can be taken on a square-foot basis or better on a cubic-foot or volume basis. No building should be penalized for a location which is remote from the center of activities, and no manufacture should be subject to higher costs because it is located on one of the upper floors of a building. The logic of multistory buildings is to obtain a lower average cost than would be possible with a single-story building.

Power. Under the caption, power, are included the costs of obtaining or generating and distributing such items as

Process steam.
Electricity.
Compressed air.
Water.
Vacuum.
Gas.
Refrigeration.

Process Steam. The principal cost elements of process steam are as follows:

Depreciation.
Taxes.
Fire insurance.
Expenses (boiler insurance, etc.).
Supplies (oil, packing, etc.).
Fuel.
Pay roll (firemen, engineers, etc.).
Labor maintenance.
Building space.
Power (for handling coal, water for boilers, etc.).
Transportation (ash removal, etc.).
Mechanical service (pipe-line and boiler-room repairs, etc.).
Technical service (boiler-room control).
Supervision.

Process-steam costs must always be calculated delivered at the consuming points; any other practice would penalize remote places, which is not proper accounting practice. Credit can be given to a department for heat units in condensed water from utilized process steam, but steam condensed during pipe-line transmission should not be so handled as improperly to credit or debit an operating department merely because it has a disadvantageous or more advantageous location. If meters are not in use, then the best procedure is to allocate the steam according to the theoretical or estimated consumption for each process. The usual unit of measurement is per 1000 lb. and occasionally per net ton.

In the chemical industries, it is often advantageous to use for process work steam which has been exhausted at a suitable back-pressure from electrical power generating equipment. The costs of steam, so utilized, can be allocated satisfactorily in some cases on a heat unit basis. However, as a heat unit at a high temperature is more valuable than a heat unit at a low temperature, consideration usually must be given to the amount by which the water rate of a turbine is increased when the turbine is bled for process steam. For a detailed discussion of this subject see "Power and Heat in the Industrial Plant," R. J. S. Pigott, *Trans.* A.S.M.E. 1929, FSP-52-38.

Electricity. The cost elements for electricity are similar to those outlined for process steam. Power costs should be calculated delivered at the power-house bus bars and to this must be added the average cost for delivery at the factory motor.

The cost distributions are for such things as (1) motors, (2) lighting systems, (3) furnaces, etc. If meters are not available, daily records of the number of hours each device is used will help in making allocations. Machine ratings should be checked by a portable recording wattmeter to establish the average current consumption for existing load conditions. Similar methods can be employed for the lighting system and for furnaces, appliances, etc.

Compressed Air. The usual unit for calculating compressed-air costs is per 1000 cu. ft. free air compressed per minute. If the system is large, it is advisable to meter the air because it is a costly factory commodity. The cost should be figured as an average delivered value. If the system is small, it is sufficient to allocate the total cost according to the amount of air estimated as having been consumed by each department. The problem of vacuum costs is the same as that for compressed air, but the unit is per 1000 cu. ft. free air evacuated per minute.

Gas. If gas is manufactured, cost calculations similar to those outlined for process steam are required. If the gas is purchased, the average cost delivered at the various departments should be established.

Refrigeration. The cost unit involved is per ton of refrigeration, *i.e.* the abstraction, in 24 hr., of an amount of heat equal to the heat of fusion of one net ton of ice. This is equivalent to 286,600 B.t.u. per day or 199.028 B.t.u. per min. The round number 200 B.t.u. per min. is usually used. If steam-driven compressors are used, credit must be given for the exhaust steam which is usually desired for feed-water make-up, drying operations, etc.

Water. The usual units for water are per 1000 cu. ft. (municipal practice) or per 1000 gal. The average conversion factors for these units are 1000 cu. ft. = 7480.4 gal. and 1000 gal. = 133.68 cu. ft. The water may be purchased or pumped from a stream or a well. As the distribution system is usually extensive, costs must be figured on the delivered basis. If the input is metered, and the departments are all metered, the amount of water used in fire protection, yard cleaning, etc., can be found by the difference. For process work, water often must be treated, and this cost must be added. It is also possible that more than one grade of water will be furnished in the factory, *i.e.*, hot, cold, treated, untreated, etc.

Transportation. Materials must be moved within the factory and finished goods must be sent to customers. Some of the more usual costs of transportation are assignable to:

Raw-material unloading.
Fuel handling.
Ash removal.
Departmental shifts of material
  Raw.
  In process.
  Finished.
Materials for construction, repairs, etc.
Yard cleaning, snow removal.
Shipping of goods to customers.
Warehouse work.

The cost of transportation work has two parts: (*a*) cost of operating trucks and other transportation equipment, and (*b*) cost of labor. Each of these can be established on an hourly basis in accordance with the principles that have been laid down for the other service departments. A daily record of work performed is an essential background for a proper cost allocation.

Mechanical Service. The cost problem of the mechanical department is analogous to that of the transportation department. Two unit costs, namely, straight labor hours and shop hours are obtainable. With large shops, a detailed breakdown of costs as with production departments is essential. The allocation of costs also requires daily labor records similar to those that have been described under the labor section. Work is generally controlled by an order system; suggestions as to this are given under Production Order.

Technical Service. The cost of laboratory, engineering, etc., can be worked out on an hourly basis using the full department costs, or such parts of them as are required, on a standard-cost basis. With routine work, a standard cost per test should be devised. Research work should be charged to general overhead or be set up as an asset in the capital accounts.

Supervision. The cost of supervision can be completely allocated to the various departments on a percentage basis. Another method is to work out a standard amount for each department running at normal capacity. Each department is charged that amount every month—differences between the total actual and total standard cost being written down as profits or losses. The advantage of this method is that it always tends to control costs by offering models for comparison.

Interest on Investment. In spite of the fact that interest on investment is frequently included in cost sheets, it is not a legitimate element of cost. A manufacturer hopes to get a greater profit than the rental value of a sum of money equal to his investment. The place to give interest on investment consideration is in a comparative study of profits. However, interest on money borrowed for construction purposes is a legitimate item in the cost of the erected asset.

MATERIAL ACCOUNTING

Materials. To be classified as a material, the item under consideration must add an appreciable amount to the cost of the manufactured article, and its consumption must vary in direct proportion to the rate of production. It is not essential for the material itself to be present in the finished goods. With this viewpoint, some of the expensive supply items can be treated as raw materials and some of the unimportant materials can be controlled by the less expensive methods used for supplies.

Material Control. The primary object of a warehouse control is to check the receipts and the disbursements of materials, and the secondary object is to obtain the stock on hand, or inventory. The two places to control materials are (1) in the warehouse and (2) in the factory where goods are put into process. In addition to the many technical controls for quality, materials can be quantitatively controlled in connection with any of the following 10 transactions:

1. By the seller when he ships goods.
2. By the warehouse when goods are received.
3. By the warehouse when goods are delivered for process use.
4. By the production department when the goods are received.
5. By the production department when goods are put into process.
6. By the production department when goods are removed from process.
7. By the production department when goods are delivered to warehouse.
8. By the warehouse when goods are received from the production department.
9. By the warehouse when goods are shipped.
10. By the customer when he receives goods.

Stock Sheets. From a controlling viewpoint there are two kinds of materials: (1) package goods, and (2) bulk goods. A much closer check is possible with package goods because each container, being serially numbered, can be recorded as having been shipped to a particular customer or as having been put into a definite manufacture. Such facts can be noted on the same horizontal line with the original entry denoting the receipt of the package. With bulk goods similar principles can be applied to separate shipments. Figures 5 and 6 show two types of stock sheets. Cards do not offer so satisfactory a medium as a loose-leaf book with post binders. Formal sheets can be used for warehouse control and an ordinary note-book for factory control.

Receipts. The differences between shippers' weights and warehouse weights must either be written off, or a credit must be obtained for this discrepancy. A company policy must be established to cover such losses.

STOCK RECORD																
RECEIVED								DELIVERED								Remarks
Date	Identification Nos.			From	Gross	Tare	Net	Date	Order No.	To	Gross	Tare	Net	Error		

Fig. 5.—Package-goods stock sheet.

STOCK RECORD											
RECEIVED						BALANCE ON HAND	DELIVERED				Remarks
Date	Identification Nos			From	Quantity		Date	Order No	To	Quantity	

Fig. 6.—Bulk-goods stock sheet.

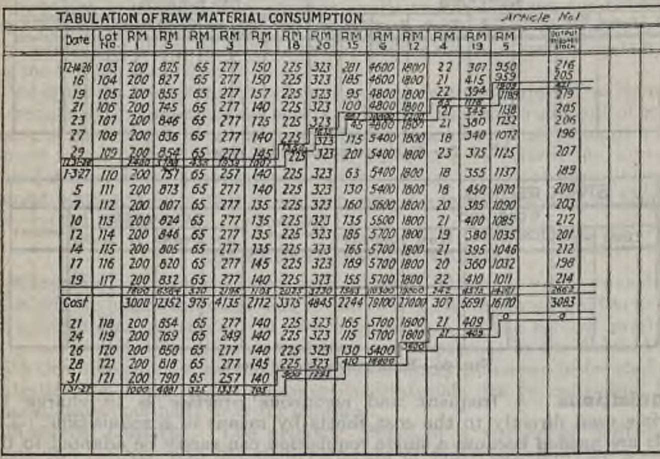
Requisitions. A frequent and erroneous practice is to charge the materials used directly to the cost sheets by means of a requisition. Two records are needed because a single requisition can rarely be adapted to this double purpose, denoting (1) the withdrawal of material from the warehouse, and (2) the entry of raw material into a process.

CONTROLLING FACTORY MATERIALS			
RAW MATERIAL CONSUMPTION			
FIXED	VARIABLE		
Single Product	Single Product	Group Products	
		Class Products	Joint Products
Charge through standard bill of materials	Charge through process card recording consumption	Charge through process card recording consumption and apportion by weight volume area numerical units	Charge through process card recording consumption and apportion by empirical methods
To a Job, Order, Lot, Batch or a Continuous Operation			
Article Costs			

Fig. 7.—Principles of material control.

Inventories. A physical inventory should be taken before a purchase order is placed. If this procedure is followed and the receipts and disbursements are carefully checked, then the book balance will always give the true inventory. Similar methods of inventory control should be used in the factory.

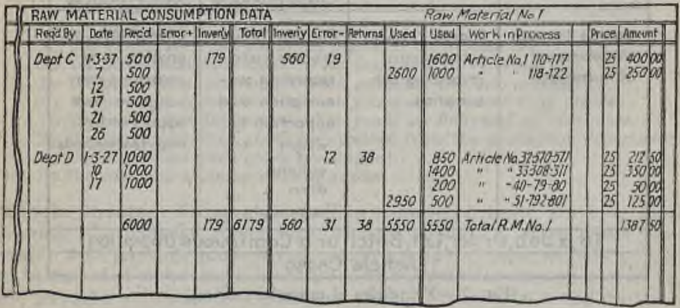
Raw-material Consumption. Figures 7, 8, and 9 show the essential steps in factory-material control. Figure 7 gives the general principles which underlie the charging of raw materials to an article cost sheet. Figure 8 illustrates the well-known tabulation of raw-material consumption. Figure 9 shows the reconciliation which must be made between the tabulation of consumption and the raw-material receipts from the warehouse.



TABULATION OF RAW MATERIAL CONSUMPTION Article No. 1

Date	Lot No.	RM 1	RM 5	RM 11	RM 3	RM 7	RM 18	RM 20	RM 15	RM 6	RM 12	RM 4	RM 19	RM 5	Output Shipped to Stock
12-14-26	103	200	825	65	277	150	225	323	281	4600	1800	22	307	950	216
16	104	200	827	65	277	150	225	323	185	4600	1800	21	415	959	205
19	105	200	855	65	277	157	225	323	95	4800	1800	22	394	1189	219
21	106	200	745	65	277	140	225	323	100	4800	1800	21	345	1138	205
23	107	200	846	65	277	135	225	323	45	4800	1800	21	380	1232	206
27	108	200	836	65	277	140	225	323	115	5400	1800	18	340	1072	196
29	109	200	854	65	277	145	225	323	201	5400	1800	23	375	1125	207
12-31-26		1400	5788	455	1939	1007									
1-3-27	110	200	757	65	257	140	225	323	63	5400	1800	18	355	1137	189
5	111	200	873	65	277	140	225	323	130	5400	1800	18	450	1070	200
7	112	200	807	65	277	135	225	323	160	5600	1800	20	385	1090	203
10	113	200	824	65	277	135	225	323	135	5500	1800	21	400	1085	212
12	114	200	846	65	277	135	225	323	185	5700	1800	19	380	1035	201
14	115	200	805	65	277	135	225	323	165	5700	1800	21	395	1046	212
17	116	200	820	65	277	145	225	323	169	5700	1800	20	360	1032	198
19	117	200	832	65	277	140	225	323	155	5700	1800	22	410	1011	214
		1600	6564	520	2196	1105	2025	3230	1563	103400	19800	242	4375	14287	2862
Cost		3000	12352	975	4135	2112	3375	4845	2244	79100	27000	307	5691	16170	3083
21	118	200	854	65	277	140	225	323	165	5700	1800	21	409	0	0
24	119	200	769	65	249	140	225	323	115	5700	1800				
26	120	200	650	65	277	140	225	323	130	5400					
28	121	200	618	65	277	145	225	323	410	16800					
31	121	200	790	65	257	140	900	1292							
1-31-27		1000	4081	325	1337	705									

FIG. 8.—Tabulating raw-material consumption.



RAW MATERIAL CONSUMPTION DATA Raw Material No. 1

Req'd By	Date	Rec'd	Error +	Inven'y	Total	Inven'y	Error −	Returns	Used	Used	Work in Process	Price	Amount
Dept C	1-3-37	500		179		560	19			1600	Article No. 1 110-117	25	400 00
	7	500							2600	1000	" " 118-122	25	250 00
	12	500											
	17	500											
	21	500											
	26	500											
Dept D	1-3-27	1000					12	38		850	Article No. 32-570-571	25	212 50
	10	1000								1400	" " 33-308-311	25	350 00
	17	1000								200	" " 40-79-80	25	50 00
									2950	500	" " 51-792-801	25	125 00
		6000		179	6179	560	31	38	5550	5550	Total R.M. No. 1		1387 50

FIG. 9.—Checking raw-material consumption.

Raw-material Pricing. Materials should always be priced at values which experience has shown to be reasonably safe figures. It is not desirable to upset cost calculations by trying to record each minor change in material costs. Such differences can be written off as a profit or a loss, or can be charged to an inventory reserve. Raw-material costs must be complete; *i.e.*, they must be figures laid down in the factory warehouse so as to include all freight and handling costs. Storage costs, however, are a general overhead item and must not be included (material is usually worth less, and not more, when it lies around).

Material Ledgers. A convenient detailed technique for recording material values is shown in Figs. 10 and 11. A yearly sheet is used for each material. The debit and credit entries can be standardized as shown for raw materials in Fig. 11.

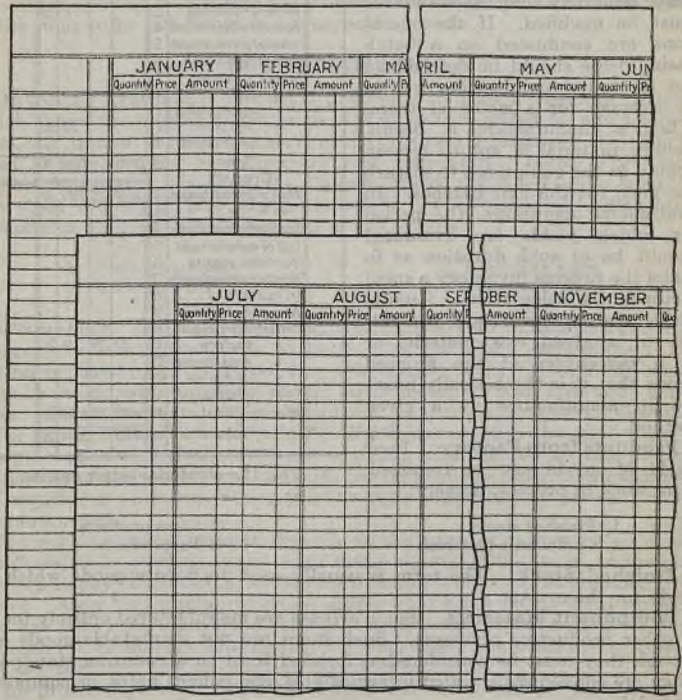


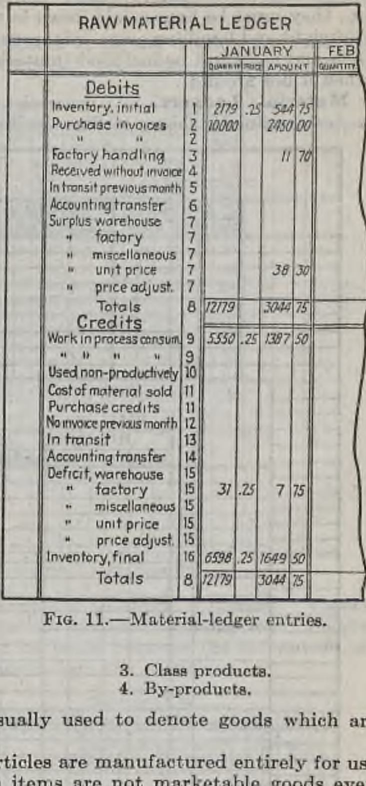
FIG. 10.—Material-ledger sheet.

PRODUCT COSTS

Work in Process. The standard costs for processing and the actual consumption of materials should be charged to an account Work in Process where these values remain as an inventory until such time as finished goods are credited at standard cost. The account must be checked at frequent intervals, otherwise the variation between standard and actual performance will not be known promptly.

Yield Variations. The most important consideration in the chemical manufactures is the yield of finished product obtained from a specified amount of raw material. Differences between standard practice and current performance should be written off as a profit or a loss until a sufficient period has elapsed to prove definitely that the standards must be modified. If the operations are conducted on a batch basis, yields should be determined not for individual batches but as an average for a series of them. When a manufacture is begun, enough material is apt to remain behind in the equipment to disturb the yield for the first batches. In continuous operations, the period for which yields are evaluated should be of such duration as to make the process inventory a small factor in the calculation. Usually yields are expressed as a percentage of a given raw material or as a percentage of the finished goods that it is theoretically possible to manufacture by a given method.

Products from Factory. Four kinds of goods can be removed from work in process, namely:



RAW MATERIAL LEDGER

		JANUARY				FEB
		QUANTITY	PRICE	AMOUNT		QUANTITY
Debits						
Inventory, initial	1	2179	.25	544	75	
Purchase invoices	2	10000		2450	00	
" "	2					
Factory handling	3			11	70	
Received without invoice	4					
In transit previous month	5					
Accounting transfer	6					
Surplus warehouse	7					
" factory	7					
" miscellaneous	7					
" unit price	7			38	30	
" price adjust.	7					
Totals	8	12179		3044	75	
Credits						
Work in process consum.	9	5550	.25	1387	50	
" " " "	9					
Used non-productively	10					
Cost of material sold	11					
Purchase credits	11					
No invoice previous month	12					
In transit	13					
Accounting transfer	14					
Deficit, warehouse	15					
" factory	15	31	.25	7	75	
" miscellaneous	15					
" unit price	15					
" price adjust.	15					
Inventory, final	16	6598	.25	1649	50	
Totals	8	12179		3044	75	

FIG. 11.—Material-ledger entries.

1. Finished stock.
2. Component materials.
3. Class products.
4. By-products.

Finished Stock. The term is usually used to denote goods which are ready for the market.

Component Materials. Some articles are manufactured entirely for use in other productive processes. Such items are not marketable goods even though they must be considered as finished from an accounting standpoint. They are conveniently called intermediates, component parts, or component materials.

Class Products. The distinguishing feature of class products lies in the fact that they are related only through a voluntary association in the manufactures. Both the material and the processing may be joint. The operator can choose the particular ceramics he will bake together in a kiln, the castings he will pour on a particular day, or the stampings he will plate in the same bath. Such articles are called class products because they are manufactured together at one time. The cost of all the operations is allocated on the basis of separate weights, surface areas, volumes, etc.

By-products. When in the manufacture of an article several products result from a splitting up of the original raw material, then the major product is called the finished stock and the other items are called by-products. The refining of petroleum is one of the best known of the by-products manufactures. It is often possible to vary, within certain limits, the ratio between the various end products.

If the cost work is conducted on a standard basis, it is a simple matter to credit the work-in-process account for the major product with the value of by-products. The way the problem is handled depends on the point of view. One extreme is to consider each by-product a salable waste and to credit the major product with all revenue obtained through the sale of by-products. The other extreme is to juggle the values so that each one carries a fair share of the joint costs and brings in a reasonable profit.

COST ESTIMATING

Outlining Problem. The usual practice is to draw a schematic diagram which shows the routing of materials; the quantities added to, and removed from, the process; the general nature of the processes; equipment; etc. Such a chart is sufficient for a preliminary cost calculation. Later, when more precise information is needed, the diagram can be expanded into a complete flow sheet which goes into all minute details. The four parts leading to an estimate of product costs and profits are as follows:

1. Fixed investment.
2. Working capital.
3. Processing costs.
4. Material costs.

Estimating. As the invested value must be known to establish the fixed charges, the usual practice is to begin with the plant calculations. The section on fixed-property accounting will suggest many of the items which must be evaluated. Buildings should be estimated on both area and volume bases. Percentages and ratios should not be used until their applicability to the problem has been ascertained. For instance, with the small units of pharmaceutical manufacture the cost of piping might be 100 per cent or more of the cost of the machinery; and with the large costly units used in some of the continuous processes, piping might not be more than 10 per cent of the machinery cost.

The working capital required depends on the inventories (raw, finished, and in process) to be carried; on the credit terms allowed to the trade; on the cash required to pay current bills; and on the amount of credit to be obtained from others. After the number of days' supply that must be on hand for each item has been ascertained, the value of the inventories, accounts receivable, and cash can be quickly determined. If from these values the accounts payable are deducted, the net working capital is obtained.

The processing costs should be worked out on an annual basis because many items are themselves encountered only once a year. The departmental basis is the only safe procedure to follow in estimating costs, as percentages are often misleading. Suggestions as to proper items of cost will be found under Process Costs. From the annual costs it is a simple matter to get at monthly, daily, per batch, per ton, or per pound costs. Plant operations will not usually be 100 per cent of capacity, and allowance for this must be made in the calculations. The material quantities and yields are facts that are circumscribed by the problem. All materials must be figured laid down in the factory, and allowance must be made for putting them into process.

Product Costs. The cost of the finished article must be figured complete laid down in the warehouse with all charges included except general overhead. Package costs are figured in when the problem requires it—sometimes goods must be figured both packaged and in bulk. These complete costs are frequently called the mill cost.

Storage costs should not be added to the cost of the product beyond the point that such a charge is a normal element in business. For instance, if inventories must be carried for 6 months, then fixed charges on the storage facilities are in reality a part of the normal cost of doing business.

The cost of the investment per annual ton of capacity should be figured, as it gives a good basis for comparison with other manufacturers.

The return on the investment can be established from the current selling prices, the mill cost, the net investment, and the normal operating capacity. If the return is not substantially greater than the rental value of money, the venture is unattractive.

Checking the Estimates. A determination of the probable cost of manufacturing an article is one of these open-ended problems which involves assumptions that must be tested by experience before the true worth of the accepted answer can be established. All assumptions should be listed and closely scrutinized. The reasonableness of deductions must be questioned at all times—whether the immediate purpose be nothing more than to define the upper and lower limits of the prospective cost or whether the analysis is being conducted with a refinement calculated to reveal the precise risks of the undertaking. Great care must be exercised in all estimating. Every possible avenue which affords a check on the work should be used, because the final check through actual operations might involve serious losses.

BUDGETING

Importance of Budgeting. The management of a business enterprise is greatly simplified if all of its activities are budgeted. Cost accounting, according to the standard method, is not possible until all the essential activities of a manufacturing establishment have been budgeted. By means of the budget, a definite goal of accomplishment is established, and, with the mind definitely focused on a particular objective, the objective is more certain of being attained. Budgeting is therefore an important part of cost work and consequently of management activities.

Budget Schedules. In a manufacturing enterprise, the cornerstone of the budget is the sales quota. Product costs, over a period of years, are in part fixed by the amount of goods which actually can be sold. To secure profitable operations, all expenditures must be coordinated with the estimated probable income and this income can be established only by most carefully estimating future sales; *i.e.*, a sales quota is fixed. For maximum utility, this sales quota must be known in terms of individual products and each one must be classified by industries, by territories, by customers, and by salesmen. These schedules must be known in great detail because actual performance should come within 10 per cent of perfect accomplishment. Once the sales quota is fixed, it is a simple matter to continue the schedules on, to cover the detailed departmental activities—provided a suitable cost-accounting technique has been developed.

SECTION 28

SAFETY AND FIRE PROTECTION

BY

H. L. Miner, Manager, Safety and Fire Protection Division, Service Department, E. I. du Pont de Nemours & Company; Member American Society of Mechanical Engineers; Army Ordnance Association; Past-President, National Fire Protection Association; Member of Executive Committee, National Safety Council, American Society Safety Engineers.

P. V. Tilden, B.S. in M.E.; Safety and Fire Protection Division, Service Department, E. I. du Pont de Nemours & Company; Member N.F.P.A.; Army Ordnance Association; American Society Safety Engineers.

G. H. Miller, Industrial Relations Division, Service Department, E. I. du Pont de Nemours & Company.

REVISED BY **H. L. Miner** and **P. V. Tilden**

CONTENTS

SAFETY AND FIRE PROTECTION

INTRODUCTION

Accident prevention and fire protection have come to be so much a part of good business, particularly in recent years, that at least certain fundamental phases of these subjects have a logical place in a handbook of this nature. It is recognized, however, that space will not permit more than a brief treatment of them, and students of safety and fire protection are referred to the two national organizations: the National Safety Council, 20 North Wacker Drive, Chicago, Ill., and the National Fire Protection Association, 60 Batterymarch Street, Boston, Mass., as well as to other organizations and societies, some of which are included in the Bibliography.

Safety and fire protection begin with the design of plants—buildings, equipment, and processes. Because careful attention has been given these matters in the past, the majority of present-day accidents and fires result from what is commonly known as human causes. Plans should always be examined by someone competent to decide upon their safety and fire protection aspects. The object is to remove, as far as possible, the human factor from the hazards by making protection permanent or automatic.

LEGISLATION

Most states and most cities have regulations and ordinances governing safety and fire protection measures for buildings, equipment, and processes, and it is always necessary to examine these to make certain that their provisions are being followed. If the information desired cannot be obtained from the local authorities, the two national organizations previously mentioned might be of some assistance in procuring these regulations and ordinances. Many states now have compensation laws which require that compensation insurance be carried and further require certain reports when accidents do occur. These provisions must of course be followed.

THE INDUSTRIAL PLANT

Location. Many factors, such as raw-material supplies, transportation facilities, and labor supply, must be considered in locating industrial plants. From the standpoint of safety and fire protection, it is important, particularly where corrosive, poisonous, or explosive materials are to be handled or produced, to make certain that the location of the operation does not constitute undue danger to adjacent life and property. State laws and municipal ordinances must be consulted and satisfied.

Arrangement of Buildings. Buildings will first of all be arranged to permit economic production, since it is usually possible to provide safety and fire protection under any given arrangement through type of construction, by fire walls, barricades, etc. However, an effort should always be made to limit the height of factory buildings, to provide good separation—50 ft. or more between important units—and most important, to minimize conflagration areas by definite fire stops. Buildings housing hazardous occupancies should be segregated and isolated or barricaded to prevent life loss and/or extensive property damage. Buildings in which explosives are handled or stored should be separated and barricaded in accordance with the "American Table of Distances," published by The Institute of Makers of Explosives, 103 Park Avenue, New York, N. Y.

The distance to be maintained between regular plant-operating buildings depends largely upon the type of construction; the area and height; impor-

Table 1. American Table of Distances*

Blasting and electric blasting caps		Other explosives		Inhabited building barricaded† ft.	Public railway barricaded,† ft.	Public highway barricaded,† ft.
No. over	No. not over	Lb. over	Lb. not over			
1,000	5,000	....	....	15	10	5
5,000	10,000	....	....	30	20	10
10,000	20,000	....	....	60	35	18
20,000	25,000	....	50	73	45	23
25,000	50,000	50	100	120	70	35
50,000	100,000	100	200	180	110	55
100,000	150,000	200	300	260	155	75
150,000	200,000	300	400	320	190	95
200,000	250,000	400	500	360	215	110
250,000	300,000	500	600	400	240	120
300,000	350,000	600	700	430	260	130
350,000	400,000	700	800	460	275	140
400,000	450,000	800	900	490	295	150
450,000	500,000	900	1,000	510	305	155
500,000	750,000	1,000	1,500	530	320	160
750,000	1,000,000	1,500	2,000	600	360	180
1,000,000	1,500,000	2,000	3,000	650	390	195
1,500,000	2,000,000	3,000	4,000	710	425	210
2,000,000	2,500,000	4,000	5,000	750	450	225
2,500,000	3,000,000	5,000	6,000	780	470	235
3,000,000	3,500,000	6,000	7,000	805	485	245
3,500,000	4,000,000	7,000	8,000	830	500	250
4,000,000	4,500,000	8,000	9,000	850	510	255
4,500,000	5,000,000	9,000	10,000	870	520	260
5,000,000	7,500,000	10,000	15,000	890	535	265
7,500,000	10,000,000	15,000	20,000	975	585	290
10,000,000	12,500,000	20,000	25,000	1,055	635	315
12,500,000	15,000,000	25,000	30,000	1,130	680	340
15,000,000	17,500,000	30,000	35,000	1,205	725	360
17,500,000	20,000,000	35,000	40,000	1,275	765	380
		40,000	45,000	1,340	805	400
		45,000	50,000	1,400	840	420
		50,000	55,000	1,460	875	440
		55,000	60,000	1,515	910	455
		60,000	65,000	1,565	940	470
		65,000	70,000	1,610	970	485
		70,000	75,000	1,655	995	500
		75,000	80,000	1,695	1,020	510
		80,000	85,000	1,730	1,040	520
		85,000	90,000	1,760	1,060	530
		90,000	95,000	1,790	1,075	540
		95,000	100,000	1,815	1,090	545
		100,000	125,000	1,835	1,100	550
		125,000	150,000	1,900	1,140	570
		150,000	175,000	1,965	1,180	590
		175,000	200,000	2,030	1,220	610
		200,000	225,000	2,095	1,260	630
		225,000	250,000	2,155	1,295	650
		250,000	275,000	2,215	1,330	670
		275,000	300,000	2,275	1,365	690
		300,000	325,000	2,335	1,400	705
		325,000	350,000	2,390	1,435	720
		350,000	375,000	2,445	1,470	735
		375,000	400,000	2,500	1,500	750
		400,000	425,000	2,555	1,530	765
		425 000	450,000	2,605	1,560	780
		450,000	475,000	2,655	1,590	795
		475,000	500,000	2,705	1,620	810

---

* By permission of The Institute of Makers of Explosives.

† Barricaded, as here used, signifies that the building containing explosives is screened from other buildings, railways, or from highways by either natural or artificial barriers. Where such barriers do not exist, the distances should be doubled.

tance of the unit to continued plant production; hazard of the processes; the land available on the chosen site for present manufacture and for future expansion; values involved and the available protection. In addition, the extent of the exposure from outside properties not under the control of the plant will affect the distance to be decided upon between that property and the nearest plant buildings.

The following minimum *overhead clearances* should be provided for footways, roads, and railroads:

Table 2. Overhead Clearances

Projects	Clearance	Measured above
Floors, platforms, stairs, and other regular routes of foot travel	6′ 6″	Footway
Roadways—main thoroughfares	15′ 0″	Road
Other roadways	12′ 0″	Road
Broad-gage tracks	22′ 0″	Top of rails
Narrow-gage tracks, outside buildings	10′ 6″	Top of rails
Narrow-gage tracks where passing through barricades	6′ 6″	Top of rails
Narrow-gage tracks inside buildings	6′ 6″	Platform of cars, lorries, and other conveyances

Where present installations of pipe lines, barricades, doorways, valve handles, belt-shifter levers, etc., cannot be rearranged to secure this clearance, their presence should be clearly indicated from both directions of approach.

Side clearance, railroads, and trams are measured from center to center of tracks, or from center of tracks to nearest point of structure.

Table 3. Side Clearances

Projects	Broad gage	Narrow gage with rolling stock, 6 ft. wide	Narrow gage with rolling stock, 4 ft. wide
Between parallel lines of track of same gage*	13′ 0″	8′ 6″	6′ 6″
Between parallel lines of broad- and narrow-gage track*	.....	11′ 0″	10′ 0″
From tracks to buildings and structures other than loading and unloading platforms	7′ 6″	5′ 6″	4′ 6″

* Where tracks are on a curve, allowance should be made for projection of car bodies over track, so that a clearance equivalent to the above is maintained under all conditions of use.

Side clearances for loading and unloading platforms where locomotives are operated are to be measured from center of track to nearest point of platform or structure:

Broad-gage track	6′ 6″
Narrow-gage track with rolling stock, 6 ft. wide	4′ 6″
Narrow-gage track with rolling stock, 4 ft. wide	3′ 6″

On narrow-gage tracks where cars are pushed by hand, it is permissible to bring platforms for loading or unloading, raised transfer tracks, etc., as close as possible to car platforms. As far as practicable, this is to be avoided where cars are operated by locomotives of any description. In all such cases, clearance between car platform and structure, if below the standard, is to be as small as possible and is not to be over 2 in., unless varying width of cars makes this impossible. If platform is above floor of car, front of platform is to be sheathed down to 6 in. below car-floor level. If side clearance from structures, etc., is less than the standard on tracks on which locomotives are operated, signs, "Caution—Insufficient Clearance" (18 by 12 in., red with white letters) should be posted 72 in. above the rails (and at the standard side clearance) at each end of the obstruction or group of obstructions. The distance away from the obstruction at which these signs are to be posted is to be governed by speed conditions. For medium speeds a distance of 30 ft. is recommended.

Table 4. Minimum Width for Footways, Doors, Runways, Etc.

Regular routes of foot travel, inches	36
Passage between machines, inches	24
Safety exits and doorways, inches	28
Gangplanks and runways, inches	30
Runways for two-wheel hand trucks (allowing two trucks to pass), inches	60

Water supplies for plant fire protection should be available from at least two independent sources. The **primary supply** which maintains pressure on the mains should be automatic and of good pressure but may be of limited capacity. One or more connections from a reliable public water system or a gravity tank of 100,000 gal. capacity elevated on a 125-ft. trestle provides a good primary supply. Pressure should be adequate to give at least 15 lb. per sq. in. flowing pressure on highest sprinkler heads, or in their absence at least 65 lb. per sq. in. on yard hydrants. This should be augmented by a **secondary supply.** One or more fire pumps located in an isolated or properly protected fire-resistive pump house with an unlimited suction supply forms a most satisfactory secondary source. Where rivers, ponds, lakes or similar natural water supply are not available, concrete reservoirs or suction tanks of at least 100,000 gal. capacity for each 1000 g.p.m. pumping capacity are necessary. With such reservoirs, adequate filling connections should be provided. A reliable power supply for operating the pumps, whether steam or electric, is of utmost importance and should be given serious consideration, particularly with the view of preventing its interruption during a serious fire. Where a public fire department is available, steamer connections are provided on the plant mains so that city pumpers or fire boats can discharge water at high pressure into plant mains.

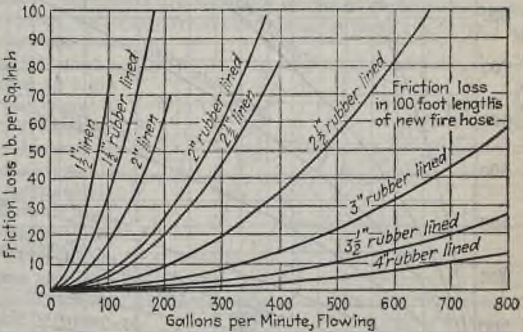


FIG. 1.—Friction loss in fire hose. Curves based on the following assumptions: (1) 14 lb. friction loss per 100 ft. of 2½ in. cotton rubber lined hose with 250 g.p.m. flowing (smooth rubber lining). Where lining is rough or old, friction losses may be 50 to 100 per cent higher. (2) Friction loss varies directly with the square of the quantity flowing. (3) Friction loss varies inversely as the fifth power of the diameters. (4) Friction loss in linen hose is taken from data in "Fire Engine Tests and Fire Stream Tables" published by the National Board of Fire Underwriters.

Fire Mains. Where the plant is of sufficient size, an independent system of underground fire mains supplying sprinkler systems and yard hydrants should be provided. These mains should not supply water for domestic purposes except in the event of an emergency. The size of the mains should depend upon the size of the system, but, in general, mains should not be smaller than 8 in. where hydrants as well as sprinkler systems are supplied. The branches to individual hydrants should be not smaller than 6 in. (For friction losses in mains, see Flow of Fluids section.) Due to its much longer life in underground service, cast-iron pipe Class C or D, American Water

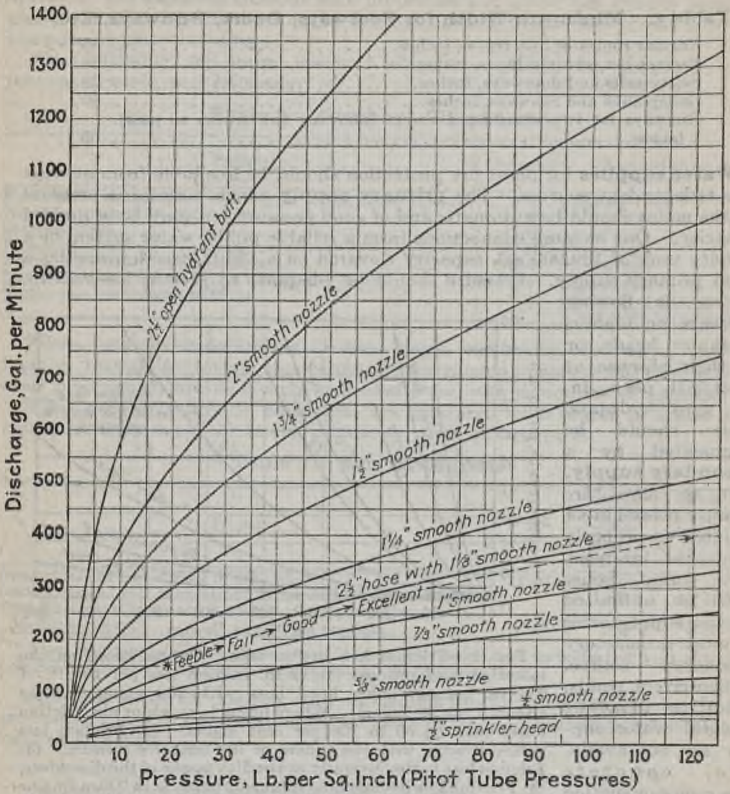


FIG. 2.—Discharge of water through smooth nozzles and sprinkler heads. Assumed coefficients of discharge (0.99 to 0.997) based on experiments by John R. Freeman, *Trans. Am. Soc. Civil Eng.*, **21, 24.** Sprinkler-head data from tests by Underwriters' Laboratories. The 2½-in. open-hydrant butt-curve data from Crosby, Fiske, Forster, p. 713, Table 18, Van Nostrand, 1924—two-way Mathews Hydrant. The 2-, 1¾-, 1½-, 1¼-, 1⅛-, and 1-in. nozzle curve data from Crosby, Fiske, Forster, p. 698, Table 12. The ⅞-, ⅝-, and ½-in. nozzle curve data from "Fire Stream Tables" (pamphlet by Factory Mutuals, Boston). The ½-in. sprinkler-head curve data from Crosby, Fiske, Forster, p. 717.

Works Association Standard (130 lb. or 173 lb.), should be used. Where the ground contains corrosive substances, it is necessary to protect the outside of the pipe by dipping each length in hot tar, wrapping it in tar paper wired tightly in place, and again dipping in hot tar. All joints should be similarly protected. Care must be exercised in laying pipe that no foreign materials, such as stones or pieces of wood, are allowed to enter the pipe as these form serious obstructions. The depth of the mains below the earth surface should be sufficient to prevent the water in them freezing in the coldest weather experienced in that locality. Further information can be obtained from the "Regulations for Outside Protection" published by the National Board of Fire Underwriters, 85 John Street, New York, N. Y.

In order to provide flexibility and to facilitate repairs, sectional control valves are installed in the mains. Check valves are necessary to prevent back flow of water where pressures of the different supply lines vary. State laws must be complied with, and special precautions must be taken where there is a cross connection between public water supplies and a fire-main system which is supplied by non-potable water.

Hydrants and Hose. Hydrants are specially designed to be substantial, to provide an unobstructed waterway from the mains to the outlets, and are easily operated and arranged so that when the hydrant is shut a drain at the base opens to allow the water in the hydrant barrel to drain, thus preventing freezing. Hydrants are usually located so that at least two streams are available for each building without the use of more than 150 ft. of hose. This generally necessitates locating hydrants not more than 300 ft. apart; but where areas are congested, it may be necessary to place them much nearer together. Hydrants are placed not closer than 40 ft. to the buildings which are to be protected. The threads on the hydrant outlets and the threads on the hose should be uniform throughout the plant and should conform to the public standard in the locality. Where possible, threads conforming to the National (American) Standard Fire Hose Coupling Screw Threads should be used (see "A.S.M.E. Code," Engineering and Industrial Standards B-26-1925).

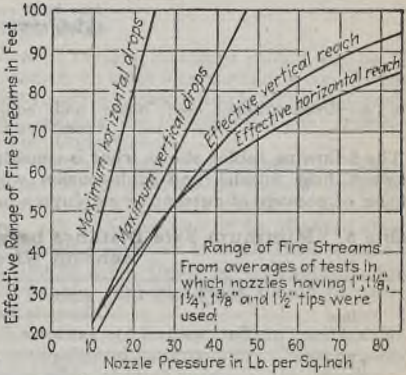


FIG. 3.—Range of fire streams. Nozzle pressures as indicated by pitot tube. Data based on experiments by John R. Freeman, *Trans. Am. Soc. Civil Eng.*, **21**, 303 (1889).

Standard single-jacketed, cotton, rubber-lined mill-yard hose is preferred for outside fire protection on industrial plants, due to its flexibility and relatively low friction loss; 2½-in. hose provided with 1⅛-in. nozzles is considered minimum for outside protection. See the tables on Friction Loss in Fire Hose; Discharge of Water through Smooth Nozzles and Sprinkler Heads; and Range of Fire Streams.

To save valuable time at the start of a fire, the hose is ordinarily connected directly to the hydrant and is kept in hose houses or in hose boxes at the hydrants. These also contain miscellaneous equipment. This hose is supplemented by a reserve supply on hose reels conveniently located throughout the plant yard.

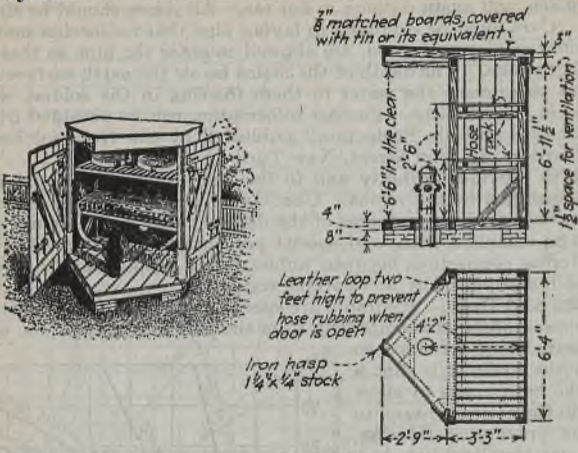


FIG. 4.—Underwriters' hose house and equipment. (*National Fire Protection Association.*)

The following table* shows what is considered to be the minimum distance between hose nozzles and high-tension wires, under which there may be danger of passage of current to employees holding the hose line.

Table 5. Minimum Safe Distance between Hose Nozzles and High-tension Wires

Voltage	Safe distance 1⅛-in. nozzle, ft.		Safe distance 1½-in. nozzle, ft.	
	Fresh water	Salt water	Fresh water	Salt water
1,100	6	25	9	30
2,200	11	25	16	30
6,600	19	30	29	35
11,000	20	30	30	35
22,000	25	30	33	40
33,000	30	35	40	45

Fire-alarm Equipment. Private fire-alarm systems are necessary on practically all plants for promptly notifying the fire brigade and for exit of employees. These systems should operate on normally closed, electrically supervised circuits. Fire-alarm boxes are of three types:

1. Interfering.
2. Positive non-interfering.
3. Positive non-interfering and successive.

The **interfering** type of box has no ability to control its circuit once it has been started in operation; and hence if more than one box is pulled at or about the same time a mixed, a confused or a totally mutilated signal will result.

---

* By permission of the National Fire Protection Association.

The **positive non-interfering** type of box, which is the one most generally used for plant service, is so arranged that should two or more be pulled at the same time a definite, accurate signal will be received from one of the stations.

The **positive non-interfering and successive** type of box is non-interfering as just described but has the added feature that after the box selecting control of the circuit has completed its signal the other box or boxes will automatically take control, one after the other; hence the term successive.

Fire-alarm boxes should be installed in conspicuous locations and mounted so that accidental operation will not be caused by vibration or jarring. When provided inside buildings they should preferably be located near exits. Boxes located outside the buildings should be well detached from the buildings. Gongs may be located throughout manufacturing buildings, automatically coded so that they ring the number of the box pulled. The plant whistle may be automatically or manually operated. Recording punch registers provide visual as well as permanent records of all signals passing over the fire-alarm system. It is good practice to operate a fire-alarm system from storage batteries installed in duplicate sets, one set operating the system while the other set is in reserve or is being charged. The system should be tested daily by actually pulling at least one box. Every box on the system should be operated once a month.

BUILDINGS

Type of Construction. Permanent factory buildings should not be of frame construction except where necessary to meet manufacturing conditions. Non-combustible construction or fire-resistive construction is to be preferred. For further details of construction, consult the "Building Code" recommended by the National Board of Fire Underwriters.

Outside Walls (Buildings Other than of Frame Construction). Where distance between buildings is limited by uncontrollable factors, and the various buildings expose one another, (1) door openings should be protected with standard automatic fire doors. (2) Windows should be of wired glass in approved steel sash without ventilating sections or with these sections arranged to close automatically at time of fire. Under severe exposure, fire shutters or cornice sprinklers are advisable. (3) Fire walls should be parapeted to prevent spread of fire through roofs.

Horizontal Cut-offs. Where floor areas are large, or where combustible construction is involved, fire-wall cut-offs should be provided. These cut-offs should be standard fire walls of at least 13 in. of brick or of equal construction, with all door openings properly protected with single or double automatic fire doors. All other necessary openings through fire walls, such as belt, shaft, or conveyor openings, should be suitably guarded to prevent spread of fire. The fire walls, where roofs are combustible, should extend through the roofs and should be parapeted to a height of 3 ft. Where side walls of the building are combustible, the fire walls should be extended out several feet beyond the exterior walls.

All new interior partitions should preferably be fire-resistive or at least of non-combustible construction.

The following chart shows the relative fire resistances of various types of wall and partition construction.

Vertical Cut-offs. In multiple-story buildings, each floor should preferably be entirely cut off from the other floors. Where it is necessary for stairways, elevators, or dumbwaiters to extend through floors, enclosures should be provided at least equivalent to the existing floor construction of the building. All openings in these enclosures should be equipped with automatic fire doors.

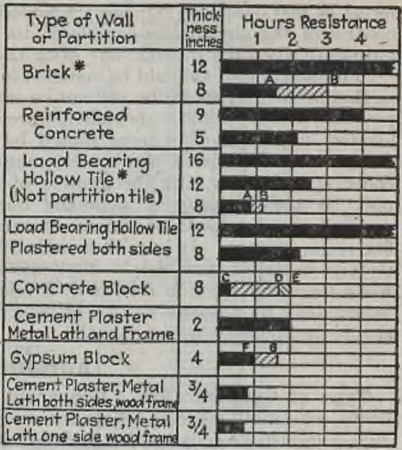
All floors should be maintained in a good state of repair, and when interior changes are made, which in any way affect vertical cut-offs, proper regard for the spread of fire should be considered.

Elevators. Elevators are a frequent source of accidents, and experience has shown that certain safety requirements must be met in their construction. Consideration must be given to top clearances, bottom clearances, overtravel, overhead structures, car and shaft-way gates, car speeds, emergency stopping devices, and general guarding. Experience has demonstrated the value of the elevator as a life-saving device in case of fire, and it is therefore important from this standpoint that the shaft way be of fire-resistive construction in passenger elevators.

Safety requirements for elevators, hoists, dumbwaiters, and escalators are fully covered in the "Safety Code for Elevators," American Society of Mechanical Engineers, New York, to which reference should be made for detailed information.

Exits. While the following covers the subject in a general way, reference should be made to the "Building Exit Code," the National Fire Protection Association, for information in greater detail.

Table 6. Fire Resistance of Walls and Partitions*



* Three-quarter-inch cement or gypsum plastering on one side of a brick or tile wall will increase the fire resistance by ½ hr.

Fire-resistance Limits (see chart above):

A. When combustible members frame into wall.
B. When framing is noncombustible.
C. For bearing walls due to characteristic splitting along center of wall.
D. For non-bearing wall.
E. For cinder concrete, load-bearing, or non-load-bearing wall.
F. For unreinforced partition.
G. For partition reinforced with steel rods in joints.

(*Compiled from test data of Bureau of Standards and Underwriters' Laboratories, 207 E. Ohio St., Chicago.*)

It is important that exits be of ample capacity to accommodate safely and properly the employees in case of fire and panic and to enable them to reach a place of safety outside of the building in which they are working. Every factory or working space should have at least two means of egress from each story. These should be so located and constructed that the occupants will use them if a fire occurs, as a matter of course—preferentially.

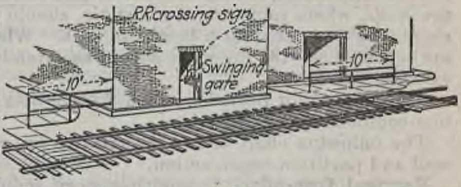
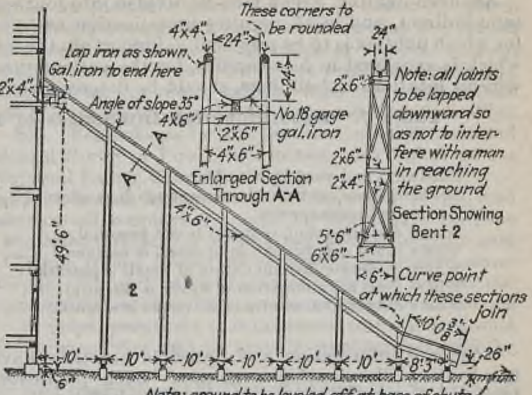


Fig. 5.—Arrangements of guards for door openings.

In planning exits for new and old buildings, such factors as the following should be thoroughly considered: (*a*) number and location; (*b*) types; (*c*) construction characteristics; (*d*) maintenance; (*e*) drills.

As far as practicable, doors should not be placed so as to open directly upon moving machinery, electrical equipment, open containers of hot or corrosive liquids, pits, railroad or tram tracks.

Where doorways open abruptly upon railroad, tram, telpherage, lorry, or crane tracks, or at crossing of routes or paths with these, where view of approaching cars, etc., is obstructed, provisions are to be made momentarily to stop pedestrians (see Fig. 5).



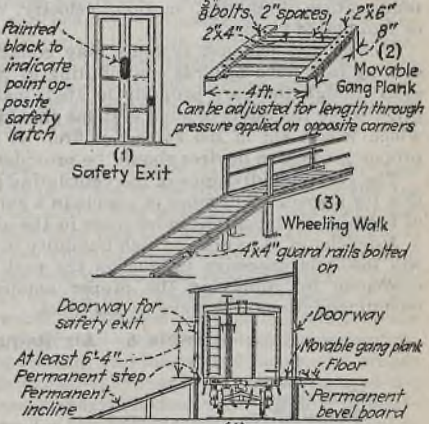
Note: ground to be leveled off at base of chute to afford an unobstructed path

FIG. 6.—Emergency exit chute.

Doorways from which there is a vertical drop of 4 ft.-0 in. or more are to be provided with a hinged gate to open inward or sliding gate, of strength equivalent to the standard railing, not less than 42 in. in height with midrail.

Where quick egress from hazardous operations is necessary, emergency chutes and doors should be installed. See Fig. 6 and Fig. 7 (1) and (4).

Lighting (see also pp. 2264 to 2270). Certain factors which have been found to affect the eyes must be considered in the design and installation of lighting sources. Among these are reflection, absorption, distribution, diffusion, intensity, steadiness, color, and glare. Among the controlling practical factors are: height, spacing, and size of lamps and type of reflectors or globes.



Arrangement of Emergency Exit Through Box Car Where Necessary to Provide in Special Cases

FIG. 7.—Arrangements for exits, gangplanks, and wheeling walks.

Adequate daylight illumination, properly applied, is the ideal light. Light from above is generally better than light from side windows only. Skylights and monitor windows should be provided wherever practicable. Large window areas, equipped when necessary with awnings, window shades or blinds, and diffusive or refractive glass, together with light interiors, are desirable.

Artificial lighting, which may be divided into four systems, direct, indirect, semi-indirect, and local, requires investigation as to the nature of the work for which lighting is to be provided. First of all, the intensity of illumination, which is expressed in foot-candles, for the various types of work and for the various locations in buildings, should be determined.

Table 7. Recommended Intensities of Illumination

	Recommended Foot-candles
Roadways and yard thoroughfares	0.5– 2
Storage spaces; aisles and passageways in work rooms excepting exits and passageways	2 – 5
Where discrimination of detail is not essential	2 – 5
Where slight discrimination of detail is essential	2 – 5
Where moderate discrimination of detail is essential	5 –10
Where close discrimination of detail is essential	5 –10
Where discrimination of minute detail is essential	10 –20 and above

Adequate information and desirable practices have been formulated by the leading illuminating engineers. Such publications as the "Code of Lighting: Factories, Mills, and Other Work Places," published by the Government Printing Office, Washington, D. C., should be referred to when investigating lighting requirements.

Ventilation. In considering the subject of ventilation, the following factors must be taken into account: amount and distribution of air supply, temperature, humidity, motion, velocity, odors, dust, bacteria, and toxic or flammable vapors.*

Blower and exhaust systems may constitute fire hazards in themselves or may introduce hazards contributing to the causes and the spread of fires, even though they are properly located, installed, and safeguarded. Where practicable the ducts of such systems should not pierce floors, fire walls, etc., which would aid in the spread of fire. If such installations are necessary, proper protective devices should be provided at these cut-offs.

The two main divisions of the ventilating problem are: first, proper ventilation for a specified number of people in a confined space; second, the removal of harmful or dangerous substances in the air.

A deficiency of oxygen or high humidity in a room where people are working will result in excessive fatigue on the part of the occupants. This can be overcome by supplying the proper amount of fresh air. The following recommendations will serve as examples:

Table 8. Air Requirements

Project	Renewals of Air, Hour
Work shops	10–15
General offices	4– 5
Engine rooms	10–20
Boiler rooms	10–20
Stairways and halls	1– 2
Mills (light work)	4– 6

Whether the air be supplied by windows, a forced-draft system, a suction system, or a combination of these, depends on the conditions involved.

---

* Used here as recommended by The National Safety Council.

If harmful or flammable dusts or vapors are present in the air, it is essential to keep them below the harmful or hazardous concentration for the particular dust or vapor in question.

The means of ventilating an enclosure where harmful or flammable dusts or vapors are present will usually consist of a hood with a suction fan placed in the vortex constructed over the source of the dusts or vapors, to keep them from escaping into the room. Where this is not possible, ventilating ducts are distributed throughout the room, usually with the suction duct being near the floor when the vapors are heavier than air and near the ceiling when lighter than air. See "Regulations for the Installation of Blower and Exhaust Systems," National Board of Fire Underwriters, also *Safe Practices Pamphlets* 32 and 37, National Safety Council.

Sanitation. Change houses with lockers, wash rooms, lunch rooms, and toilet facilities adequate for all employees should be provided. Hygiene among workers is an important factor in combating health hazards and makes for efficiency. Change rooms should be separated or located away from the working operations. Shower baths should be provided for occupations where workmen necessarily become dirty or are exposed to poisonous dust, fumes, or vapors. In some operations, it is necessary to provide special working clothes and also to supervise and to provide washing facilities for this clothing.

To insure comfortable and desirable conditions, it is essential to have all locker rooms, lunch rooms, showers, and toilets adequately heated, ventilated, and illuminated. Cleanliness is necessary in these places, and attention should be given to their maintenance.

A drinking-water system with bubbling fountains located at convenient places should be installed throughout the plant. When taking water from a city drinking-water supply, it can generally be assumed that the water is properly protected against disease germs; but if taken from other sources, it is essential to make frequent chemical and bacteriological analyses. The water should be kept at a constant and desirable temperature, approximately 45°F., and this is best accomplished by providing a centralized refrigerating and cooling plant. Modern electrical refrigerating units can be used if such systems are warranted.

Automatic Sprinkler Systems. The automatic sprinkler is the best single safeguard against loss of life or property by fire. It is a device for automatically discharging water on a fire, and with proper water supplies will either extinguish the fire or hold it in check. The automatic feature is secured through the heat from the fire which melts a soft solder compound and allows the head to operate. In the case of the bulb type of sprinkler, operation is secured through bursting of the bulb owing to expansion of the liquid which it contains. The water is supplied by a piping system, the sprinklers being at intervals along the pipe. Sprinkler-system design requires expert knowledge to secure proper spacing and location of heads, the proper pipe sizes, control valves, and adequate water supplies. All portions of the building should be protected. Sprinkler installations should follow present-day standards. (See "Regulations for the Installation of Sprinkler Equipments," National Board of Fire Underwriters.) These regulations require adequate water supplies, standard and approved material, heads properly spaced and located under various types of ceiling construction, and piping properly supported.

Automatic sprinklers are designed to operate at various fixed temperatures, their selection depending upon the normal temperature of the space to be protected as follows:

Table 9. Operating Temperatures of Automatic Sprinklers

Room temp., °F.	Rating of sprinkler heads, °F.		
	Solder type	Quartz bulb	
Under 100	155–165	135	Ordinary
100–150	212	175	Hard or high test
150–200	286	225	
Above 200	360	325	

When corrosive vapors are present, special heads, such as chromium-plated, lead-plated or wax-covered (corroproof) heads, or a combination of these or the bulb type should be employed. Where the heads are subject to mechanical injury they are provided with guards.

When the space to be protected is heated, a wet system with water in the pipe lines under pressure to the sprinkler heads at all times is installed. When installed in unheated spaces, a dry system is used with water held in check by a dry valve located in a small heated enclosure, the sprinkler piping being filled with air. When a sprinkler head opens, the air pressure is reduced. The dry valve then operates and water flows into the system.

Unsealed sprinkler heads (ordinary heads with struts or links removed) may be used in conjunction with quick-opening or deluge valves to discharge large quantities of water over an entire area or at certain critical points. **Cornice or window sprinklers** may be used for producing a water curtain on the outside of buildings to prevent the spread of fire. Unsealed heads, and cornice or window sprinklers, may be used in addition to standard automatic sprinkler systems. For quantities of water discharged from automatic sprinkler heads, see Fig. 2.

Inside Hose. Linen hose is usually used for such equipment, although in buildings containing acid fumes, or where there is excessive moisture in the air, rubber fire hose may be necessary. Small hand hose can be attached to wet pipe sprinkler systems under conditions specified in "Regulations Governing the Installation of Sprinkler Equipments," National Board of Fire Underwriters. In most cases inside hose is either 1½ or 1¼ in. with ½-in. brass nozzles and stored on semiautomatic hose racks or so located that all parts of each story of the building are within 20 ft. of a nozzle for a first-aid stream when attached to not more than 75 ft. of hose. For quantities of water discharged from inside hose, see Fig. 2.

First-aid Fire Extinguishers. These are designed for incipient fires and should be installed in addition to other equipment, such as sprinklers and inside hose. Incipient fires are divided into three groups:

Class A fires are those in ordinary combustible material, where the quenching and cooling effects of quantities of water or solutions containing a large percentage of water are of primary importance.

Class B fires are those in oils, greases, flammable liquids, etc., where the blanketing or smothering effect of the extinguishing agent is of greatest importance.

Class C fires are incipient fires in electrical equipment, where the nonconducting property of the extinguishing agent is of prime importance.

A unit has been adopted for convenience in measuring fire protection afforded by first-aid fire appliances. This unit is composed of from one to five appliances, depending upon the extinguishing value of the kind of appliance comprising the unit. For ordinary combustible occupancies and extra hazardous occupancies at least one unit should be provided for every 2500 sq. ft. of floor area or not over 50 ft. of travel from any point to reach the nearest unit. For light hazard occupancies one unit should be provided for every 5000 sq. ft. or not over 100 ft. of travel from any point to reach the nearest unit.

Table 10. First-aid Fire Appliances' Classification

Kind of extinguisher	Classification and number of devices	Protection from freezing required	Recharging
1¼ gal. soda-acid	A-2	Yes	Renew yearly
1½ gal. soda-acid	A-2	Yes	Renew yearly
2½ gal. soda-acid	A-1	Yes	Renew yearly
17 gal. soda-acid (wheeled)	A-1	Yes	Renew yearly
30 gal. soda-acid (wheeled)	A-1	Yes	Renew yearly
33 gal. soda-acid (wheeled)	A-1	Yes	Renew yearly
40 gal. soda-acid (wheeled)	A-1	Yes	Renew yearly
60 gal. soda-acid (wheeled)	A-1	Yes	Renew yearly
80 gal. soda-acid (wheeled)	A-1	Yes	Renew yearly
1¼ gal. foam	A-2; B-2	Yes	Renew yearly
1½ gal. foam	A-2; B-2	Yes	Renew yearly
2½ gal. foam	A-1; B-1	Yes	Renew yearly
5 gal. foam	A-1; B-1	Yes	Renew yearly
17 gal. foam (wheeled)	B-1	Yes	Renew yearly
33 gal. foam (wheeled)	B-1	Yes	Renew yearly
2½ gal. calcium chloride	A-1	No	Renew yearly
2½ gal. pump tank	A-2	Yes	Keep full
5 gal. pump tank	A-1	Yes	Keep full
22 gal. bucket tank containing 5 standard fire pails	A-1	Yes	Keep full
50 gal. cask with 3 standard fire pails	A-1	Yes	Keep full
12 qt. standard fire pail	A-5	Yes	Keep full
12 qt. foam fire pail	A-5; B-5	Yes	Renew yearly
12 qt. sand fire pail	A-5; B-5	No	Keep full
1 gal. loaded stream	A-2	No	Renew yearly
1¾ gal. loaded stream	A-1; B-2	No	Renew yearly
1 qt. carbon tetrachloride	B-2; C-2	No	Keep full
1¼ qt. carbon tetrachloride	B-2; C-2	No	Keep full
1½ qt. carbon tetrachloride	B-2; C-2	No	Keep full
2 qt. carbon tetrachloride	B-2; C-2	No	Keep full
1 gal. carbon tetrachloride	B-2; C-1 or B-2; C-2	No	Keep full
2 gal. carbon tetrachloride	B-2; C-1 or B-2; C-2	No	Keep full
3 gal. carbon tetrachloride	B-2; C-1 or B-2; C-2	No	Keep full
7½ lb. carbon dioxide cylinder	B-2; C-1	No	Weigh semi-annually
10 lb. carbon dioxide cylinder	B-2; C-1	No	Weigh semi-annually
15 lb. carbon dioxide cylinder	B-1; C-1	No	Weigh semi-annually
20 lb. carbon dioxide cylinder	B-1; C-1	No	Weigh semi-annually
50 lb. carbon dioxide cylinder (wheeled)	B-1	No	Weigh semi-annually
100 lb. carbon dioxide cylinder (wheeled)	B	No	Weigh semi-annually

Extinguishers are designed for certain types of fires, as shown in the table above. This table also shows: (*a*) the number of devices required to form one unit; (*b*) whether the device needs protection against freezing; and (*c*) the necessary frequency of recharging. For further details, see "Regulations Governing First Aid Fire Appliances," National Board of Fire Underwriters.

EQUIPMENT

Type. In considering the type of equipment from the standpoint of safety, particularly common equipment or that which is not peculiar to certain processes, the principal considerations are that it be properly guarded, free from projecting shafts, exposed moving parts, and projecting setscrews. Machines which are automatically fed should be given preference, and the point of operation should in all cases be adequately guarded. Rolls and other such equipment where there is the possibility of the employes being injured between moving parts should be provided with quick-stopping devices either automatically or manually controlled. Remote controls are frequently used for machines in which explosive substances are handled. In all cases, suitable means should be provided for adequate cleaning without undue hazard to the employees. Pressure vessels should be installed in accordance with the "Boiler Construction Code," American Society of Mechanical Engineers. Further information on this subject will be found on pages 1804 to 1806 and in the *Safe Practices Pamphlets*, National Safety Council.

Location. Equipment should be located to provide adequate working areas, aisles, and light. It is advisable to locate machines upon which work requiring good light is performed along the walls near windows where daylight may be secured.

Guarding. The following section divided into two parts, guarding of general machinery, and protection against falling and slipping, covers the subject in a general way. Further information may be obtained from the *Safe Practices Pamphlets*, National Safety Council; and "Safety Code for Mechanical Power Transmission Apparatus," Government Printing Office.

Guarding of General Machinery

Alteration or Replacement of Existing Protective Appliances. These recommendations are not to be construed as requiring alteration or replacement of present protective appliances except under the following conditions:

a. Where the protection at present afforded is not adequate and is clearly less than that afforded by the standard.

b. Where present protective appliances fall below the requirements of the state.

c. Where present protective appliances are in need of general repair or replacement.

Operation without Protection. New and existing machinery is not to be put into operation until adequate permanent or temporary protective appliances are in place. If necessary to operate machinery for adjustment temporarily without guards, warning signs to be displayed. Defective guards are to have warning signs displayed and to be repaired or replaced as soon as possible.

Strength. To be of substantial and durable construction. No guard or railing to be installed or kept in use which can be broken, collapsed or displaced by the weight of a man's body lurching violently against it.

Material. To be of metal, except under the following conditions when wood may be employed:

a. Existing wooden guards which are substantial and otherwise satisfactory.

b. Temporary conditions, where, on account of the uncertain future of the operations, impossibility of immediately applying metal guards, etc.

c. In buildings containing explosives, etc., which are readily ignited or exploded by spark, and in which the addition of metal is to be avoided.

d. In situations where, on account of the peculiar nature of the process, iron is objectionable or is subject to rapid deterioration.

Wood is not to be used for protection against belt breakage where likely to be splintered and thrown, nor for any but temporary installation in buildings where its use materially increases the general fire risk, particularly in forming a flammable connecting link by which flame may travel from the floor to a flammable roof or ceiling.

Provisions for Oiling and Greasing. Guards to be applied so that it will be unnecessary to enter, open, or remove them for routine oiling. Except where unavoidable, closed guards shall not enclose bearings, and as far as practicable outboard bearings on shafts should be outside filled railings. It is permissible to install extension oil pipes (not smaller than ½-in. standard pipe) through guards, but preferably only under the following conditions:

a. In heated buildings.
b. In buildings not containing explosives.
c. On low- and medium-speed drives.

Where necessary for the lubrication of gears, an opening not over 6 in. square may be left in the guard away from the inrunning side of the point of mesh, if provided with an attached shutter or cover closing by gravity or springs.

Under exceptional conditions, where access to oil and grease cups, etc., while machinery is in motion, is essential and can be secured in no other way, closed guards may be hinged; or railings and closed guards may be provided with a hinged section. In either case these are to be held closed by a latch. If this provides access to a moving part, the opening should be ample in size and should be situated at the point of travel where least danger is introduced. In this case, covers or gates should not be self-closing. No gates or removable section for employes to enter railed enclosures to do routine oiling while machinery is in motion are permissible. Gates provided for repairs and adjustments must be locked.

Oiling Walkways and Platforms. The oiling of overhead shafting by means of portable ladders is to be avoided as far as practicable by the installation of platforms or walkways.

Platforms and walkways should be at least 30 in. in width and are not to be less than 24 in., protected on open or exposed sides with standard railings. As a general rule, the best location for oiling walkways and platforms is 24 to 48 in. below the level of the shafting.

Oiling walkways and platforms are not to be located more than 24 in. (vertical distance) above the shafting for which they are provided or more than 30 in. (horizontal distance) to one side—for best distances see Table 17.

Material and Construction. Sheet or Perforated Metal Guards. Not recommended for very large guards or filling in railings on account of obstructing view of machinery. Should be galvanized or painted.

a. Gage. In no case, less than No. 22 (0.0313 in.) to be used.

When Largest Side of Guard Does not Exceed 4 Sq. Ft. in Area. (1) Not lighter than No. 22 gage to be used if constructed on supporting framework. (2) Not lighter than No. 16 gage (0.0625 in.) to be used if not constructed on supporting framework.

When Largest Side of Guard is 4 Sq. Ft. in Area or Greater. (1) To be constructed on supporting framework. (2) Not lighter than No. 18 gage (0.05 in.) to be used. (All specifications for U. S. Standard Gage.)

b. Seams and Joints. To be welded, riveted or bolted.

c. Edges and Corners. No exposed sharp edges or corners.

d. Framework. To be constructed of strap iron not less than 1 by $\frac{3}{16}$ in., or angle iron not less than 1 by 1 by ⅛ in. made up by welding, rivets, or bolts. If bolts are used, corners of large guards to be braced with angles or gussets. Strength of framework to be proportioned to size of guard.

Expanded Metal Guards. Expanded metal is recommended for constructing closed guards for pulleys and large gears and for filling in railings.

a. Gage and Mesh. (1) Guards within 4 in. of nearest moving part: No. 18 gage, ¾-in. mesh. (2) Guards not included in (1): No. 13 gage, 1½-in. mesh.

b. Seams and Joints. See Fig. 8 for permissible methods.

c. Edges and Corners. No exposed edges or corners where metal has been cut to be left unprotected (see Fig. 8, for permissible methods). Roughest side of metal to be placed inward. All metal to be dipped or painted.

d. **Framework.** All guards to have supporting framework of not lighter than 1¼-in. iron pipe; not lighter than 1 by 1 by ⅛-in. angle iron; not lighter than 1 by 3/16-in. strap iron. Angle and strap-iron frames to be made up by welding or with rivets or bolts. If bolts are used, corners of large guards to be braced with angles or gussets. The strength of framework to be proportioned to size of guard.

See Fig. 8, (3), (5) and (6) for methods of fastening expanded metal to framework.

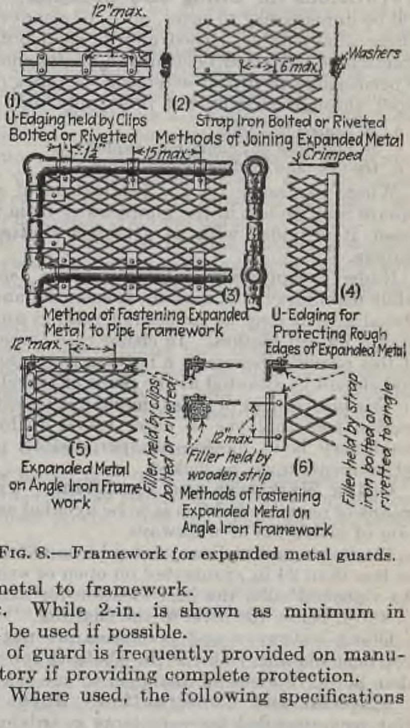


FIG. 8.—Framework for expanded metal guards.

e. Clearance from Moving Parts. While 2-in. is shown as minimum in Tables 14, 15, and 16, 4-in. should be used if possible.

Cast-iron Guards. This type of guard is frequently provided on manufactured machinery and is satisfactory if providing complete protection.

Woven Wire-mesh Guards. Where used, the following specifications to be observed:

a. Guards within 4 in. of nearest moving part: ½-in. mesh, wire not smaller than 0.063 in. diameter.

b. Guards not included in (*a*): 1-in. mesh, wire not smaller than 0.105 in. diameter.

Should be galvanized or painted. General construction to conform with that specified for expanded metal.

Wooden Guards. *a. Material.*

Distance of Guard from Nearest Moving Point, In.	Construction
6 or less	Solid panels, not less than ¾ in. thick.
Over 6	Solid panels not less than ¾ in. thick or slats not less than ⅞ by 2 in. with not over 1½ in. between slats and not over 4 ft. 0 in. unsupported length.

As far as practicable to be of sound, straight-grained stock, free from knots and cracks, and all exposed surfaces dressed.

b. Construction. No nails to be used for securing removable permanent guards to structure or machine. Where explosive dusts are present, tongue-and-groove boards should not be used and all seams and cracks should be filled.

c. Edges and Corners. Edges and corners to be beveled.

d. Framework. The use of a framework is recommended for paneled guards and is necessary for slatted guards. Stock should be not less than 1½ by 1½ in. or equivalent for small guards. Where metal is not objectionable, the use of angle braces for reinforcement and for securing to structure or machine is recommended.

e. Painting. Permanent wooden guards exposed to acid fumes should receive at least two coats of acid-resisting paint.

Method of Securing to Structures, Machines, Etc. Guards protecting parts which must be frequently repaired or adjusted to be applied so that they are readily removable and readily replaceable. Such guards to be strong enough to retain their shape when removed. If sheet-metal or built-up metal guards for this purpose have their edges fastened to the structure, machine, etc., these edges must be reinforced with angle or strap iron for their full length.

Nails or staples not to be used for attaching metal guards. Use bolts, screws, or sockets. No unreinforced sheet metal is to be used for supporting guard unless washers are placed under heads of bolts and screws.

Table 11. Design of Railing Guards. (*a*) **Material and Sizes**

Project	Minimum requirements		
	Pipe railings	Angle-iron railings	Wood railings
Height of top rail above floor	42 in.	42 in.	42 in.
Height of center of mid-rail above floor	21 in.	21 in.	21 in.
Spacing of posts or stanchions	8 ft.	8 ft.	8 ft.
Size of posts	1¼-in. pipe	2 × 2 × ¼ in.	2 × 4 in.
Size of top-rail	1¼-in. pipe	2 × 2 × ¼ in.	2 × 4 in.
Size of mid-rail	1¼-in. pipe	1¼ × ⅜ in. (strap iron)	1 × 6 in. (full) or 2 × 6 in.

The use of railings without mid-rail is not permissible.

Toeboards. Iron toeboards to be ¼ by 6 in. on edge; wood toeboards to be 1 by 6 in. on edge; concrete toeboards (curbing) to be at least 4 in. wide by 6 in high, bonded in.

Where toeboards are objectionable from their tendency to collect snow or ingredients, they may be raised a maximum of 1 in. above the platform.

b. Pipe Railings. Screwed fittings to be used. Where subject to rough usage, as on trestles, posts should be 2 in.

c. Angle-iron Railings. All joints riveted, bolted or welded. The use of angle iron where expanded metal is used as filler is recommended as preferable to iron pipe.

d. Wood Railings. Material to be sound. Hand and mid-rails to be dressed on all four sides, and edges of hand rails beveled. Hand rails to be set with longer dimension horizontal. Posts to be set with longer dimension perpendicular to hand rail. Where subject to rough usage or where iron pipe or cable is used for hand or mid-rails, posts to be at least 4 by 4 in.

Method of Securing to Structures. Pipe and angle-iron railings to be secured as follows:

a. To wood by flanges or angles secured by lag or through-bolts.
by sockets at least 4 in. deep, securely fastened down by flanges.
by socketing in holes bored through timbers not less than 4 in. thick, provided that a flange or similar device is employed to keep the post from slipping down.

b. To concrete by setting in the concrete at least 6 in. deep when poured.
by drilling at least 6 in. deep and grouting with cement grit.
by flanges secured through multiplex floors by bolts.
by sockets at least 6 in. deep set in or grouted into the concrete.

c. To brickwork by flanges secured by through-bolts.
by drilling to a sufficient depth and setting in pipe or flange with anchor bolts and concrete.

d. To metal work by flanges with through-bolts or rivets.
by sockets at least 4 in. deep, bolted or riveted.

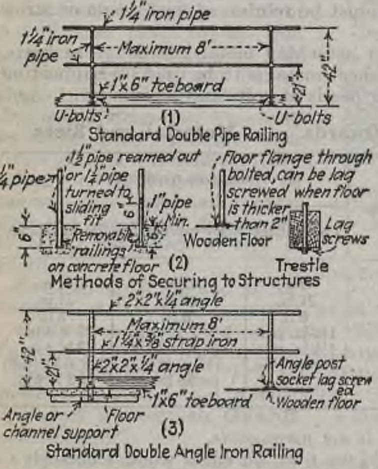


FIG. 9.—Methods of securing railing to structures.

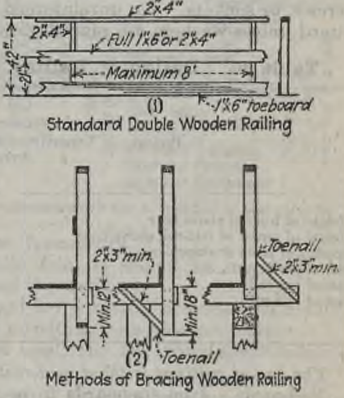


FIG. 10.—Methods of securing wood railings to structures.

The Following Methods Are Not to Be Employed. Flattening pipe, drilling, and bolting; fastening flanges with nails or wood screws, with bolts set in lead, with screws set in wooden plugs, or with expansion bolts.

Wooden railings are to be secured as follows:

By nailing only where at least 12 in. bearing surface on cross-timbers may be secured. See Fig. 10.

By nailing where at least 4 in. bearing surface on cross-timbers may be secured together with one diagonal bracing on each post. See Fig. 10.

By toe nailing only where post may be rigidly braced. Post bracing must not project into footways.

Fillers. Expanded metal, sheet and perforated metal to be used for filling in railings. Method of attachment is shown in Fig. 8 (3), (5), and (6). U edging is recommended for binding the edges of expanded metal.

Toeboards (as protection for machinery). To be used where a slip or projection of the foot under the railing would bring about a close proximity to moving machinery and around the edges of belt pits and belt openings.

Guards for Belts and Pulleys

General Protection. All belts and pulleys within 7 ft. 0 in. of ground, floor, platforms, stairs, and walkways to be protected against contact.

Minimum requirements for guarding are given in:

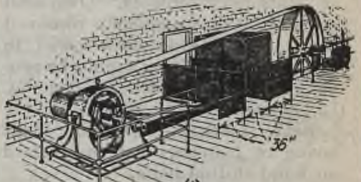
Table 14. Guarding Specifications for Simple Vertical Belt Drives.
Table 15. Guarding Specifications for Simple Inclined Belt Drives.
Table 16. Guarding Specifications for Simple Horizontal Belt Drives.

Where more than one drive is protected by the same railing or closed guard, the protection must conform to the minimum requirements for any of the enclosed drives.

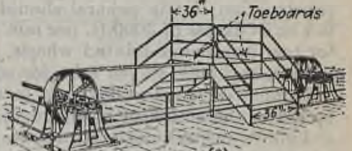
Passage through Belts. To be prevented by installation of railings or closed guards. Where absolutely essential, a railed passageway not less than 36 in. wide between upper and lower run of belt may be provided, to have solid floor and to be completely closed with expanded metal on sides and top and carried out to side railings. Side railings along belt must be filled with expanded metal from floor or toeboard to top of railing for a distance of 36 in. on each side of passageway, if within 36 in. of belt. See Fig. 11 (1).

Passage over Belts. To be avoided as far as practicable. Where unavoidable, double railed steps at least 36 in. wide with solid riser or backing, leading up to a double railed platform at least 36 in. long to be provided over belt. In the case of belts 18 in. in width or over, railings to be filled in (expanded metal, etc.); otherwise toeboards to be installed. See Fig. 11 (2).

Protection against Belt Breakage. Belt barriers for very high speed belts not generally recommended because of the probability of their being thrown. Where used, to be firmly anchored to floor only.



(1) Passage through Belt



(2) Passage over Belt

FIG. 11.—Arrangement of passages through belts and over belts.

Routes* and stations† as well as important control centers, such as switchboards and grouped steam valves, and, in general, any equipment the break-

---

* Refers to general paths, passages, walkways, exits, stairs, ladders or aisles used frequently by building employes and others.

† Refers to regular working stations of employes where they are apt to remain for some time—including work benches, desks, lunch tables, lavatories, bulletin boards and similar congregating places.

ing of which would cause a serious accident should not be located under or in direct line with heavy high-speed belts. Where this is unavoidable, the following protection to be applied:

Belt width	Belt speed	Overhead protection	End protection
Up to and including 8 in. or over 18 in.	Any	None	None
8 to 12 in., inclusive	3000 ft. per min. and over	Continuous belt trough	Belt barriers
Over 12 in., and up to and including 18 in.	Any	Continuous trough	Belt barriers

Belt barriers to consist of structural steel with vertical members extending from floor to not less than 24 in. above run of belt. Maximum height of barrier, 84 in. Barriers to be securely braced or stayed.

Belt Clearance. A clearance exceeding the width of the belt by one-fifth to be provided between edge of pulley and structures and between edge of pulley and bearings or stationary objects. Overhead belts that are frequently removed from the pulleys and allowed to hang on the shafting should be provided with belt perches. See Fig. 12 (1) and (2).

Belt Lacing. Metallic belt lacing or fasteners not to be used on hand-shifted belts.

Pulley Speeds and Maintenance. Maximum permissible rim speed of pulleys or flywheels varies with the material used and type of construction but in general should not be in excess of 3000 ft. per min. for cast-iron flange-jointed wheels, 4700 ft. per min. for cast-iron link-jointed wheels, and 6000 ft. per min. for solid cast-iron wheels.

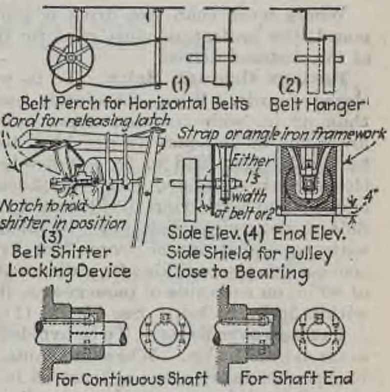


Fig. 12.—Belt perches, clearances, and side shields.

Laminated wood pulleys not to be installed or run where continually subjected to the action of moisture.

Pulleys not regularly in use to be removed from shafting, etc., which is in continued operation.

The use of short, heavy drives with adjustable belt tightener pulleys to be avoided wherever practicable.

Belt guides of any character (except special pulleys designed for the purpose) not to be installed for the purpose of keeping belts on pulleys or keeping belts from working off into contact with other objects.

Belt Shifters and Clutches

Provision. If in a group drive it is necessary to stop the individual units in the course of normal operation or for frequent adjustment, sufficient tight and loose pulleys with belt shifters, clutches, etc., to be installed to obviate the necessity of throwing belts on or off moving pulleys by hand.

Arrangement. Control levers, etc., to be accessible from a safe position outside railings or guards and located where not readily exposed to accidental contact. All control levers, etc., to automatically lock themselves in the off position and to be arranged for permanent locking in this position by means of a padlock. See Figs. 12 (3) and 14.

Tight and loose pulleys to be arranged so that the loose pulley is readily accessible for lubrication.

Belt shifters and clutches controlling the transmission of power to rooms other than those in which they are located to have extension controls in such rooms, arranged for padlocking in the off position.

Belt shifters, clutches, etc., controlling the motion of such machines as mixing rolls, calenders, and pickers, which, while in motion, are fed by hand from a point close to the intake, to have control within easy reach of the operator's working position. In the case of group drives, group control to be within easy reach of operators of all machines in the group.

Guarding. Friction clutches, unless of the safety type (no revolving projections whatsoever), to be completely enclosed. See Fig. 13 (6).

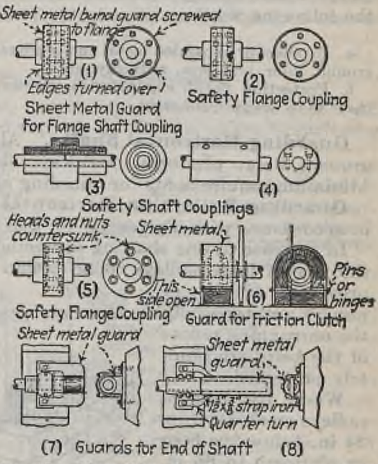


FIG. 13.—Guarding clutches, couplings, and shaft ends.

Gear and Friction Drives. Guarding. All power-driven or hand-operated gears or friction drives up to 6 ft. 0 in. in diameter, wherever located, to be completely enclosed as specified under "Use and Design of Closed Guards." Provisions for oiling and greasing to be made. Gears over 6 ft.-0 in. in diameter constitute special cases which must be considered on their own merits. Where it is impracticable to totally enclose them, pinion and point of mesh should be covered and operators or oilers protected from contact with gear teeth and spokes.

Where frequent adjustments are necessary with the machinery in motion the installation of a railing or general guard, protecting other moving parts as well as the gears, does not obviate the necessity of separately enclosing the gears.

Chain and Sprocket Drives. Guarding. All hand-operated or power-driven chain and sprocket drives to be protected as specified for belts and pulleys.

Sprocket chains located over 7 ft. above routes and stations, which in breaking would endanger those beneath, to be provided with troughs or netting.

Provisions for oiling and greasing to be made.

Rope and Cable Drives. Guarding. To be protected as specified for belts and pulleys.

Shafting

Revolving Projections. No projecting setscrews, bolts, keys, etc., on shafting, pulleys, gears, etc., unless protected by closed guards (not railings). See Fig. 13 (1) and (5). **Safety couplings** (flanged) and **safety collars** without guards are permissible if bolts and setscrews do not project beyond flanges.

Ends of shafts and arbors which project beyond bearings, etc., to be cut off flush or completely enclosed. See Fig. 13 (7 and 8). Absolutely smooth shafts over 2 in. in diameter without keyways are allowed an unguarded projection beyond bearing of not over 1 in.

Guarding Vertical and Inclined Shafts. All vertical or inclined shafts (regardless of speed) which are exposed to contact to be protected in either of the following ways:

a. By sheet or expanded metal enclosure covering all exposed portions up to 6 ft. above ground, floor, platform, walkway, etc.

b. Protection on all exposed sides by standard 42-in. double railing located not less than 36 in. away from shaft.

Guarding Horizontal Shafts. All horizontal shafts within 7 ft. 0 in. of ground, floor, platform, walkway, etc., to be protected against contact. Minimum requirements for guarding are given in Table 17 and Fig. 28.

Guarding Pulleys on Horizontal Shafting. Protection to be applied in accordance with Tables 14, 15, and 16 combined with Table 17.

In the case of line shafting protected by open railings, it is permissible for the face of large pulleys to be within 6 in. (horizontal distance) of railing, provided sides of pulleys and especially the on-running points of the belts are properly protected.

Where the sides of pulleys on shafts from 24 in. below the floor or platform to 66 in. above are closer than 12 in. (horizontal distance) to the bearing, side shields must be installed. Where the sides of pulleys on shafts more than 66 in. above the floor or platform are closer than 18 in. to the bearing, side shields must be installed. See Fig. 12 (4).

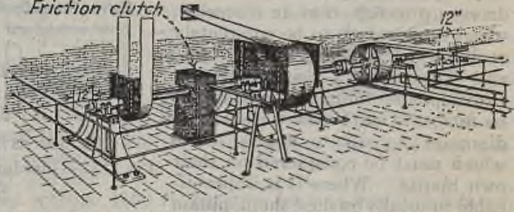


FIG. 14.—Guards for line shafting and pulleys.

Side shields must cover at least one-half (180°) the side of pulleys up to 48 in. diam. and should extend out at least 2 in. beyond the face of the pulley and 4 in. beyond the bearing. On pulleys oiled from above the shield is to cover the upper half on the horizontal diameter. On pulleys oiled from the side or from below, the shield is to cover the nearer half on the vertical diameter except that in case of exposure on both sides of shaft, entire side is to be covered. In relation to the edge of the pulley, shields should be either located with an intervening space exceeding the width of the belt by one-fifth, or located within 2 in. or less of the pulley.

For arrangement of side shields and for guarding of typical line shafting, see Fig. 14.

Protection against Falling and Slipping

Elevated Platforms, Walkways, Etc.

Railings. Standard railings to be installed on all open sides and ends from which there is a vertical drop of 4 ft. 0 in. or more. Railings may be desirable at even lesser heights under special conditions.

Toeboards. To be installed

a. On railings set at the edge of platforms, walkways, floor openings, etc., from which there is a vertical drop of 6 ft. 0 in. or more.

b. When immediately above moving machinery, open tanks of hot or corrosive liquids.

Fillers. Fillers between platform and mid-rail or platform and top rail are desirable under special conditions—for instance, on fire escapes and as protection against moving machinery.

Floors, Platforms, Paths, and Roadways

Slipping Hazards. Where grease, caustic, graphite, sawdust (on wood floors), paraffin, etc., contribute toward slippery conditions, choice of surfacing to receive careful consideration and non-slip material, such as feralun, lead, rubber matting, and expanded metal, to be inserted where necessary and practicable, especially at all dangerous hand-fed machines (calenders, circular saws, jointers, etc.). See Fig. 15 (1).

Floor Plates. Floor plates, flush covers, trapdoors, etc., to be set flush with the floor with no projecting fittings. Smooth metal not to be used for this purpose.

Gratings. As far as possible to be set with parallel members across the direction of travel, especially at emergency exits. Ample bearing surface to be provided, and gratings to be prevented from shifting.

Changes in Level. Abrupt minor changes in floor level to be avoided. Where existing or unavoidable to be treated as follows:

a. By railing to prevent passage across the obstruction.

b. By installing inclines at an angle of not over 20° with the horizontal.

c. By steps. If difference in level is between 6 and 10 in., which may be considered as one step, a $1\frac{1}{4}$-in. projection or "nosing" should be provided along its edge.

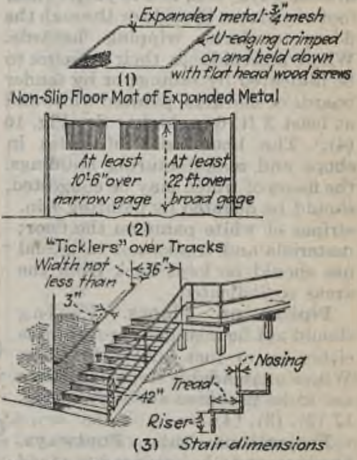


FIG. 15.—Protection of floors, roadways, and platforms.

Trenches and Excavations. Floor openings, trenches, excavations, etc., of a temporary nature to be clearly indicated, if necessary by railing or roping off during the day with the addition of temporary lights after dark if area is not thoroughly illuminated. All permanent openings of this character inside buildings or across regular routes outside to be covered or fenced off.

Traps and Hatches. To be provided with flush doors or covers, having swinging guards with stops to hold them open in a position somewhat beyond the vertical [see Fig. 16 (1)] or to be protected on at least three sides by railings with toeboards, etc. See Fig. 16 (2). Traps and hatches set flush in platforms, floors or other footways to be protected by fixed or hinged grids with strong bars not over 6 in. apart, in addition to the trapdoors or covers, unless standard railing protection is provided. Where materials are discharged through such grids, bars may be placed farther apart, but should be as near 6 in. as practicable.

Foot Clearance for Footways. Piping, valves, steamtraps, diagonal braces, guys, etc., not to project into footways from the side or through the floor to form tripping hazards. Where unavoidable, their presence to be indicated by railings or by fender boards or markers, painted white and at least 3 ft. 6 in. high. See Fig. 16 (4). The boundaries of aisles in shops and manufacturing buildings, the floors of which may be congested, should be denoted by painting 4-in. stripes of white paint on the floor; materials and trucks not in actual use should be kept back from the areas so indicated.

Piping on Floors. Piping should not be installed across routes either on or just above the floor. Where unavoidable steps or inclines are to be provided as shown in Fig. 17 (2), (3), (4).

Projections into Footways. Equipment to be arranged to avoid the projection into footways of such obstacles as clutch and belt-shifter levers, post indicator and hydrant wrenches, valve stems, etc. Where unavoidable, they are to be guarded by railings or indicated by fender boards, etc. Valve wheels to be used instead of inverted L- or T-extension valve handles.

Lighting. Ample natural or artificial lighting to be provided

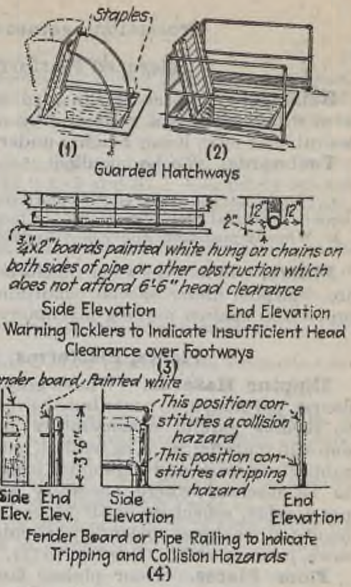


FIG. 16.—Hatchway protection and guards for tripping hazards.

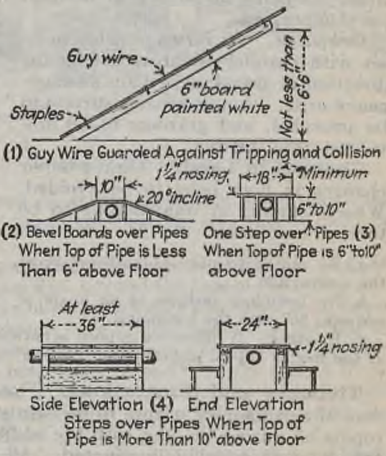


FIG. 17.—Protection for some hazards of tripping.

and maintained for routes and footways, stairs, exits, and where any of the foregoing obstacles to safe travel are present. Such lights to be so arranged that deep shadows and glare are minimized.

Stairs

Angle. Angle with the horizontal should not exceed 45°; best angle, 35°.

Risers and Treads. Height of riser [see Fig. 15 (3)] should not exceed 9 in. and be preferably 8 in. **All risers must be equal,** including first step above and below floors. Width of tread should be not less than 9 in. and preferably 10 in. Nosing to be not less than ¾ in. or more than 1¼ in. **All treads and nosings must be equal.** Best practice gives sum of riser and tread (exclusive of nosing) as 17½ in.

Table 12. Angles for Various Risers and Treads

Riser, in.	Tread, in.	Angle of stairs with horizontal
6½	9	35° 50′
	10	33° 2′
	11	30° 38′
	12	28° 28′
7	9	37° 53′
	10	35° 0′
	11	32° 30′
	12	30° 15′
7½	9	39° 42′
	10	36° 53′
	11	34° 18′
	12	32° 0′
8	9	41° 36′
	10	38° 40′
	11	36° 1′
	12	33° 42′
8½	9	43° 21′
	10	40° 22′
	11	37° 42′
	12	35° 18′
9	9	45° 0′
	10	41° 59′
	11	39° 17′
	12	36° 52′

Width of Stairs. Width of stairs consisting of over three steps should be not less than 36 in. Special stairways for infrequent use (oiling walkways, etc.) may be narrower but should not be less than 30 in.

Landings. Length and width to be not less than width of stairs. Spiral stairs and winders (steps set at an angle to the main flight) not permissible on new construction. Landing platform or clear space at least 36 in. deep to be provided at the foot of all stairs.

Doors. No doorways with doors hung to swing toward stairs except where stair platform at least 36 in. wide is provided. Door sills should be on same level as platform. Doors to be arranged to swing back so as not to block platform or stairs extending to upper landings.

Material and Construction. *a. Wood.* Treads not less than 1-in. board, clear sound stock, securely nailed. Treads to be supported directly by side members and not by cleats nailed to same or to wall. Front edges of treads to be slightly rounded. No cleats or other projections on treads are permissible.

b. Metal. Treads of smooth metal or parallel rods not permissible. Metal stairs with narrow treads (less than 9 in.) are classed as ladders and should not be used except at angles less than 65° with the horizontal. Except in special cases (step ladders, engine-room ladders, etc.) they are objectionable and round-rung ladders are preferred.

c. Concrete. Treads should be roughened, preferably by mixing in abrasive materials, unless (as is desirable on much used stairways) patented non-slip treads are installed.

Railings. Handrails with mid-rails to be provided as specified for platforms and walkways. **Standard height of railings: 42 in. above center of tread.** Lower posts should be set at extreme bottom of stairs with hand- and mid-rail terminated at this point, thus avoiding dangerous projections. Stairways built against the wall to have outside railing and if 4 ft. or more in width to have inside handrail with 3 in. clearance from wall. Completely enclosed stairways to have at least one handrail and if over 4 ft. in width, handrails on both sides. If a space is left between side of stairs and wall, complete double railing to be provided. See Fig. 15 (3).

If materials are piled around stair wells, railings should be entirely filled or stairwell enclosed.

Ladders

General Use. For continuous operation or frequent attendance, stairs should be installed in preference to ladders wherever practicable. Ladders should not be used for emergency exit.

Specifications for Iron or Steel Ladders (Fixed Ladders). *a. Stringers.* Not less than 2 by ½ in. or equivalent. Iron pipe should not be used for stringers. Width between stringers not less than 14 in. Sharp edges, burrs, etc., to be removed.

b. Rungs. Not less than ¾-in. diam. round bar to be used. Rungs to be spaced uniformly not less than 12 in. or more than 15 in. center-to-center. To be smooth and secured against turning.

c. Splice Plates. Cross section equivalent to stringers. To be double riveted or bolted to stringers.

d. Brackets. Cross section of supporting brackets to be at least equivalent to cross section of stringers. To be riveted or bolted to stringers and structure. Brackets to be spaced not more than 10 ft. center-to-center, except in the case of ladders 14 ft. or less in length.

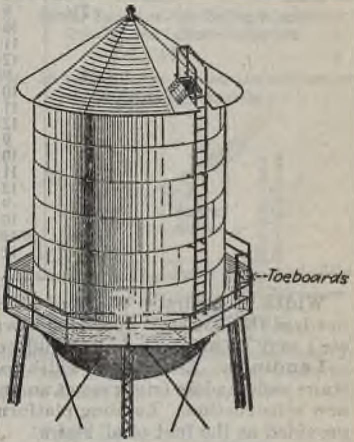


FIG. 18.—Arrangement of ladders on elevated tanks.

e. Stack and Elevated Tank Ladders. U-bars without stringers are permissible for stack ladders but are not recommended for use elsewhere. Diagonal bracing (lattice) on structural steel of elevated tanks, etc., not to be accepted as equivalent to a ladder. Elevated tank ladders not to be carried up outside of balcony but through a hatchway provided for the purpose. Fig. 18.

f. Ladders with Flat Tread. Not to be used except on special work such as engine rooms. Vertical distance between treads should not exceed 10 in., and width of treads should not be less than 4½ in. Angle of ladder should not exceed 65°. Handrails to be provided.

Placing of Permanently Fixed Ladders. *a. Angle.* Ladders should not be set at less than 70° with the horizontal.

b. Length. Where practicable, ladders over 30 ft. in length which are used frequently should be broken by installing one or more platforms. It is further recommended that such ladders, if vertical, be equipped with cages of vertical slats surrounding them. See Fig. 19 (2).

c. Clearance. Minimum permissible clearance for stationary ladders:

Table 13

Clearance, back of rungs (to structure)	5 in. (7–8 in. preferable)
Clearance, front of rungs	30 in. (36 in. preferable)
Clearance, side of stringers	3 in. (6 in. preferable)
Clearance from any part of ladder to shafting, pulleys, etc.	36 in. unless same are enclosed

d. Cleating and Bracing. Ladders to be securely fastened at top and at every 10 ft. of length.

e. Platforms and Hatchways. As far as practicable ladders to be so placed that a rung is directly opposite the platform level. See Figs. 19 and 20.

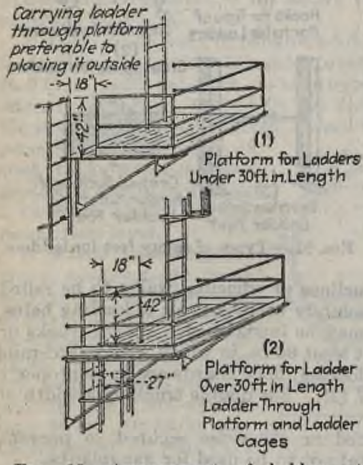


Fig. 19.—Arrangement of ladders at platforms.

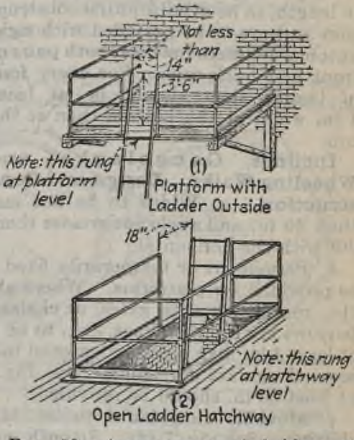


Fig. 20.—Arrangement of ladders at hatchways and platforms.

Stringers to extend 42 in. above the platform at which they terminate. If in special cases this is impracticable, suitable hand grips to be provided.

If both stringers rest directly against the outside edge of the platform at which they terminate, the rungs are to be omitted above the platform and the railings are to terminate close to the side of each stringer. In this case the inside width of the ladder at the top must be at least 14 in. If ladder is placed at right angles to railing, a 24-in. opening in the railing should be left at one side for access. See Fig. 19 (1) and Fig. 20 (1).

Ladders carried continuously up through intermediate platforms should, wherever practicable, pass through hatchways rather than outside the balcony railing, unless ladder is provided with cage.

All open-ladder hatchways to be protected on three sides with standard railing and toeboard. The opening in the railing is to be on the less exposed of the two sides next the ladder. See Fig. 20 (2).

Portable Ladders (Straight Side, Round-rung Wooden Ladders). *a. Length.* Maximum length 30 ft.

b. Non-slip Feet. All portable ladders to be shod with spikes, sharpened points, or safety feet securely fastened. Where ladders are used exclusively on shafting at a uniform height, metal shafting hooks may be substituted. See Fig. 21 (1), (2), (3), and (4).

NOTE. **Portable ladders are unsafe when resting on metal flooring.** They should be secured or held.

c. Portable ladders for permanent use in certain buildings should be clearly marked with the name of the building.

Step-ladders. Not to exceed 15 ft. in length, to be of substantial construction and are to be provided with rigid automatic braces between both pairs of front and back rails. For every foot of length they should be at least 1 in. wider at the bottom than at the top.

Inclines, Gangplanks, and Wheeling Walks. Design and Construction. *a.* Width to be not less than 30 in. and angle not greater than 20° with the horizontal.

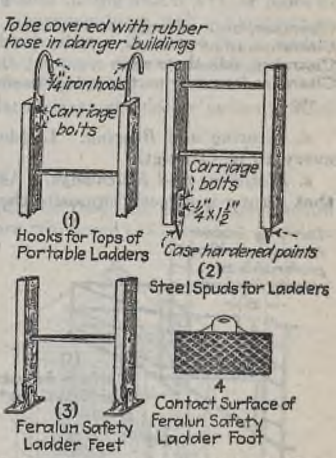


FIG. 21.—Types of safety feet for ladders.

b. Permanent or temporarily fixed inclines or wheeling walks to be railed as provided for platforms. Where absolutely necessary for unloading bales, etc., removable rails, gates, or chains may be introduced. Where trucks or barrows are used, inclines, etc., to be at least 36 in. in width with guard-rails at least 4 in. high securely fastened to flooring; floor boards are to be properly secured against spreading. See Fig. 7 (3). For double trucking a width of at least 60 in. should be provided.

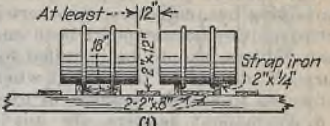
c. Movable gangplanks to be cleated or otherwise secured to prevent shifting. See Fig. 7 (2). Smooth metal not to be used for gangplanks.

Skids. Permanent skids used for receiving or storage of drums or barrels, which are elevated above the ground or floor, are to be planked solid between each pair of rails. Between each pair of skids a 12-in. running board should be provided and the clearance between the ends of drums at this point should not be less than 12 in. See Fig. 22 (1).

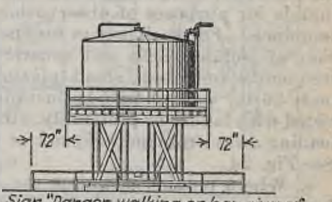
Tanks, Vats, and Pans. Location. At least 36 in. separation to be provided from open containers of hot, corrosive, or otherwise dangerous liquids to routes passing alongside and 72 in. separation to stations. Passageways between two rows of open containers or between a row of open containers and wall, railing, etc., to be at least 72 in. wide.

Protection of Routes and Stations below Tanks. Routes or stations should not be situated directly below open containers of hot, corrosive, or otherwise dangerous liquids, or where subject to their leakage or overflow. Where this cannot be avoided, they are to be protected by roofs, tight floors, pans, etc., properly drained to discharge to a safe point or by the provision of adequate overflow pipes or gutters, or, if necessary, by both.

Unused spaces directly below any tanks containing hot, corrosive, or otherwise dangerous liquids to be railed off and standard danger signs posted. Such railings should be located 72 in. out from outermost edges of tanks or platform overhead. See Fig. 22 (2).



(1)
Permanent Skids for Drums



Sign, "Danger, walking or hanging of clothes underneath tanks is prohibited"

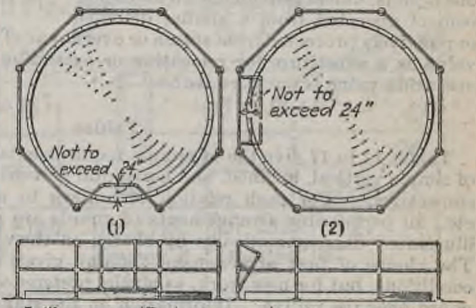
(2) Space below Tanks Containing Hot or Corrosive Liquids Railed off

FIG. 22.—Arrangement of drum skids and railings under tanks.

Railings and Curbs. *a.* No open tanks, vats, or pans to be installed with upper edge less than 6 in. above the adjacent floor level.

b. Open tanks, vats, or pans over 5 ft. 0 in. in diameter (or, if rectangular, over 4 ft. 0 in. in width or length) to have upper edges at least 36 in. above adjacent floor level or to be protected on all accessible sides by standard railings, covers, or hoods. Where operating conditions permit, the upper edges of such tanks not otherwise protected should be at least 3 ft. 6 in. above the adjacent floor level.

c. Where necessary for filling, sampling, etc., such railings may be broken with offsets or inclined sections, provided that distance between top of tank and mid-rail and between mid-rail and top rail is not greater than 24 in. Or tanks may be traversed by walkways with standard railings. Gates or chains may be introduced into railings only where absolutely necessary and then are to be permanently secured so as to be movable but not removable (see Fig. 23).



Railings around Tanks, Arranged for Sampling and Charging

FIG. 23.—Arrangement for railings and curbs around open tanks.

Walkways and Platforms. *a.* To be provided on or above any tanks for reaching bearings, valves, and other appliances, so that it will be unnecessary in the course of normal operation and attendance to step or stand directly on any tank or to place ladders close to open tanks.

b. If directly over any uncovered portion of a tank containing hot, corrosive, or otherwise dangerous liquids, to be provided with standard railings with toeboards, and, if necessary, with splash boards.

c. Walkways over 30 ft. 0 in. in length, passing over or close to tanks containing hot, corrosive, or otherwise dangerous liquids should be provided with facilities for escape at both ends. Wherever practicable, steps, rather than ladders, should be installed for this purpose and are to be located where not subjected to splash, leakage, or overflow.

d. Permanent ladders are not to be located close to the sides of tanks containing hot, corrosive, or otherwise dangerous liquids for purposes of observation or attendance. For such uses or for the operation of isolated acid and caustic valves frequently operated, small platforms at least 36 in. wide should be installed provided with ladder or preferably with stairs leading away from side of tank or valves. See Fig. 24.

e. Where repairs, cleaning, etc., are of frequent occurrence, suitable permanent platforms or walkways to be provided for this purpose.

Location of Controlling Devices. Valves, belt shifters, clutch levers, and other controlling devices to be located at least 36 in. away from edges of tanks containing hot, corrosive, or otherwise dangerous liquids, unless provided with extension control operated from a similar distance or positively protected from splash or overflow. The addition of an emergency valve as a substitute for relocating or extending the control of the regular operating valve is not permissible.

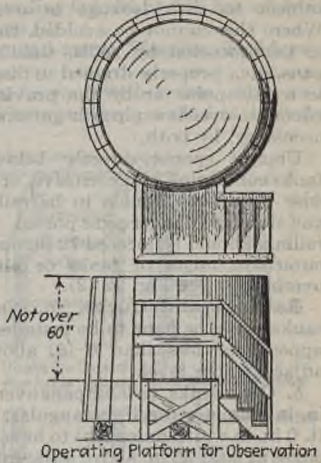


FIG. 24.—Platforms for observing tanks.

Tables

Tables 14 to 17 give the minimum requirements for all *permissible* guarding of simple vertical, inclined, and horizontal belt drives and horizontal shafting, respectively. For each relation of pulleys to each other or to the floor, etc., all permissible arrangements of guards are given in tabular form, each illustrated diagrammatically by sketch on the corresponding Figs. 25 to 28. The choice of best arrangement for any given case must depend on local conditions, but for new work, as a rule, preference should be given the closed guard over the railing. Information on the standard materials and detailed construction of these guards and railings is contained in the foregoing text.

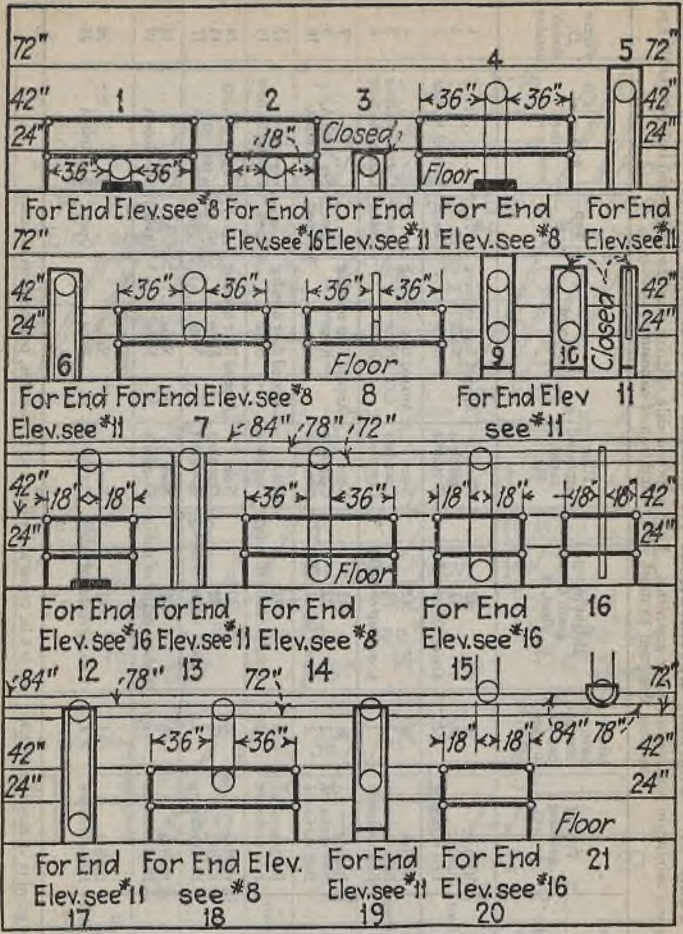


FIG. 25.—Guarding of simple vertical belt drives.

Table 14. Guarding Specifications for Simple Vertical Belt Drives

The following table gives minimum requirements for guarding. Each exposed side of drive must be considered independently and guarding designed to meet worst condition of exposure.

Height of center of shaft above floor		Type of guarding	Minimum horizontal distance from guard to edge of pulley, in.	Height of top of guard above floor, in.	Filler (expanded metal, etc.)	Toeboard	Top of guard	Bottom of guard	Figure 25, sketch number
Upper pulley, in.	Lower pulley, in.								
0–24	0 or below floor	Railing	36	42	None	Around floor opening	.....	....	1
		Railing	18	42	Complete	None	.....	....	2
		Belt enclosure	2	Above top of pulley	Complete	None	Closed	Carried to floor	3
24–72	0 or below floor	Railing	36	42	None	Around floor opening	.....	....	4
		Belt enclosure	2	72	Complete	None	Open	Carried to floor	5
		Belt enclosure	2	Above top of pulley	Complete	None	Closed	Carried to floor	6
24–72	0–72	Railing	36	42	None	Around floor opening	.....	....	7
		Belt enclosure	2	72	Complete	None	Open	See Note A	9
		Belt enclosure	2	Above top of pulley	Complete	None	Closed	See Note A	10
72 or over	0 or below floor	Railing	18	42	None	Around floor opening	.....	....	12
		Belt enclosure	2	6 above center of pulley	Complete	None	Open	Carried to floor	13
72 or over	0–24	Railing	36	42	None	Around floor opening	.....	....	14
		Railing	18	42	Complete	None	.....	....	15
		Belt enclosure	2	6 above center of pulley	Complete	None	Open	See Note A	17
72 or over	24–72	Railing	36	42	None	None	.....	....	18
		Belt enclosure	2	6 above center of pulley	Complete	None	Open	See Note A	19
Over 72	Over 72 but bottom of pulley less than 84 above floor	Railing	18	42	None	None	.....	....	20
		Belt enclosure	2	6 above center of pulley	Complete	None	Open	See Note A	21

NOTE A. Bottom of guard to be closed unless lower edge is 6 in. or less above floor and at least 2 in. below bottom of pulley.

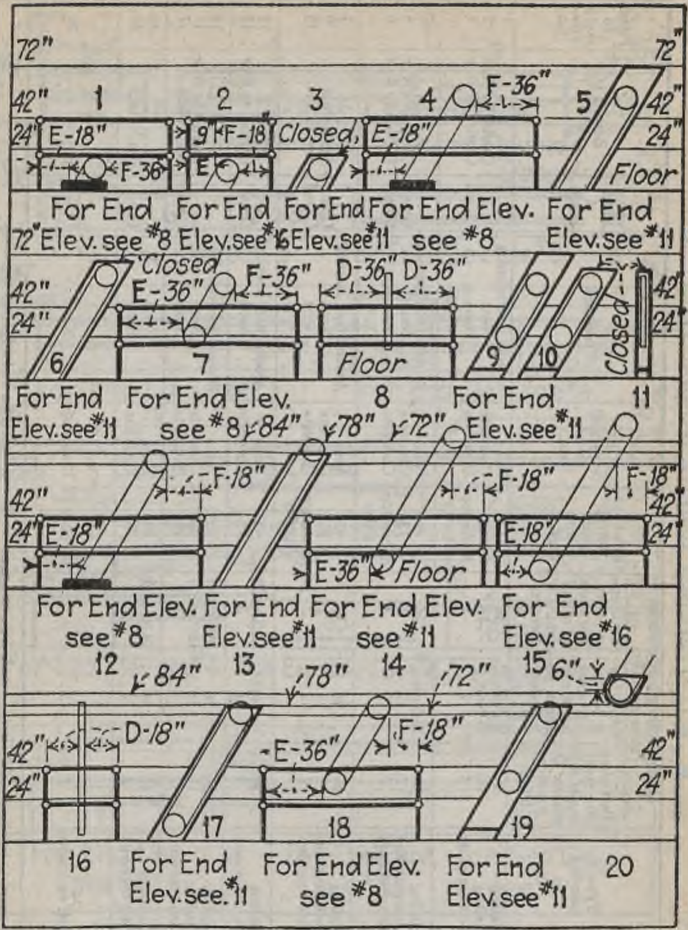


Fig. 26.—Guarding of simple inclined belt drives.

Table 15. Guarding Specifications for Simple Inclined Belt Drives

The following table gives minimum requirements for guarding. Each exposed side of drive must be considered independently, with guarding designed and railings spaced to meet worst conditions.

Height of center of shaft above floor		Type of guarding	Minimum horizontal distance from guard to:			Height of top of guard above floor, in.	Filler (expanded metal, etc.)	Toeboard	Top of guard	Bottom of guard	Figure 26 sketch number
Upper pulley, in.	Lower pulley, in.		Belt edges. (For spacing of side guards. Dimension D), in.	Face of lower pulley. (For spacing of end guards. Dimension E), in.	Face of upper pulley. (For spacing of end guard. Dimension F), in.						
0–24	0 or below floor	Railing	36	36-Note C	36	42	None	Around floor opening	.....	....	1
		Railing	18	18-Note C	18	42	Complete	None	.....	Carried to floor	2
		Belt enclosure	2	2	2	Above top of pulley	Complete	None	Closed	Carried to floor	3
24–72	0 or below floor	Railing	36	36-Note C	36	42	None	Around floor opening	.....	....	4
		Belt enclosure	2	2	2	72	Complete	None	Open	Carried to floor	5
		Belt enclosure	2	2	2	Above top of pulley	Complete	None	Closed	Carried to floor	6
24–72	0–72	Railing	36	36	36	42	None	Around floor opening	.....	....	7
		Belt enclosure	2	2	2	72	Complete	None	Open	See Note A	9
		Belt enclosure	2	2	2	Above top of pulley	Complete	None	Closed	See Note A	10
72 or over	0 or below floor	Railing	36	36-Note C	18-Note B	42	None	Around floor opening	.....	....	12
		Belt enclosure	2	2	2	78	Complete	None	Open	Carried to floor	13
72 or over	0–24	Railing	36	36	18-Note B	42	None	Around floor opening	.....	....	14
		Railing	18	18	18-Note B	42	Complete	None	.....	Carried to floor	15
		Belt enclosure	2	2	2	78	Complete	None	Open	See Note A	17
72 or over	24–72	Railing	36	36	18-Note B	42	None	None	.....	....	18
		Belt enclosure	2	2	2	78	Complete	None	Open	See Note A	19
Over 72	Over 72, but bottom of pulley less than 84 above floor	Belt enclosure	2	2	....	See Note D	Complete	None	Open	Closed	20

NOTE A. Bottom of guard to be closed unless lower edge is 6 in. or less above floor at least 2 in. below bottom of pulley.
NOTE B. If center of shaft of upper pulley is higher than 72 in. above floor this dimension to be taken to nearest part of belt at a height of 72 in.
NOTE C. If lower pulley is below floor, one-half this distance may be used as spacing of end railing from belt at floor level.
NOTE D. Guard to be carried up 6 in. above center of pulley.

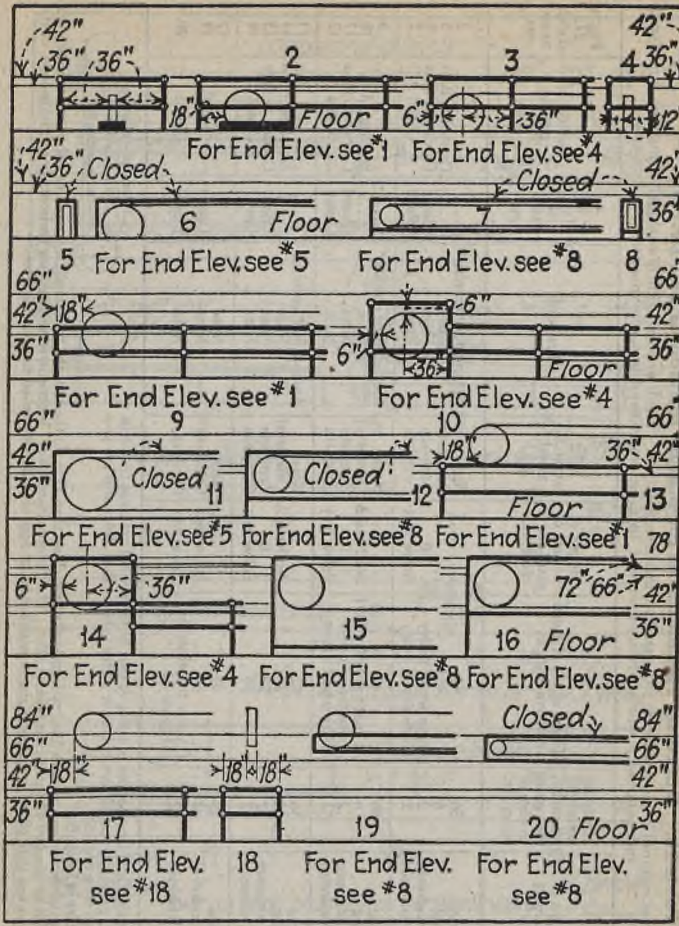


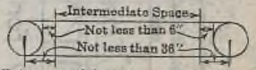
Fig. 27.—Guarding of simple horizontal belt drives.

Table 16. Guarding Specifications for Simple Horizontal Belt Drives

The following table gives minimum requirements for guarding on sides of drives. Each exposed side of each drive must be considered independently and guarding designed to meet worst condition of exposure.

Height of pulley above floor		Type of guarding	Minimum horizontal distance from guard to edge of pulley, in.	Height of top of guard above floor, in.	Height of bottom of guard above floor, in.	Filler (expanded metal, etc.)	Toeboard	Top of guard	Bottom of guard	Figure 27, sketch number
Upper edge, in.	Lower edge, in.									
0–36	....	Railing	36	42	....	None	Around floor opening*	.....	....	2
		Railing	6	42†	....	Complete†	None	.....	Carried to floor	3
		Belt enclosure	2	Above top of pulley	0	Complete	None	Closed	Carried to floor	6
		Belt enclosure	2	Above top of pulley	Below belt	Complete	None	Closed	Closed	7
36–66	....	Railing	36	42	....	None	Around floor opening*	.....	....	9
		Railing	6	At least 6 above pulley†	See Note A	Complete†	None	.....	See Note A	10
		Belt enclosure	2	Above top of pulley	0	Complete	None	Closed	Carried to floor	11
		Belt enclosure	2	Above top of pulley	Below belt	Complete	None	Closed	Closed	12
66 and over	Under 66	Railing	36	42	....	None	Around floor opening*	.....	....	13
		Railing	6	78†	See Note A	Complete†	None	.....	See Note A	14
		Belt enclosure	2	78	0	Complete	None	Open	Carried to floor	15
		Belt enclosure	2	78	Below belt	Complete	None	Open	Closed	16
66 and over	66–84	Railing	18	42	....	None	None	.....	....	17
		Belt enclosure	2	84	Below belt	Complete	None	Open	Closed	19
		Belt enclosure	2	{ Above top of pulley { if below 84	Below belt	Complete	None	Closed	Closed	20

NOTE A. If filler extends 2 in. below bottom of pulley it may be terminated not more than 6 in. above floor, provided that if part of belt or pulley is below floor, toeboards are provided around floor opening.



* If part of belt or pulley is below floor.

† In case of long horizontal drives guarded with filled railings located not closer than 12 in. to the belt, the "intermediate space" (see above sketch) between the pulleys may be protected by *standard 42-in. railings without filler.* The "intermediate space" is considered as extending horizontally to within 36 in. of the center of the shaft of pulleys less than 60 in. in diameter and to within 6 in. of edge of pulleys 60 in. or more in diameter.

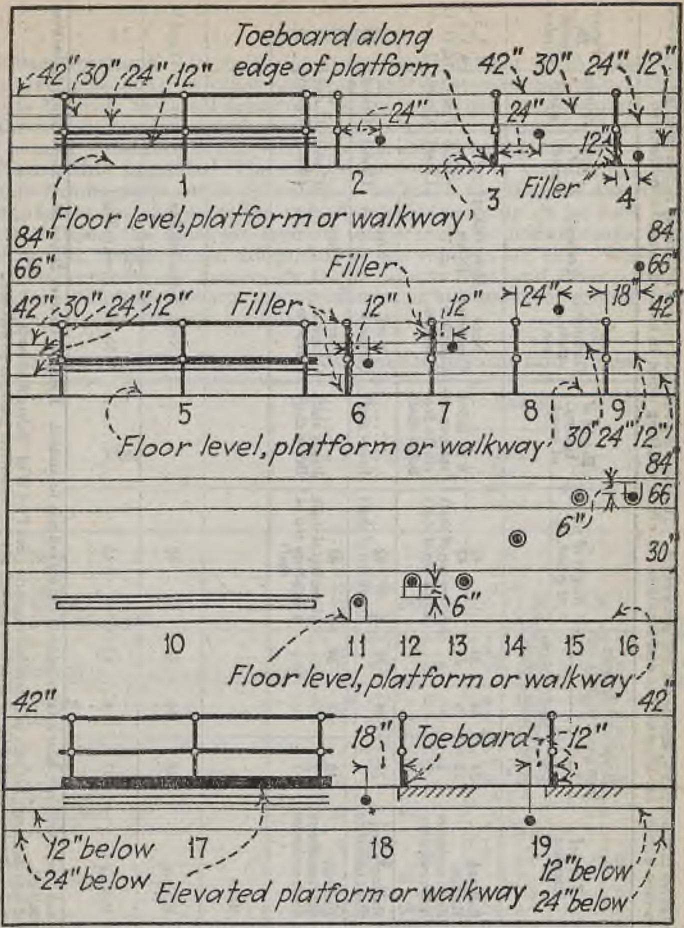


FIG. 28.—Guarding of horizontal shafting.

Table 17. Guarding Specifications for Horizontal Shafting

The following table gives minimum requirements for protection. In designing shafting guards minimum requirements for protection of belts and pulleys (see Tables 14, 15, and 16) and facilities for oiling shafting while in motion must be considered.

Shafting accessible from:	Vertical distance from shaft to floor etc., in.	Type of guarding	Horizontal distance from guard to shaft, in.			Height of top of guard, in.	Height of bottom of guard	Filler (expanded metal, etc.)	Toeboard	Top of guard	Bottom of guard	Figure 28, sketch number
			Best practice	Mini-mum	Maxi-mum							
Floor level platform or walkway	0–30 above	Railing§ Railing Shaft enclosure Shaft enclosure	24 12 2 2	24 6	30 24	42 42 Above top of shaft Above top of shaft	.... 6 in. below shaft or carried to floor Below shaft	None Complete* Complete‡ Complete	Along railing †None	.... Closed Closed	.... Open Closed	1, 2, 3 4, 5, 6, 7 10, 11, 12 13
Floor level platform	30–66 above	Railing§ Shaft enclosure	24 2	24 ..	30 ..	42 Above top of shaft	.... Below shaft	None Complete	†None	..... Closed	..... Closed	8 14
Floor level platform or walkway	66–84 above	Railing§ Shaft enclosure Shaft enclosure	18 2 2	18	24	42 Above top of shaft 6 in. above top of shaft	.... Below shaft Below shaft	None Complete Complete	†None	..... Closed Open	..... Closed Closed	9 15 16
Floor level (by portable ladder)	Over 84 above	None										
Elevated platform or walkway	0–12 below	Railing	18	18	24	42	....	None	Along railing	.....	.....	17 and 18
Elevated platform or walkway	12–24 below	Railing	12	8	12	42	....	None	Along railing	.....	.....	19

* If shaft is between 0 and 12 in. above floor, filler between mid-rail and top rail may be omitted. If shaft is between 24 and 30 in. above floor, filler between floor and mid-rail may be omitted.

† Not required as guard for machinery, but may be otherwise necessary. See Part II of "Safety Standards."

‡ Not necessary to extend enclosure to protect inaccessible side of shaft which is away from platform or walkway.

§ If end of shaft is accessible, railing to be extended around it and located not less than 12 in. (horizontal distance) from end bearing.

PROCESSES

Closed Systems. In the design of process equipment which is to handle corrosive, poisonous, or otherwise dangerous substances, preference should always be given to closed systems. Where this is not possible, particularly in the case of poisonous dust or gas, special ventilating units which remove the harmful substance at the point of origin should be considered.

Flammable Liquids. The storage and handling of volatile, flammable liquids require that certain precautions be taken to minimize the hazard. The inherent fire and explosion hazard depends not only on its flash point, but also upon the apparent ignition temperature, explosive range, vapor density with respect to air, diffusibility of the vapor in air, etc. While data on these properties are incomplete at the present time, and those published by different authorities vary, the following table indicates the relative hazard of certain liquids (see also pp. 2914 to 2916).

Table 18. Fire Hazard of Flammable Liquids and Gases*

Fluid	Flash point, °F.	Apparent ignition, °F.	Explosive range		Vapor density (Air = 1)
			Lower limit, per cent in air	Upper limit, per cent in air	
Acetate, Ethyl	19 to 41	903 -1055	2.3	11.4	3.04
Acetone	−4 to 36	1118 -1235	2.0	9.0-13.0	2.0
Alcohol, Amyl	79 to 107	660 - 856	1.48	....	3.04
Alcohol, Methyl	30 to 90	887	5.5 - 7.05	21 -36.5	1.12
Alcohol, Ethyl (absolute)	51 to 70	737 - 850	2.8 - 4.0	9.5-17.0	1.613
Acetylene	....	762.8- 824	2.5 - 2.6	55.0-82.0	0.898
Benzine	−5 to 10	475	1.4 - 2.5	4.8- 5.9	4.48
Benzol (benzene)	−16 to 50	932 -1079	1.41- 2.7	6.3- 8.0	2.77
Butane	....	805	1.5 - 1.9	6.0- 8.5	2.046
Carbon bisulfide	−22 to −4	212 - 275	1.0 - 1.06	50.0	2.6
Ether, Ethyl	−49 to 29	356 - 398	1.0 - 1.71	6.0-48.0	2.565
Ethane	....	950 -1166	3.0 - 3.3	10.6-15.0	1.03
Ethylene	....	1007.6-1016.6	3.0 - 3.4	34.0-35.0	0.97
Ethylene glycol	232	775 - 975	3.2	....	2.14
Ethyl chloride	−37 to 0	....	3.6 - 4.0	11.2-14.8	2.22
Formaldehyde	90	....	4.0	13.6	1.03
Gasoline (casinghead)	−10 to 20	497	1.3	6.0	
Gasoline (motor)	0.4	513 - 536	1.4 - 2.0	6.0	3.5
Hexane	−7 to 0	479 - 516	1.1 - 1.2	4.2	2.97
Hydrogen	....	1076 -1094	4.1 - 8.0	75.0-80.0	0.07
Hydrogen sulfide	....	482	4.5	45.5	1.17
Methane	....	1202 -1382	5.0 - 5.5	13.0-16.0	0.55
Pentane	−40 to 80	588	1.26- 2.5	4.5- 8.0	2.48
Propane	....	870	2.0 - 2.4	7.0- 9.5	1.52
Propylene	....	....	2.0 - 2.4	9.0- 9.7	1.45
Pyridine	0 to 80	900	1.8 - 1.81	12.4	2.73
Toluol	44.6 to 55	1026 -1027	1.0 - 3.0	6.0- 7.0	3.14

* Based on "International Critical Tables" (McGraw-Hill), and publications of the Interstate Commerce Commission, the Underwriters' Laboratories, Inc., and other sources—by permission.

Any volatile liquid which gives off flammable vapors at or below ordinary room temperature is considered hazardous from the fire prevention standpoint and warrants special consideration. The following definitions may be helpful when analyzing situations which involve the use of such materials:

Combustion,[1] generally speaking, implies some form of chemical change accompanied by the evolution of both heat and light. Although there are examples of true combustion in the entire absence of oxygen, the term usually means oxidation. There are two types of combustion; namely, slow combustion which is not accompanied by light such as the oxidation of iron; and rapid combustion which we know as "fire," due to the presence of flame. The term "fire" includes both combustion and flame, and the results of both.

Apparent ignition temperature of an element or compound, whether solid, liquid, or gaseous, is the temperature required to initiate or cause oxidation sufficiently rapid to be self-sustained when the heating or heated element is removed.

Flash point of a liquid is the temperature at which it gives off vapor sufficient to form an ignitable mixture with the air contained in the vessel used. It does not mean that no evaporation takes place below that temperature, but that vapor does not come off sufficiently freely to exceed flash-point classification requirements. This term applies more especially to flammable liquids, although there are certain solids, such as camphor and naphthalene, that slowly evaporate or volatilize at ordinary room temperature and therefore have flash points.

Flash fire is very rapid combustion; it can take place only (*a*) in highly flammable solids or liquids which contain sufficient oxygen for complete or nearly complete combustion, or (*b*) when the particles of a combustible vapor or dust are suspended in a diffused state in the air, close enough to each other to allow propagation of flame through the vapor or dust cloud and still sufficiently separated to leave room for the necessary amount of oxygen for combustion. In either case, there must be a source of heat sufficiently high in temperature to equal the apparent ignition temperature of the material.

Explosive range refers to the definite limitations of combustibility and rate of burning of flammable vapor or dust mixed with the air. When the particles are so widely separated that those set on fire by the igniting medium will not set fire to others that are nearest, the mixture is called too "lean" for combustion and will not burn. When the particles are so close together that they exclude the oxygen necessary for combustion, the mixture is called too "rich," and it will not burn. The concentration, or percentage by volume, between the leanest and the richest mixtures that will burn is called the "explosive range." Between the minimum and maximum limits of the explosive range will be found various phases of slow and rapid combustion.

Vapor or dust explosion results when the products of combustion from a flash fire are sufficiently confined to generate pressure.

Explosion or detonation presupposes a sudden violent change of pressure, characteristically involving the liberation and expansion of a large volume of gas due to high temperature. The change taking place is a progressive one proceeding from one part of the exploding material to the next adjoining part. The term "detonation" usually is applied to an explosion of very high order, *i.e.* where high velocities are involved.

Pressure rupture, such as failure of an air receiver, steam boiler, or closed receptacle containing fluids, is a rupture resulting from internal pressure exceeding the ultimate strength of the container. The terms explosion and detonation, particularly explosion, are commonly and errone-

---

[1] MINER, *J. Soc. Automotive Eng.*, **21,** No. 6.

ously used to include pressure ruptures. While the results may be similar, the mechanism of a pressure rupture is entirely different.

A true explosion or detonation involves chemical change, while a pressure rupture is merely the failure of the enclosing receptacle because of too great internal pressure.

Spontaneous combustion (according to C. T. Kingzeth, "Chemical Encyclopedia," Baillière, Tindall & Cox, London) "ensues as a result of the development of heat arising from chemical changes; thus, cotton waste soaked in linseed oil will sometimes fire in consequence of the rapid oxidation of the oil. . . . Similarly phosphorus exposed to air will take fire. . . . "

Decomposition (Chemical Encyclopedia by C. T. Kingzeth, Bailliere, Tindall & Cox, London). "When a chemical compound is broken up into its constituent parts or simpler compounds by chemical or physical means, it is then said to be decomposed, the decomposition being effected by the exercise of a greater force than that of the original binding chemical affinity."

Location and Arrangement of Storage Tanks for Flammable Liquids. Flammable liquids are usually divided into classes depending upon their flash point. However, flash point is not a true indication of the hazard of a flammable liquid because other items such as ignition temperature, explosive range, diffusibility of the vapors, and density of the vapors also affect the hazard. The potential hazard involved in the storage and handling of flammable liquids in industrial plants requires that certain items, such as strength, capacity, and location of storage tanks as well as vents, dikes, and quick-closing valves in pipe lines, be given consideration. For information on installation of both underground and inside storage tanks see "Containers for Storing and Handling Flammable Liquids," National Board of Fire Underwriters.

The most common method of storing large quantities of flammable liquids in industrial plants is in above-ground, outside storage tanks. Where such storage tanks are used, the relation between capacity of individual tanks and the permissible distance from other property is shown in the following table quoted from "Containers for Storing and Handling Flammable Liquids," National Board of Fire Underwriters.

Table 19. Capacity and Minimum Distances of Tanks

Capacity of Tank, Gal.	Minimum Distance to Line of Adjoining Property or Nearest Building, Ft.
3,000 or less	20
21,000 or less	25
31,000 or less	30
45,000 or less	40
64,000 or less	50
80,000 or less	60
128,000 or less	75
200,000 or less	85
266,000 or less	100
400,000 or less	150
666,000 or less	250
1,333,000 or less	300
2,666,000 or less	350

For tanks of over 400,000 gal. capacity, a minimum distance of 175 ft. to adjoining property or nearest building may be permitted provided that an approved type of extinguishing system is installed for the tank and covering other parts of the yard or system.

For tanks permitted from 50 ft. and up to 175 ft. of building or property line, the capacity may be increased 33 per cent if the tank is provided with an approved extinguishing system.

The minimum distance from tank to tank should conform to the following table also quoted from these regulations:

Table 20. Capacity and Minimum Distances between Tanks

Capacity of Tank, Gal.	Minimum Distances to Any Other Tank, Ft.
300 or less	3
500 or less	3
1,000 or less	3
8,000 or less	3
12,000 or less	3
18,000 or less	3
24,000 or less	5
30,000 or less	10
48,000 or less	10
75,000 or less	13
100,000 or less	15
100,000 to 2,500,000	One tank diameter

Three methods of handling these liquids are in common use—by pumping, by gravity, and by compressed air or inert gas. Compressed air is the least desirable and should not be used unless it is impossible to use other methods. Pumping is preferable to gravity discharge and should be used wherever possible.

Testing Devices for Flammable Vapors. Flammable-vapor indicators, which read in terms of the lower limit of the explosive range, are now available for determining the approximate amount of flammable vapor in air. Most of the instruments will operate satisfactorily, not only in any given solvent vapor but also in any mixture of vapors even though the composition may be unknown.

Static Electricity. Static electricity may develop from handling materials—solids, liquids, or gases—or from the operation of equipment such as belts. The handling of dry granular substances, such as sulfur in metal chutes, bins, or even through relatively dry air, will generally result in the generation of static electricity. Similarly, static electricity may be generated by the flow of certain liquids through pipe lines.

The discharge of static electricity in the presence of flammable dusts or vapors is considered hazardous, and, where this may occur, a proper system of grounding, humidification of the air in the room, or the use of certain static eliminators which are available today is usually warranted. For additional information on the hazards of static electricity, consult *Safe Practices Pamphlet* 52, National Safety Council.

Compressed Gases in Cylinders. The Interstate Commerce Commission has established definite regulations that must be complied with in **filling and transporting** cylinders of compressed gases. These rules also include specifications for the construction of the containers. In **using** these cylinders, a few of the more important precautions to be taken are listed below; additional safeguards are outlined in the following regulations, published by the National Board of Fire Underwriters: Acetylene Equipment for Lighting, Heating and Cooking, 1929; Gas Systems for Welding and

Cutting, 1931; Compressed Gas Systems Other than Acetylene for Lighting and Heating, 1927.

1. Extreme care should be exercised in handling so that cylinders are neither dropped nor permitted to strike against each other.

2. Cylinders should be protected against mechanical injury when in use.

3. Precautions should be taken that the safety devices with which the cylinders are equipped are not tampered with.

4. When cylinders are not in use, outlet valves should be kept tightly closed even though cylinders may be considered empty.

5. When the cylinders have been exhausted, the discharge valves should be closed and the protecting caps screwed securely into position.

6. Before attaching the required pressure regulator to the cylinder the valve should be slightly opened and then closed to be sure it is in proper working condition.

7. Only the special wrenches and tools provided by the manufacturer should be used on valves.

8. Make sure that the thread connections on regulators correspond to those on the cylinder outlet as the threads may vary somewhat for different gases.

9. Regulators and pressure gages provided for use for a particular gas must not be used on cylinders containing other gases.

10. All cylinders should be protected against excessive rise in temperature and against extremes of weather. They should not be exposed to continued dampness.

11. In order to avoid confusion, full cylinders should be stored apart from the empty ones.

12. Cylinders of compressed gas should not be stored near highly flammable materials.

13. In welding or cutting, no sparks should be allowed to strike the cylinders.

14. Acetylene is shipped in cylinders completely filled with a porous material which is charged with a suitable solvent (usually acetone).

15. Every possible precaution should be taken to prevent oxygen from coming in contact with oil or grease.

16. The cylinders received by the plant should bear a conspicuous standard label indicating the kind of gas. The color of the label shows whether the gas is flammable, corrosive, or inert.

17. Oxygen should never be used as a substitute for compressed air.

Dust Explosions (see definition under Flammable Liquids). Dust explosions can occur in any combustible dust or any material that will burn or oxidize, which includes some metallic dusts.

When there is a proper mixture for rapid combustion, it is said to be within the explosive range, which includes all percentages of dust and air from the lower to the upper limit. Below the lower limit, flame propagation will not take place and, above the upper limit, there may be burning, but an explosion or flash fire is not likely to occur in apparatus or enclosed spaces unless air is admitted. The lower and the upper limit may be widely separated depending upon the character of the dust.

The degree of flammability of a dust depends upon its moisture content and upon its fineness. The drier and the finer a dust is, the greater its flammability.

While the intensity of the explosion may vary, according to the percentage of dust and air, any dust combination within or above the explosive range is dangerous and should not be permitted. The safeguarding of this hazard, as with flammable gases, involves the dilution below the lower limit of the explosive range and immediate removal while in the diluted state.

In most disastrous dust explosions there are usually two explosions—a primary and a secondary. The primary involves a relative small quantity of dust and is local in character, but of sufficient intensity to dislodge dust which has collected in buildings—on ledges, side walls, floors, or which may be released due to rupture of equipment or otherwise thrown into the air. If ignited, a more general explosion, the secondary, may occur which may result in major structural damage. Precautions against dust explosions consist of keeping the premises scrupulously clean and eliminating sources of ignition.

SPECIAL SAFETY PROTECTION EQUIPMENT

The use of protective equipment is a secondary measure against injury and health hazards which cannot be eliminated by the design and arrangement of equipment, and by the installation of adequate ventilation. Since the human factor enters into the wearing of such equipment, its dependability is variable and at times uncertain.

For the protection of the head and eyes, such equipment as goggles, shields, helmets, hoods, masks, and hats should be worn. The breathing of poisonous gases and fumes, or of an atmosphere in which there is a deficiency of oxygen, must be guarded against by the use of respirators, canister masks, hose masks, or oxygen helmets. The hands, arms, legs, and feet may be protected by the use of rubber or canvas gloves, rubber boots and aprons, leggings, shoes, and other clothing.

Protective equipment should not be transferred from one employee to another unless it is sterilized. A 3 per cent solution of Lysol, or some other suitable disinfectant, may be used for this purpose.

Table 21. Recommended Head and Eye Protection

Conditions	Protection
Chipping, riveting, calking, etc.	Types 2 and 6
Scaling, grinding, etc.	Type 1, 2, or 6; preferably Type 2
Exposure to dust and wind	Type 1, 2, 5, or 7; depending on conditions
Babbiting	Babbiting mask
Handling corrosive chemicals	Types 4 and 5, or face masks, or hoods; depending on conditions
Sand blasting	Sand blast cabinets or helmets
Sunlight, snow, and similar sources of glare	Type 1, 2, 3, or 5 goggles; the shade of lenses to be determined by conditions
Oxyacetylene welding and furnace work	Helmets, shields, or type 3 goggles; shade 6 lenses
Electric welding	Helmets or shields with shade 10 to 12 windows
Nitrometer work	Special nitrometer shield

Type 1. Non-shatterable lenses in strong frames with no side shields.
Type 2. Non-shatterable lenses in strong frames with side shields.
Type 3. Oxyacetylene welding.
Type 4. Close-fitting rubber, light lenses, little or no ventilation.
Type 5. Cup type, close fitting, wide vision, light lenses, little or no ventilation.
Type 6. Cup type, close fitting, wide vision, with non-shatterable lenses, ventilated.
Type 7. Cup type, tight-fitting, light lenses, cloth sides.

Goggles and Helmets. Goggles may be divided into approximately seven distinct types. Table 21 will serve as a guide in selecting the general type for the work to be done.

Respiratory Protection. Devices for respiratory protection may be divided into the following general classes:

1. Respirator (mechanical filter).
2. Respirator (canister).
3. Respirator (air line).
4. Gas mask (canister for certain gases or combinations).
5. Gas mask (canister for all gases—"All Service").
6. Gas mask (hose or air line).
7. Oxygen-breathing apparatus (self-contained).

The following may be used as a guide in determining the device required.

1. Respiratory protection for non-toxic dusts and vapors. Filter renewable.

2. Respiratory protection for brief period against low concentrations (not over 2 per cent), of the toxic or harmful dusts or vapors for which the canister is designed. Canister renewable. Should not be used where there is a deficiency of oxygen (less than 16 per cent).

3. Respiratory protection in any dust or vapor and in the case of oxygen deficiency. Fresh air is conveyed by hose from an uncontaminated source, the limit of travel depending on length of hose through which wearer may breathe.

4. Respiratory and face protection for limited period in low concentrations as in case 2. Canister renewable.

5. Respiratory and face protection for limited period in low concentration of any one or any combination of dusts or vapors. Canister renewable. Should not be used where there is a deficiency of oxygen.

6. Respiratory and face protection in any dust or vapor or in the case of oxygen deficiency. Fresh air from an uncontaminated source is drawn by wearer or forced by blower through hose, the limit of travel depending on length of hose wearer may be able to handle.

7. Respiratory protection for limited period in any dust or vapor or in the case of oxygen deficiency. Oxygen supplied by apparatus; cumbersome.

The colors approved for gas-mask canisters by the American Standards Association are as follows:

Table 22. Standard Colors for Gas-mask Canisters

Gases, Vapors, Smoke, Etc.	Color
a. Acid	White
b. Organic vapor	Black
c. Ammonia	Green
d. Carbon monoxide	Blue
e. Acid and organic vapors	Yellow
f. Acid and organic vapors and ammonia	Brown
g. Dusts, smokes, mists in combination with any of the above gases	½-in. contrasting black or white stripe
h. All of the above gases	Red. Filters are included in this canister, but stripes to indicate them are unnecessary
i. Other special gases	Color to be assigned

SPECIAL FIRE PROTECTION EQUIPMENT

Special fire protection equipment is receiving an increasing amount of attention. New or unusual processes should be carefully analyzed, the hazards determined, and the best methods of fire extinguishing provided. In some cases more protection is required than that afforded by the standard system of automatic sprinklers or by first-aid extinguishing appliances. The following are some of the more common systems in use.

Foam (Two-solution System). Foam is generated by mixing a solution of aluminum sulfate, which is known as A solution, with a solution of bicarbonate of soda and a foam stabilizer, which is known as the B solution. The solutions are stored in separate tanks until required for use when they are pumped to a special mixing device, which discharges foam. Foam is particularly useful in class B fires, which are those in oils, greases, flammable liquids, etc., where the blanketing or smothering effect of the extinguishing agent is of greatest importance.

Foam (Dry-powder Generators). Foam is generated in a special device by mixing water with dry chemicals. The discharge is continuous as long as the water is supplied and the chemicals fed into the device. The foam is piped through permanent piping or hose to the location where needed. The smaller sizes of generators are readily portable.

Carbon Dioxide. Carbon dioxide may be stored in the shipping containers under pressure and piped to the apparatus or location where required. At this point it may be discharged either automatically or manually through suitable nozzles, thereby smothering the fire. There is a slight cooling effect, although the characteristic of carbon dioxide extinguishing is the smothering effect. The automatic discharge is secured through fixed temperature or through temperature rate-of-rise devices. The former operate when a predetermined temperature has been reached, the latter when the increase in temperature exceeds a predetermined rate-of-rise.

Steam is sometimes used for special locations or in certain equipment. It is not generally recommended.

Auxiliary Sprinkler Systems. Automatic sprinklers, or in some cases sprinklers with the fusible links removed, may be supplied with water controlled by a quick opening valve or by a deluge valve which in turn may be actuated thermostatically or manually by remote pull handles. This arrangement is used where it is desirable to discharge water over extra hazardous equipment or areas.

Other inert gas, such as flue gas or nitrogen, is sometimes piped to the interior of grinding, pulverizing, mixing, or conveying equipment, where extra hazardous products are handled or where there may be acute hazards due to presence of flammable dust or gases.

ORGANIZATION AND WORK

Safety and Fire Protection Organization. This organization should be essentially the same as the production organization. There should be a central committee composed of the plant manager or the assistant plant manager as permanent chairman and the safety and fire protection supervisor, if there be one, as permanent secretary. The members of this committee should be the remainder of the plant executive staff.

There should also be a departmental committee in each department, or such subdivision, composed of the head of the department as permanent chairman and the foremen as members. In order to carry this organization to the workmen, each foreman should have a committee made up of himself as permanent chairman and his men as members, the latter being rotated if there are too many to serve at one time.

The central safety and fire protection committee will take care of general policies regarding safety and fire protection. The departmental committees will be responsible for safety and fire protection in their departments, and the foremen's committees in their particular areas.

Following is an outline of a typical plant safety and fire protection organization.

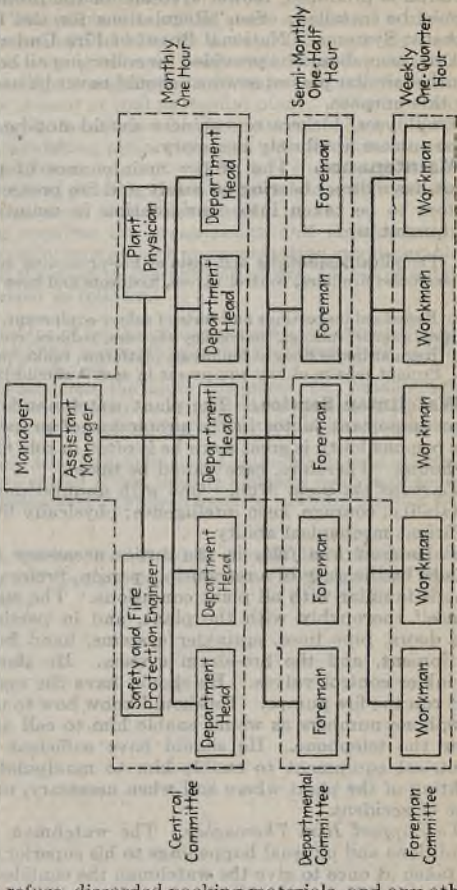


FIG. 29.—Chart of a typical safety and fire protection organization.

Housekeeping. Housekeeping has a direct bearing on the number of accidents and fires which occur, as well as on the plant operating costs. It is essential for the safety and fire protection engineer or for the employee who is charged with this work to pay particular attention to this very important item. The rule, "a place for everything and everything in its place," should be enforced. Keen interest in housekeeping should be evidenced by the plant management, who should require his subordinates, and they in turn the workmen, to be directly responsible for the housekeeping in their immediate vicinity. The maximum result from the minimum effort expended comes from daily or even more frequent sweeping and collection of refuse. Employees should be trained to minimize the cleaning required by depositing in receptacles provided for the purpose all dirty waste and general refuse, discarded packing materials, and any other waste materials as such and those used for maintenance and operation as they are produced.

Suitable metal containers with deep-lipped covers should be provided for oil-soaked waste, cleaning rags, and hazardous flammable materials. The containers should be emptied at designated periods, preferably at the end of the day's work.

Lumber if no longer needed and other waste material, whether combustible or not, should be immediately removed during construction work.

Where quantities of sawdust, shavings, or other similar combustible waste material is produced, blower systems for the proper removal of this material should be installed. See "Regulations for the Installation of Blower and Exhaust Systems," National Board of Fire Underwriters.

Drip pans should be provided for collecting oil below motors, shaft bearings, or other similar places; sawdust should never be used as an absorbing material for this purpose.

Employees' lockers or cabinets should not be located in manufacturing areas, unless absolutely necessary.

Maintenance. The proper maintenance of plant property and equipment has a direct bearing on safety and fire protection. A few of the general factors to be taken into consideration in maintaining plant property and equipment are:

a. Periodical inspections and tests of fire-protection equipment, such as fire pumps, underground fire lines, control valves, hydrants and hose equipment, sprinkler systems, etc.

b. Periodical inspections and tests of safety equipment, such as respirators, gas masks, goggles, guards, railings, emergency showers, ladders, etc.

c. Regular inspections of buildings, platforms, tanks, pressure vessels, machinery, etc.

d. Prompt repairs of any equipment in case it should be needed.

Watchman Service. The plant watchman holds a position of trust and is an important factor in the protection of property against fire and theft. His responsibility is great since he is often in sole charge over half of the daily 24 hours. Therefore, care should be taken.

To Select the Right Man. One with unquestionable character, habits, and reliability; courage, keen intelligence; physically fit and able-bodied and with sufficient mechanical ability.

To instruct him fully in the duties necessary for efficient service. This should be the duty of a responsible person, preferably the plant management who is familiar with all plant conditions. The watchman should familiarize himself thoroughly with the plant, and in particular stairways, elevators, fire doors, pipe lines, sprinkler systems, hand hose, first-aid extinguishing equipment, and the fire-alarm system. He should know the location of sprinkler control valves. He should have the necessary knowledge to start and run the fire pumps. He should know how to use the telephone, and such telephone numbers as would enable him to call assistance should be posted near the telephone. He should have sufficient instruction regarding the electrical equipment to enable him to manipulate switches to control the lighting of the plant where and when necessary, or to shut off the current in case of accident.

To Support Him Thoroughly. The watchman should report undesirable conditions and unusual happenings to his superior, action upon which should be taken at once to give the watchman the confidence to report further undesirable conditions. If decision is unfavorable on his recommendations, the reasons should be fully explained to him.

In order to insure that the watchman visits all of the important points in the plant and as evidence that he has properly performed his duty, a watch-clock system should be provided with stations well located throughout the property. The portable-clock system is a very reliable and usually an inexpensive system to install. The records made on the clocks should be carefully checked each morning by a thoroughly competent person, and any irregularities in punching or in the order in which the stations are punched should require an explanation. In most cases closer supervision can be maintained by installing a log-book in which the watchman can make entries of undesirable conditions and unusual happenings.

Plant Fire Brigade. The efficient handling of fire apparatus is absolutely essential if the maximum protection is to be derived from any given equipment. It is, therefore, customary on most properties, even if small, to select a few reliable, strong, intelligent employees and train them in the operation of all extinguishing agents present at that particular plant. Through frequent drills they become thoroughly familiar with the location of the buildings and the nearest available extinguishing apparatus, and also the interior arrangement of each building and its occupancy. It is also customary to educate these employees as to the need of using due caution when entering buildings housing processes more hazardous than the average.

Successful fire fighting requires the organization and development of a plant fire department, for only through such an organization can the employees be instructed, drilled, and directed in the fighting of fire. This phase of the subject might be summarized as follows:

a. For the prompt extinguishing of fire the employees must know the location and use of the first-aid equipment, and the minor extinguishing apparatus, such as chemical extinguishers, pails, small hand hose, etc.

b. To obtain maximum benefit from the major equipment, *i.e.*, outside hydrants, large fire hose, fire pumps, chemical engines, etc., certain employees must know the location and be trained in the proper method of handling and directing this apparatus, as well as knowing its capacity.

c. To prevent confusion and delay, a definite line of authority must be established.

d. To check successfully the spread of fire, those in authority must have knowledge as to building construction, arrangement, nature of contents, and inter-exposure between buildings.

For additional information on fire brigades, consult the "Suggestions for the Organization, Drilling and Equipment of Private Fire Brigades," National Board of Fire Underwriters.

MECHANICAL TABLES AND DATA FOR THE SAFETY AND FIRE PROTECTION ENGINEER

Table 23. Average Ultimate Strength of Common Metals

Pounds per square inch

Material	Tension	Compression	Shear	Modulus of elasticity
Aluminum	15,000	12,000	12,000	11,000,000
Brass, cast	24,000	30,000	36,000	9,000,000
Bronze, gunmetal	32,000	20,000	......	10,000,000
Bronze, manganese	60,000	120,000		
Bronze, phosphor	50,000	....	......	14,000,000
Copper, cast	24,000	40,000	30,000	10,000,000
Copper wire, annealed	36,000	....	......	15,000,000
Copper wire, unannealed	60,000	....	......	18,000,000
Iron, cast	15,000	80,000	18,000	12,000,000
Iron wire, annealed	60,000	....	......	15,000,000
Iron wire, unannealed	80,000	....	......	25,000,000
Iron, wrought	48,000	46,000	40,000	27,000,000
Lead, cast	2,000	....	......	1,000,000
Steel castings	70,000	70,000	60,000	30,000,000
Steel, plow	270,000			
Steel, structural	60,000	60,000	50,000	29,000,000
Steel wire, annealed	80,000	....	......	29,000,000
Steel wire, unannealed	120,000	....	......	30,000,000
Steel wire, crucible	180,000	....	......	30,000,000
Steel wire, susp. bridge	200,000	....	......	30,000,000
Steel wire, piano	300,000			
Tin, cast	3,500	6,000	......	4,000,000
Zinc, cast	5,000	20,000	......	13,000,000

"Machinery's Handbook."

Table 24. Tensile and Compressive Strength of Different Kinds of Steel

The ultimate strength of steel in tension and compression is practically the same and may, for different kinds of steel, be assumed as follows:

Kind of steel	Ultimate strength lb. per sq. in.	Kind of steel	Ultimate strength lb. per sq. in.
Structural steel for rivets	55,000	Machine steel	75,000
Structural steel for beams	60,000	Gun steel	90,000
Boiler steel for rivets	50,000	Axle steel	100,000
Boiler steel for plates	60,000	Spring steel	125,000

"Machinery's Handbook."

Table 25. Average Ultimate Strength of Common Materials Other than Metals

Pounds per square inch

Material	Compression	Tension
Bricks, best hard	12,000	400
Bricks, light red	1,000	40
Brickwork, common	1,000	50
Brickwork, best	2,000	300
Cement, Portland, one month old* (4½ gal. water per sack of cement)	4,000	400
Cement, Portland, one year old* (4½ gal. water per sack of cement)	6,000	600
Concrete, Portland cement* (7½ gal. water per sack of cement)	2,000	200
Concrete, Portland, one year old* (7½ gal. water per sack of cement)	3,500	350
Hemlock	4,000	6,000
Pine, shortleaf yellow	6,000	9,000
Pine, Georgia (longleaf)	8,000	12,000
Pine, white	5,500	7,000
White oak	7,000	10,000

"Machinery's Handbook" and (*) Portland Cement Association.

Table 26. Influence of Temperature on the Strength of Metals

Material	Degrees Fahrenheit							
	210	400	570	750	930	1100	1300	1475
	Strength of metal in per cent of the strength at 70°F.							
Wrought iron	104	112	116	96	76	42	25	15
Cast iron	...	100	99	92	76	42		
Cast steel	109	125	121	97	57			
Structural steel	103	132	122	86	49	28		
Copper	95	85	73	59	42			
Bronze	101	94	57	26	18			

"Machinery's Handbook."

Table 27. Strength of U. S. Standard Bolts from ¼ to 3 In. in Diameter

Bolt		Areas		Tensile strength, lb.			Shearing strength, lb.			
							Full bolt		Bottom of thread	
Diam. of bolt, in.	No. of threads per in.	Full bolt, sq. in.	Bottom of thread, sq. in.	At 10,000 lb. per sq. in.	At 12,500 lb. per sq. in.	At 17,500 lb. per sq. in.	At 7,500 lb. per sq. in.	At 10,000 lb. per sq. in.	At 7,500 lb. per sq. in.	At 10,000 lb. per sq. in.
¼	20	0.049	0.027	270	340	470	380	490	200	270
5/16	18	0.077	0.045	450	570	790	580	770	340	450
⅜	16	0.110	0.068	680	850	1,190	830	1,100	510	680
7/16	14	0.150	0.093	930	1,170	1,630	1,130	1,500	700	930
½	13	0.196	0.126	1,260	1,570	2,200	1,470	1,960	940	1,260
9/16	12	0.248	0.162	1,620	2,030	2,840	1,860	2,480	1,220	1,620
⅝	11	0.307	0.202	2,020	2,520	3,530	2,300	3,070	1,510	2,020
¾	10	0.442	0.302	3,020	3,770	5,290	3,310	4,420	2,270	3,020
⅞	9	0.601	0.419	4,190	5,240	7,340	4,510	6,010	3,150	4,190
1	8	0.785	0.551	5,510	6,890	9,640	5,890	7,850	4,130	5,510
1⅛	7	0.994	0.693	6,990	8,660	12,130	7,450	9,940	5,200	6,930
1¼	7	1.227	0.890	8,890	11,120	15,570	9,200	12,270	6,670	8,900
1⅜	6	1.485	1.054	10,540	13,180	18,450	11,140	14,850	7,910	10,540
1½	6	1.767	1.294	12,940	16,170	22,640	13,250	17,670	9,700	12,940
1⅝	5½	2.074	1.515	15,150	18,940	26,510	15,550	20,740	11,360	15,150
1¾	5	2.405	1.745	17,450	21,800	30,520	18,040	24,050	13,080	17,440
1⅞	5	2.761	2.049	20,490	25,610	35,860	20,710	27,610	15,370	20,490
2	4½	3.142	2.300	23,000	28,750	40,250	23,560	31,420	17,250	23,000
2¼	4½	3.976	3.021	30,210	37,770	52,870	29,820	39,760	22,660	30,210
2½	4	4.909	3.716	37,160	46,450	65,040	36,820	49,090	27,870	37,160
2¾	4	5.940	4.620	46,200	57,750	80,840	44,580	59,400	34,650	46,200
3	3½	7.069	5.428	54,280	67,850	94,990	53,020	70,690	40,710	54,280

Marks, "Mechanical Engineers' Handbook," McGraw-Hill.

Table 28. Safe Load for Ropes and Chains*

(In pounds)

Caution: When handling molten metal, wire ropes and chains should be 25 per cent stronger than indicated in table. Manila rope should not be used for this purpose.

NOTE: If three-leg slings are used, the safe load given for double (two-leg) slings may be increased 50 per cent and four-leg slings 100 per cent.		When used for a straight pull	Double (two-leg) sling		
			When used at 60° from the horizontal	When used at 45° from the horizontal	When used at 30° from the horizontal
	Diam. in.	Lb.	Lb.	Lb.	Lb.
Plow steel Wire rope (6 strands of 19 wires) For crucible steel rope, reduce loads one-fifth.	¼	1,060	1,835	1,500	1,060
	⅜	2,300	3,980	3,250	2,300
	½	4,000	6,920	5,650	4,000
	⅝	6,200	10,730	8,770	6,200
	¾	9,200	15,900	13,000	9,200
	⅞	11,600	20,000	16,400	11,600
	1	15,200	26,300	21,500	15,200
	1⅛	18,800	32,500	26,600	18,800
	1¼	24,000	41,500	33,980	24,000
	1⅜	28,000	48,500	39,600	28,000
	1½	32,000	55,400	45,250	32,000
	Diam. of iron, in.	Lb.	Lb.	Lb.	Lb.
Crane chain (Best grade of wrought iron, hand-made, tested short-link chain)	¼	1,060	1,835	1,500	1,060
	⅜	2,385	4,130	3,370	2,385
	½	4,240	7,345	6,000	4,240
	⅝	6,630	11,485	9,375	6,630
	¾	9,540	16,525	13,500	9,540
	⅞	12,960	22,450	18,325	12,960
	1	16,950	29,350	23,975	16,950
	1⅛	20,040	34,700	28,350	20,040
	1¼	24,750	42,875	35,000	24,750
	1⅜	29,910	51,800	42,300	29,910
	1½	35,600	61,650	50,350	35,600
	Diam. in.	Lb.	Lb.	Lb.	Lb.
Manila rope (Best long-fiber grade)	⅜	120	200	170	120
	½	250	420	350	250
	⅝	360	600	500	360
	¾	520	880	720	520
	⅞	620	1,040	840	620
	1	750	1,250	1,050	750
	1⅛	1,000	1,700	1,400	1,000
	1¼	1,200	2,050	1,700	1,200
	1½	1,600	2,700	2,200	1,600
	1¾	2,100	3,600	3,000	2,100
	2	2,800	4,800	4,000	2,800
	2½	4,000	6,800	5,600	4,000
	3½	6,000	10,200	8,400	6,000

---

* Based on table prepared by National Founders' Association.

Definitions, Cautions, and Instructions Governing the Purchase and Use of Chain

Issued by United States Chain and Forging Company, Pittsburgh, now the McKay Co.

Proof Test. The *proof test* is applied to chain for the purpose of detecting defects in material or manufacture.

The test is applied in a standard chain testing machine and shows the load in pounds which the chain, in the condition and at the time it left the factory, has withstood, under a test in which the load has been applied in direct tension to a straight length of chain with a uniform rate of speed represented by a separation of the tester heads not exceeding 10 in. per minute.

Average Ultimate Load. The *average ultimate load* is the load in pounds at which the chain, in the condition and at the time it left the factory, has been found by experience to break, under a test in which the load is applied in direct tension to a straight length of chain with a uniform rate of load represented by a separation of the tester heads not exceeding 10 in. per minute.

Safe Working Load. The *safe working load* is the maximum load in pounds which, at any time or under any condition, should ever be applied to the chain, even when the chain is in the same condition it was when it left the factory, and when the load is applied in direct tension to a straight length of chain.

Cautions. The above terms "proof test," "average ultimate load," and "safe working load," contain no implication of what load the chain will safely withstand if any of the above factors are changed.

Any change in the above factors, such as twisting of the chain, deterioration of the chain by strain, by usage, by weathering, or by lapse of time, or acceleration in the rate of application of the load, or variation in the angle of the load to some sharper angle resulting from the configuration or structure of the material constituting the load, will lessen the load that the chain will safely withstand.

Instructions Regarding Attachments. Where attachments, such as hooks or rings, are desired for use with chain in sustaining loads, care should be taken to select the attachments of the type, grade, and size recommended herein for use with the type, grade and size chain with which such attachments are to be used.

When this is done, the recommended safe working load of such chain, in the sense that the term is defined above, will apply to the chain and attachments thereon when used together.

When chain and attachments thereon are used together, it is impossible to state what load will be sustained, if the attachments thereon are not of the type, grade and size recommended for use with the particular type, grade, and size of chain used.

Purchasers will please note that all the cautions above set forth apply not only to the use of chain but also to the use of attachments thereon.

SUGGESTIONS

Take up slack and then start load slowly.
Keep chains free from twists, knots and kinks.
Lift from center of hooks, never from the point.
Distribute the load evenly on all legs.
Inspect your sling chains regularly.

Table 29. Maximum Free Air That Can Be Supplied in Cubic Feet per Minute to the Pressure Vessel for Different Sizes of Safety Valves at Stated Pressures*

Diam. of valve, in.	Gage pressure, lb. per sq. in.															
	50	100	150	200	250	300	350	400	500	600	800	1000	1200	1600	2000	2400
¼	...	...	...	....	....	....	....	53	61	70	84	97	109	128	147	160
½	20	32	42	51	59	67	74	111	129	147	177	205	230	270	304	330
¾	37	59	78	96	112	127	141	176	224	232	242	346	386	423	474	518
1	58	94	124	152	178	202	224	248	286	324	390	450	500	586		
1¼	84	135	180	221	259	293	325	...	374	...	509					
1½	114	186	248	302	354	400	444	...	472	...	634					
2	189	306	410	501	592	668	741									
2½	282	457	613	750	880	998	1114									
3	393	638	856	1050	1230	1398	1557									

The foregoing table is based on the following formulas:

$$Q = 28\,PDl \text{ for } 45^\circ \text{ bevel-seat valves}$$
$$Q = 40\,PDl \text{ for flat-seat valves}$$

where Q = discharge in cu. ft. of free air per min.
P = absolute pressure at which the safety valve opens (gage pressure + 14.7 lb. at sea level).
D = diam., in., of the inside edge of the bearing surface between the disk and seat.
l = vertical lift of the safety-valve disk from its seat, in., representing lift for minimum discharge capacity for satisfactory operation of the valve.

* From A.S.M.E. Boiler Construction Code. By permission of American Society of Mechanical Engineers.

Table 30. Identification of Piping Systems

Class of Material	Color
Fire protection	Red
Dangerous	Yellow
Safe	Green
Protective	Blue
Extra valuable	Purple

For subdivisions of these classifications see "Scheme for the Identification of Piping Systems," American Standards Association, New York, N. Y.

Table 31. Coefficients of Expansion*

(See also p. 434)

The following coefficients of expansion, per degree Fahrenheit, of the principal flammable liquids shall be used in determining outages:

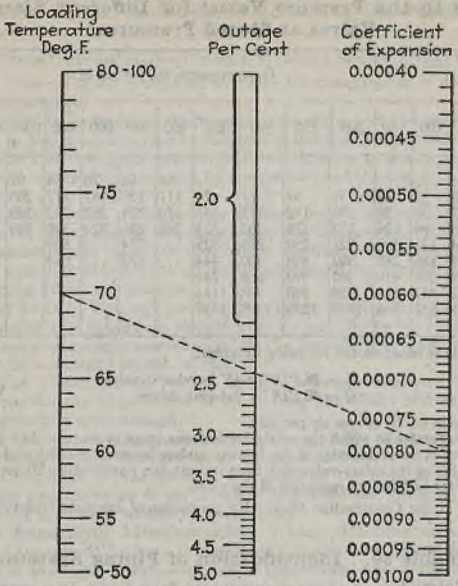
Acetone	0.00085	Gasoline (°A.P.I.*)	
Amyl acetate	0.00068	50 –55	0.00055
Benzol (benzene)	0.00071	55.1–60	0.00060
Carbon bisulphide	0.00070	60.1–65	0.00065
Ether	0.00098	65.1–70	0.00070
Ethyl acetate	0.00079	70.1–75	0.00075
Ethyl (grain) alcohol	0.00062	75.1–80	0.00080
Methyl (wood) alcohol	0.00072	80.1–85	0.00085
Toluol (toluene)	0.00063	85.1–90	0.00090

† °A.P.I. (American Petroleum Institute), according to the following formula:

$$^\circ\text{A.P.I.} = \frac{141.5}{\text{specific gravity}} - 131.5$$

* By permission, from regulations of the Interstate Commerce Commission, Washington, D. C.

Table 32. Outage Chart for Flammable Liquids



Example: Suppose the temperature of the liquid at time of loading is 70°F. and its coefficient of expansion is 0.00080. Lay a ruler on the chart running from 70°F. to 0.00080 as shown by the dotted line and the required outage is 2.4 per cent where the ruler crosses the outage scale. "Outage" is the percentage of space in the interior of the container which must not be filled with liquid, in order to allow for the expansion of the liquid and the maximum temperature to which it will be subjected in transit. (From regulations of the Interstate Commerce Commission, Washington, D. C., by permission.)

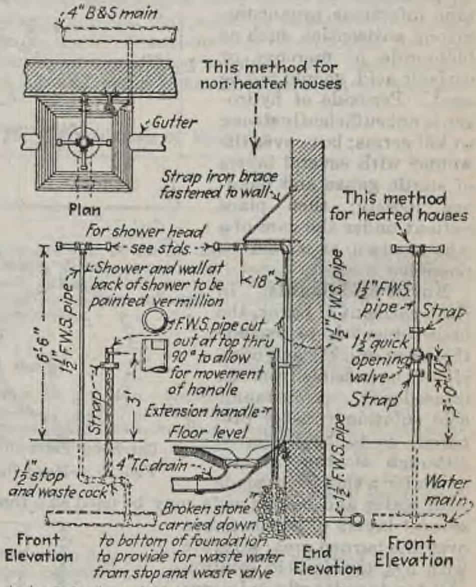
FIRST AID

Every work place should provide reasonable first aid in order that injured persons may be properly cared for in emergencies and until further treatment can be secured. Where full-time plant physicians are not employed, small industrial first-aid units may be provided, which should be placed under the care of competent persons who should be available at all times and who should have received first-aid training in a standard first-aid course given under the direction of either the American Red Cross or the U. S. Bureau of Mines.

The following outline is recommended merely as **first aid.** In all cases a physician should be summoned at once.

Treatment for Shock. Keep the patient warm. Keep head lower than feet. Cover with blankets or coats. Put hot water bag to feet. Give stimulants, such as hot coffee, hot tea, or teaspoonful of aromatic spirits of ammonia in a half glass of water.

Treatment for Sunstroke and Heat Exhaustion. First take the temperature with a clinical thermometer. Sunstroke, or thermic fever, is characterized by rise in body temperature and the immediate treatment is to reduce the fever by the application of something cold to the head, such as ice bag or cold compresses (cloths wrung out in cold water). Put the patient into a tub of cold water (after first removing the clothing), or sponge the body with cold water.



Note: Spigots or bibb cocks used for domestic purposes shall be located at or near the shower

FIG. 30.—Detail of emergency water shower.

Heat exhaustion is the same as shock, and the body temperature is below normal. Keep the patient covered with blankets and put a hot water bag to the feet. Give stimulants to drink, such as strong tea or coffee or aromatic spirits of ammonia.

Treatment for Fainting. Place patient in lying-down position with head lower than the rest of the body so that the brain will receive more blood. Loosen clothing around neck—see that there is plenty of cold air. Sprinkle face and chest with cold water. Put smelling salts or ammonia to nose. Rub limbs toward body. Give stimulants when patient can swallow.

Treatment for Sprains. Absolute rest until doctor arrives so as not to do more damage. Do not allow joint to be used—elevate it if possible and apply cold. Wring cloths in cold water and apply to joint or shower the joint with very cold water.

Treatment of Ordinary Wounds. Do not allow the wound to be touched—exposure to the air is much safer than the application of dressings which are not surgically clean. Ordinary water is dangerous as it may contain infectious organisms. Strong antiseptics, such as bichloride of mercury or carbolic acid, should not be used. Peroxide of hydrogen is not sufficiently strong to kill germs; but cover the wound with several layers of sterile gauze and bandage loosely; then place patient under the care of a physician as soon as possible.

Burns and Scalds. In the treatment of burns, the main object is to exclude the air as quickly and completely as possible from the injured part. A 5 % tannic acid solution (freshly prepared) is to be preferred although if this is not available a thin paste made with water and baking soda may be used as a temporary application. Place several layers of sterile gauze over the burned area; saturate with 5 % tannic acid solution and bandage loosely. In case of serious burns on the body, remove all clothing and treat as recommended above. If signs of collapse are shown, keep the patient warm and give hot stimulants if not unconscious.

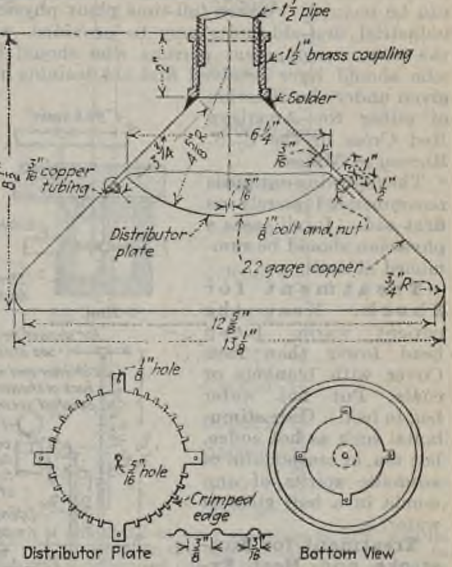


FIG. 31.—Head for emergency water shower.

Acid and Alkali Burns. With either, wash off, as quickly as possible, with a large quantity of clean water. For this purpose, emergency showers should be provided. See Figs. 30, 31 and 32. The treatment following this for either acid or alkali burns is similar to that for ordinary burns.

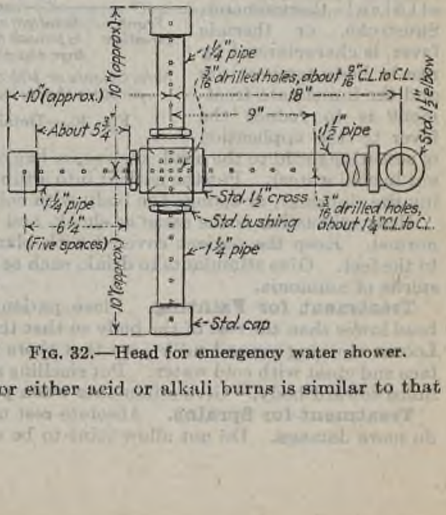


FIG. 32.—Head for emergency water shower.

When emergency showers are not available, water from a tap or from an ordinary drinking fountain may be used; the latter is particularly advantageous in either acid or alkali burns of the eyes.

Gas Poisoning. In all cases of gas poisoning, particularly from nitrous fumes, hydrogen sulfide, carbon monoxide, etc., even though the persons show no apparent effects, they should be placed under the care of a physician. These cases frequently do not appear serious until several hours (even 24 to 48 hr.) after the exposure. From the time of the exposure, until a physician is secured, persons affected by gas poisoning should avoid any unnecessary activity and should be kept as quiet and composed as possible—preferably put to bed. When a person is rendered unconscious by exposure to a gas and breathing has ceased, artificial respiration must be used immediately. Remove the patient to fresh air and begin artificial respiration at once and send for a physician.

Table 33. Antidotes of Poisons

Poison	Symptom	Antidote
Corrosive Poisons: Strong acids (sulphuric, hydrochloric, nitric), also strong alkalis, such as caustic soda, lime, and potash	Burn and stain on the lips and mouth. Burning pain in throat and stomach; straining and vomiting; perhaps suffocation; always shock	For **acids** give baking soda, magnesia, lime water or soap-suds For **alkalis** give vinegar, lemon juice. Also, in both poisons give large doses of olive oil, milk, eggs, or flour and water, to dilute and soothe the parts, and then induce vomiting
Irritant Poisons: Bichloride tablets, Paris green, sugar of lead, arsenic, phosphorus, poisonous plants, cyanides	These do not stain. Metallic taste in mouth; burning in throat and stomach; vomiting and purging	Dilute and soothe with large quantities of salad oil, milk, eggs, soap-suds, starch, or flour water. Then induce vomiting with salt water or mustard and water. Then stimulants like strong coffee or tea, or spirits of ammonia (No oil in phosphorus poisoning)
Nerve Poisons: Opium, morphine, paregoric, etc.	Produce deep sleep, pupils of eyes small; insensibility; face flushed	Induce vomiting if poison is just taken. Keep awake by strong coffee; slapping with wet towels; walking the victim supported on each side, and **artificial respiration**
Strychnine (the most common), belladonna, prussic or hydrocyanic acid, etc.	Convulsions, twitching, delirium, and suffocation	Induce vomiting at once, or tickle the back of throat with the finger; artificial respiration if breathing stops; stimulants
Iodine........................	Staining, burning, etc.	Two tablespoons of starch in a tea cup of water, either cornstarch or laundry starch; produce vomiting
Phenol........................	Vomiting and great pain; skin covered with cold sweat. Very rapid poison. Lips, tongue, and mouth are burned white by pure, and black by impure (phenol)	A couple of tablespoons of either Epsom or Glauber's salts in water. Lime water is not so effective. Three or four raw eggs or castor oil. Give stimulant
Nitrate of silver..................	....	Salt and water

Special Poisons. Certain materials such as aniline, benzol, etc., may be absorbed through the respiratory passages, or produce poisoning when taken internally, or even be absorbed through the skin in sufficient quantity to produce serious effects. Where materials of this nature are handled, a first-aid treatment should be developed. This should be done by the physician who regularly attends the plant.

Artificial Respiration. To be used in cases of drowning, gas poisoning, and electric shock where breathing has ceased.

Administration (Fig. 33). Start at once. The first few minutes are valuable.

1. Clean out patient's mouth.
2. Place patient as shown in 1.
3. Straddle patient as shown in 1.
4. Place hands as in 1.
5. Apply pressure as in 2.
6. Release as in 3.
7. The complete stroke should be carried on 12 to 15 times per minute.
8. Do not give up—even after 4 hr. men have been revived.

If other persons are present, send one of them for a doctor. Others should rub the patient's arms and legs, apply hot water bottles, cover the patient with blankets, and allow him to inhale aromatic spirits of ammonia. If an inhalator (a device for supplying oxygen to the patient) is on hand, someone may apply the device. In no case should artificial respiration be interrupted in order to apply the inhalator or for any other purpose until breathing is restored.

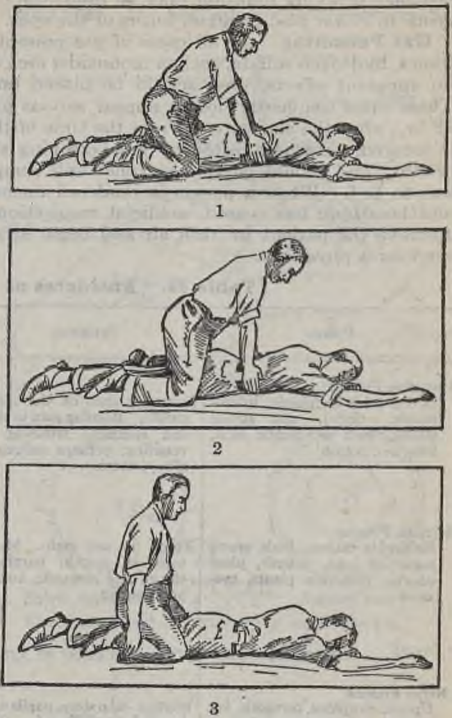
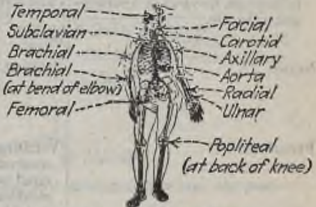


Fig. 33.—Positions for administering artificial respiration. (*By permission of The National Electric Light Association.*)

Arterial Bleeding. Arterial bleeding is recognized by bright red blood expelled in spurts. It is serious because of the rapid loss of blood, which is due to the fact that there is too much pressure at the wound to allow the blood to clot. The flow of blood cannot be stopped by



Courtesy of the U.S. Bur. Mines

Fig. 34.—Pressure points for control of arterial bleeding.

ordinary pressure over the wound. To control arterial bleeding, press with your fingers or thumb on the artery between the bleeding point and the heart. The points where the arteries come close to the surface of the skin are shown in Fig. 34. A tourniquet should be applied wherever possible at these points to control bleeding in the particular artery. The tourniquet should be released for 3 or 4 sec. at 20-min. intervals.

OCCUPATIONAL HAZARDS AND THEIR SYMPTOMS[1]

Occupational hazards can be controlled by one of several methods, although the particular method used may depend upon the type of hazard and its severity. It is always best to control the hazard at its source. This can be accomplished by either closed systems or ventilation. Masks, respirators, goggles, helmets, etc., are used where necessary and where the exposure is temporary and not severe.

Table 34

Health Hazards	Symptoms or Effects
Abnormalities of temperature Extreme dry heat	Anemia, general debility, catarrh, stiff joints, cramps, lumbago, Bright's disease, skin eruptions, premature old age, cataracts, retinitis, conjunctivitis
Compressed air	Weakness, vertigo, pains in the back and legs, paralysis of legs and arms, painful constriction of the chest, cerebral hemorrhage and aphasia, coma, subcutaneous hemorrhages, impairment of hearing
Dampness	Diseases of the respiratory passage, neuralgic and rheumatic affections
Dust Inorganic dust	Cough, dyspnea, pleuritic pains, hemoptysis, clubbed fingers, marked flatness of chest, deficient expansion (bilateral), dullness, diminished resonance, mucous rales, fibrosis, chronic inflammatory condition of eyes, ears, nose, and throat; colds, chronic catarrh of respiratory tract, pleurisy, pneumoconiosis
Organic dust	Dryness of nose, throat, and mouth; cough, anaphylaxis, asthma, bronchitis, emphysema
Extreme light	Cataracts, retinitis, conjunctivitis, dermatitis, ulceration and exfoliation of the skin, electrical ophthalmia, cancer
Infections Anthrax External	1. Malignant pustule. Begins as inflamed pimple or boil. Papule becomes hard, with a purple center and deep red zone of infiltration surrounding, appearance of minute vesicular areola. Central papule becomes vesicular, discharges thick bloody serum, later forming a brown gangrene. A painful lymphangitis with hard edema extending over neck and arm. Local phlebitis in the edematous area, chilliness, anorexia, vomiting, prostration, high temperature, feeble pulse 2. Malignant edema. A spreading inflammation of loose connective tissue accompanied by sloughing and gangrene. Constitutional symptoms, those of pyemia

---

[1] By permission of the McGraw-Hill Book Company; condensed from Lange, "Handbook of Safety and Accident Prevention," and the "International Critical Tables."

Table 34—(*Continued*)

Health Hazards	Symptoms or Effects
Infections (*Continued*)	
Internal	High fever, pains in head and back, vomiting, constipation, pain and tenderness in the abdomen, rapid feeble pulse, palpable spleen, dyspnea, cyanosis. May be hemorrhage from bowels. When lungs are involved, there are additional symptoms: cough, pain in chest, suffocation
Hookworm	Anemia, pallor of the face even when the blood count is not very low; a dull, heavy, listless expression, manner, speech, and gait; increasing muscular weakness; occurrence of parasites in stool. Victims often complain of gastro-intestinal pains and cramps. In exaggerated cases there are edema, ascites, progressive emaciation, protuberant abdomen, and increasing stupor
Septic infections	Skin infections such as boils, carbuncles, blood poisoning, localized lymphangitis or cellulitis
Poor illumination	Nystagmus, eye strain, deficient vision due to astigmatism or hyperopia, headache, giddiness (Eye strain contributes to neurasthenia)
Repeated motion, pressure, shock, etc	Pain of muscle used, set up by a myositis, bursitis, synovitis, or other local changes of a chronic inflammatory nature; trembling, gradual emaciation, and partial paralysis of parts; acroparesthesia
Poisons	
Acetaldehyde	Irritation of the mucous membranes of the nose, larynx, bronchi, and eyes; acceleration of the heart's action; profuse night sweats
Acridine	Irritation and inflammation of skin and mucous membranes; severe burning and itching of the skin; violent sneezing
Acrolein	Itching in the throat; irritation of the eyes, exciting lachrymation; conjunctivitis, irritation of the air passages, bronchial catarrh
Ammonia	Acute inflammation of the respiratory organs; cough, edema of the lungs, chronic bronchial catarrh, redness of the eyes, increased secretion of saliva, retention of the urine
Amyl acetate	Nervous symptoms, headache, fullness of the head, giddiness, numbness, nausea, disturbances of digestion, palpitation of the heart, inflammation of the respiratory organs
Amyl alcohol	Congestion of the head, oppression of the chest, irritation of the air passages, lowering of the blood pressure, faintness, nausea
Aniline and other amino compounds of benzol and its homologues	Pallor of the skin, vertigo, unsteady gait, loss of appetite, increased frequency of respiration, anemia, slowing of the pulse, eczematous eruptions, bloody urine, spasmodic muscular pains, cyanosis
Antimony and its compounds	Itching eruptions of the skin; inflammation of the mouth, throat, and stomach; albumin in the urine, weakness of the heart, vertigo, faintness, coryza, dyspepsia, intestinal colic, nephritis

Table 34—(*Continued*)

Health Hazards	Symptoms or Effects
Poisons (*Continued*)	
Arsenic and its compounds..........	Headache, melancholia, sleeplessness, gastric disturbances, emaciation, catarrh of the mucous membranes, skin diseases of various forms, falling out of the hair and nails, melanosis, bleeding gums, peripheral multiple neuritis, paralysis
Arsenic (Arseniureted hydrogen).....	General malaise, difficulty of breathing, fainting fits, gastric disturbances, jaundice, bluish discoloration of the mucous membrane, pain in the region of the spleen and kidney, darkened urine, fetor of the mouth resembling garlic
Benzine (Gasoline)................	Headache, vertigo, nausea, cough, irregular respiration, weakness of the heart, drowsiness, cyanosis, twitching of the muscles, psychosis, skin lesions, asphyxiation
Benzol (Benzene)..................	Acute (chronic) headache, vertigo, anemia, muscular tremor, scarlet lips, spots of extravasated blood in the skin, irritant cough, fatty degeneration of liver, kidneys, and heart; blood symptoms (leucopenia), low count of red and white corpuscles and low hemoglobin
Brass (Zinc)......................	Headache, general malaise, throat irritation, cough, nausea, vomiting, constipation, trembling, muscular pains, accelerated respiration, profuse sweating, deposit of green tartar on the teeth, metallic taste in the mouth, anemia, premature old age, respiratory and degenerative diseases. *Zinc chill* which is influenced very little by heat, followed by deep sleep and profound physical depression
Bromine*.........................	Pallid countenance, emaciation, bronchial irritation and asthma, gastric disturbances, irritation of skin. Irrespirable gas causing pulmonary edema.
Carbon dioxide....................	Anemia, cyanosis, headache, drowsiness, vertigo, ringing in the ear, and general nervousness
Carbon disulfide..................	Acute stupefaction, intoxication, pallor, relaxation of muscles, reflexes absent. Chronic headache, pain in the extremities, trembling, deafness, reduction of the reflexes, acceleration of the heart's action, nausea, digestive trouble, emaciation, disturbance of sense of vision, excitement and violent temper followed by depression, hyperstimulation of sexual instinct, later its abnormal decline, chronic dementia
Carbon monoxide...................	Headache (usually frontal), dizziness, sense of fullness of the head, fatigue, nausea, general weakness, polycythemia, rapid asphyxiation
Carbon tetrachloride*.............	Nausea, vomiting, abdominal pain, stupor deepening into coma, absence of reflexes, clonic convulsions, weak pulse, increased temperature and death
Chloride of lime..................	Irritating cough, inflammation of upper air passages, difficulty of breathing, bronchitis, asthma, sometimes hemoptysis, conjunctivitis, lachrymation, hyperhidrosis, burning eruption on the skin

Table 34—(*Continued*)

Health Hazards	Symptoms or Effects
Poisons (*Continued*)	
Chlorine	Pallid countenance, emaciation, bronchial irritation and asthma, gastric disturbances, irritation of the skin, chlorineacne, respiratory inflammation
Chloroform*	In anesthesia the untoward symptoms are shallow or irregular respiration, sudden cessation of respiration, pulse either very slow or very rapid, dilation of pupils, cyanosis, asphyxia leading to dilation of the heart, vagus stimulation, and finally failure of heart due to asphyxial condition. In delayed poisoning there are great prostration, delirium, coma, death
Chloropicrin*	Lachrymatory and respiratory irritant, with specific action on the vomiting center. Causes coughing, nausea, vomiting, and, in large quantities, unconsciousness. Secondary effects are bronchitis, shortness of breath
Chlorodinitrobenzol	See Nitrobenzol
Chloronitrobenzol	See Nitrobenzol
Chromium compounds	Pitlike, phagedenic ulcers, very difficult to heal and very painful; perforation of the nasal septum at the cartilaginous portion, irritation of the conjunctiva, small areas of inflammation in the lungs, inflammation of the kidneys, chronic gastritis, anemia
Cyanogen compounds	Acute headache. Immediate asphyxia with unconsciousness, cyanosis and death. Chronic headache, vertigo, unsteadiness of gait, nausea, loss of appetite, disturbance of gastric and intestinal functions, slowing of the pulse, albuminuria
Dichlorodiethyl sulfide* (Mustard gas)	Conjunctivitis and superficial necrosis of the cornea; hyperemia, edema, and, later, necrosis of the skin, leading to skin lesion of great chronicity; congestion and necrosis of epithelial lining of the trachea and bronchi. Systemic effects due to the absorption of the substance into the blood stream and its distribution to the various tissues of the body
Dimethyl sulfate	Strongly corrosive effect on the skin and mucous membranes, hoarseness, lachrymation, conjunctivitis, edema, photophobia
Dinitrobenzol	See Nitrobenzol
Hydrochloric acid	Irritation of mucous membranes, conjunctivitis, coryza; pharyngeal-, laryngeal-, and bronchial-catarrh; dental caries
Hydrogen cyanide* (Prussic acid)	Headache, vertigo, unsteadiness of gait, nausea, loss of appetite, disturbance of gastric and intestinal functions, slowing of pulse, albuminuria
Hydrogen sulfide*	Poisoning is of two types—acute and subacute—causing asphyxiation and irritation (conjunctivitis); bronchitis, pharyngitis and depression of the central nervous system, respectively. In low concentration the symptoms are headache, sleeplessness, dullness, dizziness, and weariness; pain in the eyes followed by conjunctivitis is

Table 34—(*Continued*)

Health Hazards	Symptoms or Effects
Poisons (*Continued*)	fairly constant; bronchitis and pains in the chest are frequent; further poisoning produces depression, stupor, unconsciousness, and death. Spasms—clonic and tonic—are present. Death from asphyxia is caused by paralysis of respiratory center, while death from subacute poisoning is associated with edema of the lungs
Hydrofluoric acid	Intense irritation of the eyelids and conjunctiva; coryza; bronchial catarrh with spasmodic cough; ulceration of the nostrils, gums, and oral mucous membrane; painful ulcers of the cuticle; erosion and formation of vesicles; suppuration under the finger nails
Iodine	Inflammation of the lungs and pulmonary edema
Lead and its compounds	Pallor, metallic taste, nausea, anorexia, constipation, lead line on the gums, asthenia, lassitude, headaches, arthralgias and neuritis, weakness of grip, tremors of fingers and tongue; lead paralyses, especially of extensor muscles used most; atrophy of optic nerve; blood symptoms are low red and white count and low hemoglobin and presence of stipple cells
Mercury and its compounds	Ptyalism; swelling, inflammation, and bleeding of the gums; blue line on the gums, rodent ulcers, pallor, mercurial tremor, digestive disturbances, localized white spots in the mucosa surrounded by pale blue or reddened area, general weakness of the hand and digital extensors, foul breath, corrosion of the teeth, furunculosis, sleeplessness and depression or drowsiness and apathy, loss of energy and initiative
Methyl alcohol	Headache, nausea, abdominal cramps, ringing in the ears, muscular prostration, insomnia, delirium, difficulty of breathing, inflammation of the throat and mucous membrane of the air passages, conjunctivitis, serious affections of the retina and optic nerve resulting in blindness, fatty degeneration of the liver
Methyl bromide	Vertigo, headache, staring look, pallor of the skin, retarded pulse, constipation, excitability, trembling
Naphtha	Headache, vertigo, nausea, vomiting, dyspnea, palpitation, insomnia, hysteria
Nitraniline	See Aniline
Nitrobenzol and other nitro compounds of benzol and its homologues	Skin gradually becomes cyanotic, methemoglobin formation, general debility, anemia, presence of hematoporphyrin, albumin, and sometimes free poison in the urine; skin eruptions, jaundice, visual disturbances, dyspnea, odor of bitter almonds in breath
Nitroglycerin	Severe headache, vertigo, nausea, paralysis of the muscles of the head and eyes as well as of the lower extremities, cyanosis, reddening of the countenance, burning in the throat and stomach,

Table 34—(*Continued*)

Health Hazards	Symptoms or Effects
Poisons (*Continued*)	disturbances of digestion, trembling, neuralgia, colic, retarded respiration and heart action, obstinate ulcers under nails and on the finger tips, eruptions on the plantar aspect of the feet and interdigital spaces, with extreme dryness and formation of fissures
Nitronaphthalene..................	See Nitrobenzol
Nitrous gases and nitric acid.........	Irritation of air passages, cough, labored respiration, inflammation of the eyes, injury to the teeth, erosion and perforation of nasal septum
Petroleum..........................	Inflammation of the skin, acne, suppurating ulcers, papilloma, numbness and irritation of the Schneiderian membrane, headache and sensory disturbances, affections of the respiratory organs
Phenol..............................	Erosion of the skin, eczema, irritation of respiratory organs, digestive disturbances, symptoms of degeneration of the blood, emaciation, nephritis, gangrene, jaundice
Phenyl hydrazine...................	Vesicular eruptions on the skin with itching and burning, diarrhea, loss of appetite, granular degeneration of the blood corpuscles, formation of methemoglobin, a sense of general malaise
Phosgene...........................	Destruction of lung tissue, emphysema and edema, myocardial insufficiency due to the emphysema, pleural thickening and adhesions, chronic bronchitis, mild diffuse bronchiectasis, nocturnal dyspnea, polycythemia
Phosphine*.........................	Oppressed feeling in chest, headache, vertigo, ringing in the ears, general debility, loss of appetite, great thirst
Phosphorus.........................	Inflammation and sclerosis of the bones and of the periosteum, necrosis of the bones of the jaw, swelling and ulceration of the gums and buccal membrane, loosening and falling out of the teeth, suppuration and destruction of jawbone with fistulous channels burrowing through the cheek, meningeal inflammation, brittleness of bones, digestive disturbances, emaciation
Phosphorus trichloride*.............	Sensations of suffocation, difficulty of breathing, lachrymation, bronchitis, inflammation of lungs and edema, with frothy, blood-stained expectoration
Picric acid.........................	Itching, inflammation of the skin, vesicular eruptions, yellow pigmentation of epidermis and conjunctiva, inflammation of buccal mucous membrane, digestive disturbances, vertigo, jaundice, nasal catarrh, nephritis
Sulphur chloride....................	Symptoms are due to the combined effects of chlorine, hydrochloric acid, and sulfur dioxide. Sulfur chloride when in contact with moisture reacts with water to form these products. Suffocation, nausea and vomiting
Sulfur dioxide......................	Irritation of the mucous membrane of the respiratory organs and eyes, spasmodic cough, bronchial catarrh, digestive disturbances, blood-tinged mucus

Table 34—(*Concluded*)

Health Hazards	Symptoms or Effects
Poisons (*Concluded*)	
Sulfur trioxide*	Irritation of the respiratory organs; bronchitis
Sulfuric acid	Inflammation of respiratory organs; injury to teeth through softening of the dentine; chronic catarrh
Tar	Tar itch, diffuse acne, eczema or psoriasis, loss of appetite, nausea, diarrhea, headache, numbness, vertigo, albuminuria, edema, ischuria, conjunctivitis, bronchitis
Tetrachlorethane (Acetylene tetrachloride)	Abnormal sense of fatigue, profuse perspiration, general discontent and grouchiness, inability to concentrate, nocturia, slight polyuria, dreaming, headache, vertigo, nervousness, insomnia, loss of appetite, constipation, gas in stomach, general abdominal pain, nausea, eructations of gas, vomiting, loss of weight, jaundice, enlarged liver, bile in the urine, abdominal tenderness, increase of mononuclear cells, appearance of many immature large mononuclears, elevation in the white count, slight increase in number of platelets
Toluidine*	Headache, weakness, difficulty in breathing, cyanosis, convulsions, psychic disturbances, air hunger, marked irritation of renal organs
Trinitrotoluol	Nose and throat irritation, obstinate cough, bluish color of the lips and lobes of the ears, yellowing of the whites of the eyes, expectoration of yellow mucus, discoloration—a mixture of lividity and jaundice, rash on the skin, shortness of breath, anemia, palpitation of the heart, bile-stained urine, rapid weak pulse
Turpentine	Irritation of the mucous membrane of the eyes, nose, and upper air passages; cough, bronchial inflammation; salivation; giddiness, headache, irritation of the kidneys, odor of violets in urine, severe irritation of the skin, eczema and hardening of the epidermis

Condensed from Lange, "Handbook of Safety and Accident Prevention," McGraw-Hill, except those marked (*), which are condensed from the "International Critical Tables," Vol. II, McGraw-Hill.

Bibliography

"American Engineering and Industrial Standards" may be secured from The American Standards Association, 29 West Thirty-ninth Street, New York, N. Y.

A 2–1926	Fire Tests of Building Construction and Materials
A 9–1929	Building Exits Code
A 11–1930	Lighting: Factories, Mill and Other Work Places, Code of
A 12	Floor and Wall Openings, Railways and Toe Boards, Safety Code for (published for trial and criticism)
A 13–1928	Identification of Piping Systems
A 14–1923	Ladders, Safety Code for
A 17–1925	Elevators and Escalators, Safety Code for (Under revision)
B 7–1930	Abrasive Wheels, Safety Code for the Use, Care, and Protection of
B 8–1922	Foundries, Safety Code for the Protection of Industrial Workers in
B 9–1930	Mechanical Refrigeration, Safety Code for
B 11–1926	Power Presses and Foot and Hand Presses, Safety Code for
B 15–1927	Mechanical Power Transmission Apparatus, Safety Code for
B17c–1927	Transmission Shafting, Code for Design of
B 24–1927	Forging and Hot Metal Stamping, Safety Code for
B 26–1925	Screw Threads for Fire Hose Couplings

B28a–1927	Rubber Mills and Calenders, Safety Code for
C 1–1930	Electric Wiring and Apparatus in Relation to Fire Hazard (National Electric Code)
C 2–1927	National Electrical Safety Code
	Discussion of National Electrical Safety Code C2–1927
C 5–1929	Lightning, Code for Protection against
	Preliminary Report of the Sectional Committee on Protection of Electrical Circuits and Equipment against Lightning
C 19–1928	Industrial Control Apparatus, Definitions, Classifications, Rating and Methods of Test for
D 3–1927	Colors for Traffic Signals, Safety Code for
D 5	Street Traffic Signs, Signals & Markings (published for trial and criticism)
K 2–1927	Gas Safety Code
K 8–1923	Flash Point of Volatile Flammable Liquids, Method of Test for
K 13–1930	Colors for Gas Mask Canisters, Code on
L 1–1929	Textiles, Safety Code for
M 2–1926	Installing and Using Electrical Equipment in Coal Mines, Safety Rules for
M 11–1927	Wire Ropes for Mines, Specifications for and Recommended Practice in the Use of
M 12–1928	Ladders and Stairs for Mines, Recommended Practice for the Construction and Maintenance of
M 13–1925	Rock Dusting of Coal Mines, Recommended Practice for
M 17–1930	Fire Fighting Equipment in Metal Mines
O 1–1930	Woodworking Plants, Safety Code for
P 1–1925	Paper and Pulp Mills, Safety Code for
X 2–1922	Protection of the Heads and Eyes of Industrial Workers, Safety Code for the
Z 8–1924	Laundry Machinery and Operations, Safety Code for
Z12a–1930	Installation of Pulverized Fuel Systems, Safety Code for
Z12b–1930	Pulverizing Systems for Sugar and Cocoa, Safety Code for
Z12c–1927	Prevention of Dust Explosions in Starch Factories, Safety Code for
Z12d–1928	Prevention of Dust Explosions in Flour and Feed Mills, Safety Code for
Z12e–1928	Prevention of Dust Explosions in Terminal Grain Elevators, Safety Code for

Safe Practices Pamphlets

Published by National Safety Council, 20 North Wacker Drive, Chicago, Ill.

General

1 Ladders—1939.
2 Stairs, Stairways, Inclines and Ramps—1934.
3 Safe Operation of Steam Boilers—1939.
4 Overhead Traveling Cranes—1939.
5 Mechanical Power Transmission Apparatus (Part I) Starting and Stopping Devices—1934.
6 Fiber Rope—1936.
7 Mechanical Power Transmission Apparatus (Part II) Belts and Belt Guards—1937.
8 Mechanical Power Transmission Apparatus (Part III) Shafting, Coupling, Keys, Collars, Set Screws, Pulleys, Gears, Sprockets and Chains—1937.
9 Mechanical Power Transmission Apparatus (Part IV) Prime Movers: Flywheels, Cranks, Connecting Rods, Crossheads, Tail Rods, Governors—1937.
10 Mechanical Power Transmission Apparatus (Part V) Bearings, Oilers and Oiling Devices—1937.
11 Floors and Flooring—1939.
12 Wood Scaffolds—1937.
13 Grinding Wheels—1936.
14 Goggles—1940.
15 Elevators—1935.
16 Protective Clothing—1930.
17 Plant Yards and Grounds—1938.
18 Power Presses, 1929.
19 Exits, Fire Alarms and Fire Drills—1929.
20 Woodworking Machinery and Equipment—1939.
21 Standard Industrial Injury Reporting System—1939.
22 Industrial Shop Lighting—1937.

23 Gas and Electric Welding—1936.
24 Fire Extinguishment—1939.
25 Acids and Caustics—1936.
26 Wire Rope—1937.
27 Industrial Sanitation (Drinking Water, Wash and Locker Rooms and Toilet Facilities)—1936.
28 Commercial Explosives—1939.
29 Electric Equipment in Industrial Plants—1939
30 Discontinued.
31 Fire Causes and Prevention—1939.
32 Exhaust Systems—1939.
33 Hoisting Apparatus—1939.
34 Industrial Explosion Hazards Gases, Vapors, Flammable Liquids, and Dusts—1930.
35 Conveyors—1939.
36 Fire Brigades—1929.
37 Industrial Ventilation—1939.
38 Safety Posters and Bulletin Boards—1939.
39 Machine Shops—1939.
40 Suggestion Systems—1930.
41 Hand Tools—1939.
42 Organizing a Complete Industrial Safety Program—1939.
43 Combined with Pamphlet 15.
44 The Safe Use of Cutting Oils and Emulsions—1931.
45 Industrial Housekeeping—1930.
46 Fuel Handling, Storing and Firing—1929.
47 Compressed Air Machinery and Equipment—1939.
48 Railroads in Industrial Plants—1931.
49 Construction and Equipment of Steam Boilers—1939.
50 Practical Methods of Reducing Fatigue—1940.
51 Discontinued.
52 Static Electricity—1937.
53 Checking Plans and Specifications for Safety—1938.
54 Handling Material (Hand and Truck)—1938.
55 Industrial Power Trucks and Tractors—1940.
56 Investigation of Industrial Accidents—1939.
57 Replaced by Health Practices Pamphlet 5.
58 Mechanical Power Transmission Apparatus (Part VI) Construction of Machinery Guards—1937.
59 Warehouses and Shipping Rooms—1934.
60 Chemical Laboratories—1937.
61 Mechanical Refrigeration—1937.
62 Discontinued.
63 Storage Tanks for Oils, Acids and Dry Materials—1930.
64 Respiratory Protective Equipment—1938.
65 Teaching Safety to New Employees—1930.
66 Pressure Vessels (Part I) Air Receivers and Hot Water Tanks—1937.
67 Maintaining Interest in Safety—1938.
68 Pressure Vessels—Fired and Unfired (Part II) Steam Jacketed Vessels, Digesters, Stills, Blow Cases and Autoclaves—1939.
69 Replaced by Pamphlet D-5.
70 Safe Practices for Maintenance and Repair Men—1937.
71 Safe Handling of Chlorine—1936.
72 Safety Committees—1939.
73 Foundries—1938.
74 Safety Contests—1937.
75 Safety Inspections—1934.
76 Portable Electric Hand Tools—1930.
77 Safety Meetings—1937.
78 Mathematical Tables and Data for the Safety Engineer—1939.
79 Accidents and the Engineer—1938.
80 Safety Rules—Their Formulation and Enforcement—1930.
81 Warning Signs—Their Use and Maintenance—1930.
82 Replaced by Health Practices Pamphlet 8.
83 Training for First Aid and Rules for First Aid Contests—1939.
84 The Safety Man in Industry—1939.
85 Safe Practices in Forging and Hot Metal Stamping—1934.
86 Combined with Safe Practices Pamphlet 21.
87 Safety in the Medium Sized Plant—1930.
88 Identification of Piping Systems—1937.
89 See Safe Practices Pamphlet CP-1.

90 See Safe Practices Pamphlet PT-1.
91 Spray Coating—1931.
92 See Safe Practices Pamphlet F-4.
93 Topics for Safety Meetings—1940.
94 State Safety Requirements in Industry—1935.
95 Compressed Gases—1939.
96 Industrial Power Departments—1931.
97 Safe Operation of Pulverized Coal Systems—1940.
98 Use and Care of Hoisting Chains—1939.
99 Falls of workers—Their Causes and Prevention—1937.
100 Safety Stunts—Part I—1935.
101 Safety Stunts—Part II—1937.
102 Off the Job Accidents—1939.
103 Purchasing for Safety—1939.

Safe Practices—Special Industries

Au-1 Safe Practices in Heat Treating—1931.
Au-2 Safe Practices in Motor Block Testing—1937.
CE-1 Accident and Health Hazards in Mercantile Establishments—1931.
CE-2 Safety in Hotels—1939.
CE-3 Safety in Hospitals—1939.
Cem-1 Cement Rock Quarrying and Crushing—1931.
Cem-2 Raw and Finished Cement Mill Grinding—1939.
Cem-3 Cement Burning (Including Fuel Handling)—1936.
Cem-4 Safe Practices in Cement Mill Shops—1930.
Cem-5 Storing, Packing and Shipping Cement—1940.
Cem-6 Cement Mill Yards and Railroads—1940.
Chem-1 Pipe Lines and Tanks as Causes of Accidents—1929.
Chem-2 Fume Poisoning from Nitric and Mixed Acids—1929.
Chem-3 Chemical Burns (Their Nature and Treatment)—1937.
Chem-4 Safety in Rayon Manufacture—1929.
Chem-5 Pyroxylin Lacquer Manufacture—1935.
Con-1 Preventing Accidents in Building Construction—1936.
Con-2 Safety and Health in Tunnel and Caisson Work—1931.
Con-3 Safety in Excavation Work—1931.
CP-1 Brick Making—1937.
D-1 Non-traffic Hazards of the Commercial Driver—1930.
D-2 Selecting Drivers for Commercial Vehicles—1935.
D-3 Training Drivers for Commercial Vehicles—1933.
D-4 Commercial Vehicle Accident Records—1939.
D-5 Preventing Vehicle Accidents—1937.
ER-1 Safe Maintenance of Electric Railway Rolling Stock—1931.
F-1 Safety in Food Preserving and Canning—1929.
F-2 Safety in Candy, Chocolate and Cocoa Manufacture—1930.
F-3 Safety in Bakery Operations—1930.
F-4 Safety in Milk Bottling Plants—1939.
F-5 Safety in Macaroni Plants—1936.
L-1 Safety in Dry Cleaning and Dyeing Establishments—1939.
M-1 Discontinued.
M-2 Mine Rescue Work—1937.
Mar-1 Fumigating Ships—1937.
Mar-2 Marine Boilers—1938.
Mar-3 Safe Practices in Handling Bunker Coal—1931.
Me-1 Safe Practices in Cleaning and Finishing Rooms in Foundries—1936.
Me-2 Blast Furnaces—1938.
Me-3 Safety in Structural and Sheet Metal Fabrication—1934.
Me-4 Rod Mills—1930.
PB-1 Safety and Health in Job and Newspaper Printing Establishments—1936.
PP-1 Paper and Pulp Mills—1918.
PP-2 Paper Box Manufacturing—1936.
PT-1 Leather Tanneries—1937.
Pet-1 Discontinued.
Pet-2 Discontinued.
Pet-3 Discontinued.
Pet-4 Discontinued.
Pet-5 Discontinued.
Pet-6 Safe Practices in Cleaning Petroleum Stills—1930.
Pet-7 Safety for the Oil Field Pumper—1929.
Pet-8 Drums and Barrels (Handling, Cleaning and Filling)—1936.
PS-1 Discontinued.

PU-1 Protecting Public Utility Employees on Streets and Highways—1938.
PU-2 Discontinued.
PU-3 Linemen's Rubber Protective Equipment—1939.
PU-4 Safe Handling of Poles—1939.
Q-1 Discontinued.
Ref-1 Safe Practices in the Delivery of Ice—1937.
RR-1 Use of Motor, Hand and Push Track Cars—1933.
Ru-1 Compounding Materials Used in the Rubber Industry Pt. I—1937.
Ru-2 The Safe Operation of Vulcanizers and Devulcanizers—1936.
Ru-3 Compounding Materials Used in the Rubber Industry Pt. II—1938.
T-1 Cotton Mills—1938.
W-1 Safety in Wood Furniture Manufacture—1931.
W-2 Safe Practices in Operating Commercial Lumber-yards—1932.

Health Practices

1 Chromium—1929.
2 Physical Examinations in Industry—1936.
3 Lead—1929.
4 Industrial Dust—1939.
5 Health Service in Industry—1939.
6 Industrial Eye Hazards—1934.
7 Carbon Monoxide—1929.
8 Caring for Injured Workers—1938.
9 Gases and Vapors—1929.
10 Skin Affections—1934.
11 Nursing Service in Industry—1930.
12 Replaced by Health Practices Pamphlet 5.
13 Physical Defects—1931.
14 Benzol—1931.
15 Replaced by Health Practices Pamphlet 5.
16 Physical Therapy in Industry—1935.
17 Back Injuries—1936.
18 Lighting and Health—1936.
19 Illness in Industry—1936.
20 Silicosis—1937.

PUBLICATIONS OF THE NATIONAL FIRE PROTECTION ASSOCIATION ON FIRES, FIRE PREVENTION, AND FIRE PROTECTION

(*Corrected to Jan.* 2, 1937)

Copies of the publications listed below will be mailed on application to National Fire Protection Association, 60 Batterymarch St., Boston, Mass.

A. *Standard Regulations for Fire Protection and the Safeguarding of Hazards*

Published by the National Board of Fire Underwriters

NOTE: The following N.F.P.A. Regulations have been adopted by, and are the official standards of the National Board of Fire Underwriters.

1. **Acetylene Equipment** for Lighting, Heating and Cooking. (1930.)
3. **Carbon Dioxide Fire Extinguishing Systems and Inert Gas for Fire and Explosion Prevention.** (1933.)
4. **City Gas**, Installation, Maintenance and Use of Piping and Fittings. (1932.)
5. **Combustible Fibres**, Storage and Handling. (1937.)
6. **Compressed Gas Systems**, other than Acetylene for Lightning and Heating. (1937.)

6*a*. **Dip Tanks** Containing Flammable Liquids, Including Hardening and Tempering Tanks; Flow Coat Work. (1936.)

7. **Dry Cleaning and Dry Dyeing Plants**, Safeguarding of. (1936.)
9. **Electric Cars and Trolley Buses, Including Houses and Yards.** (1935.)
10. **Electric Wiring and Apparatus** (National Electrical Code). (1937.)
12. **Fire Brigades, Private.** (1930.)
14. **Fire Pumps, Centrifugal.** (1933.)
15. **First Aid Fire Appliances** (fire extinguishers), Installation, Maintenance and Use of. (1931) with amendments (1932.)
16. **Foam Extinguisher Systems.** (1936.)
17. **Garages**, Construction and Protection of. (1932.)

18. **Gasoline Vapor Gas Machines, Lamps, and Systems.** (1926.)
19. **Gas Shut-off Valves, Emergency.** (1924.)
20. **Gas Systems for Welding and Cutting.** (1936.)
21. **Hose Houses for Mill Yards,** Construction and Equipment. (1936.)
22. **Internal Combustion Engines** (gas, gasoline, kerosene, fuel oil) and **Coal Gas Producers** (pressure and suction systems). (1934.)
23. **Liquefied Petroleum Gases,** Storage and Handling. (1937.)
24. **Marine Oil Terminals.** (1930.)
25. **Municipal Fire Alarm Systems.** (1934.)
26. **Nitrocellulose Motion Picture Films** (storage and handling). (1931)—with amendments (1936).
27. **Oil Burning Equipments,** and the Storage and Use of Oil Fuels in Connection Therewith. (1934)—with amendments (1936).
28. **Outside Protection,** Underground Piping Systems Supplying Water for Fire Extinguishment. (1931.)
29. **Ovens for Japan, Enamel, and Other Flammable Finishes.** (1931.)
30. **Paint Spraying and Spray Booths.** (1935.)
31. **Photographic and X-ray Nitrocellulose Film.** Storage and Handling. (1930.)
32. **Piers and Wharves,** Construction and Protection. (1935.)
33. **Proprietary, Auxiliary and Local Systems** for Watchman, Fire Alarm and Supervisory Service. (1931.)
34. **Protection of Openings in Walls and Partitions.** (1935.)
35. **Protective Signaling Systems,** Central Station, for Watchman, Fire Alarm and Supervisory Service. (1931).
36. **Pulverized Fuel Systems,** Installation of. (1935.) See also A-201.
37. **Pyroxylin Plastic,** Storage, Handling, and Use of in Factories Making Articles Therefrom. (1935)—with amendments (1936).
38. **Pyroxylin Plastic,** Storage and Sale of in Other Than Plants Manufacturing Articles Therefrom. (1922.)
39. **Sprinkler Equipments,** automatic and open systems. Standard Class B Sprinkler Equipments. (1936.)
40. **Standard Threads for Small Hose Couplings.** (1922.)
41. **Standpipe and Hose Systems.** (1931.)
42. **Sugar and Cocoa,** Pulverizing Systems for. (1930.) See also A-201.
43. **Tanks.** (Water, Gravity and Pressure, Towers, etc.) (1931.)
44. **Valves,** Controlling Water Supplies for Fire Protection. (1931.)
45. **Ventilating, Air Conditioning, Dust, Stock and Vapor Removal Systems, Power Operated.** (1937.)

Regulations and Special Committee Reports
Published by the National Fire Protection Association

100. **Anaesthetic Gases and Oxygen,** Recommended Good Practice Requirements for the Construction and Installation of Piping Systems for the Distribution of in Hospitals and Similar Institutions, and for the Construction and Operation of Oxygen Chambers. (1934.)
102. **Aviation,** Fire and Life Safety in. (1931.)
102*b*. **Building Construction Operations,** Recommended Good Practice Requirements. (1935.)
103. **Building Exits Code.** Stairs and enclosures, fire escapes, ramps, horizontal exits, doors, aisles, and corridors, elevators, escalators, slide escapes, alarm systems, fire exit drills, signs and lighting. Requirements for schools, department stores, factories, theatres and places of public assembly, hospitals, sanitariums and corrective institutions. (1936.)
104. **Coal Pneumatic Cleaning Plants.** (1930.) See also A-201.
107. **Fire Exit Drills and Alarm Systems.** (Reprint from Building Exits Code.) (1935.)
108. **Fire Hazard Properties of Certain Flammable Liquids, Gases and Volatile Solids.** (1935.)
108*a*. **Fire Hose, Care of.** (1936.)
109. **Fire Fighting Equipment in Metal Mines.** (1930.)
109*a*. **Forest Fire Fighting Equipment, Community.** (1934.)
109*b*. **Gasoline Blow Torches and Plumbers' Furnaces,** Recommended Good Practice for the Construction, Maintenance and Use of. (1934.)
110. **Lightning,** Protection of Life and Property against. (1937.)
110*a*. **Liquefied Petroleum Gases.** Automobile Tank Trucks and Tank Trailers for Transportation of. (1935.)
111. **Marine Fire Hazards.** (1930.) Appendixes also reprinted separately: A—Freeing Oil Tanks of Explosive or Toxic Gases. C—Stowage of Hazardous Commodities. D—Internal Combustion Engines.

112. **Mechanical Refrigeration,** Hazards and Safeguards of. (1931.)

112*a*. **National Standard Fire Hose Couplings and Fire Department Hose Connections for Sprinkler Standpipe Systems.** (1934.)

113. **Private Residences.** Specifications for. (1925.)

114. **Protection of Records.** Consolidated reports of the Committee on Protection of Records, 1923 to 1936, Including Regulations on **Vaults.**

115. **Rural Fire Departments,** Equipment and Organization. (1934.)

116. **Shoe Factories,** Suggestions for Their Improvement as Fire Risks. (1923.)

116*a*. **Spark Arresters.** Standard for Construction and Installation of, for Chimneys and Stacks. (1936.)

116*b*. **Spontaneous Ignition of Coal and Other Mining Products.** (1936.)

117. **Spray Application of Flammable Finishing Materials.*** (1928.)

117*a*. **Standard Fire Test Specifications.** (1934.)

118. **Standard Industrial Buildings,** Specifications for. (1929.)

119. **Structural Defects.** Suggestions for their Elimination and Protection.

119*a*. **Summer Homes in Forested Areas.** Fire Protection and Fire Prevention for. (1934.)

120. **A Table of Common Hazardous Chemicals.** (1929.)

120*a*. **Terminal Grain Elevators,** Prevention of Dust Explosions in. (1931.)

121. **The Watchman.** Suggestions for Guidance in Selection, Instruction and Duties. (1925.)

122. **Water Charges.** Report of Committee on Public Water Supplies for Private Fire Protection. (1932.)

123. **Water Systems for Fire Protection on Farms.** (1935.)

N.F.P.A Standards Published by U. S. Government

These publications may be obtained from the Government Printing Office, Washington, D. C. Remittances should be included with orders, in cash, postal money order, or coupons sold by the Superintendent of Documents, Government Printing Office. As a convenience to members, single copies will be furnished by the Executive Office when ordered with other publications.

201. **Dust Explosions,** *Safety* Codes for the Prevention of. ***Dept. Labor, Bull.* 562.** *Suppl. Bull.* 617.
    Starch Factories. (1934.)
    Flour and Feed Mills. (1933.)
    Terminal Grain Elevators. (1933.)
    Pulverizing Systems for Sugar and Cocoa. (1931.)
    Spice Grinding Plants. (1931.)
    Wood-flour Manufacturing Establishments. (1931.)
    Pulverized Fuel Systems. (1933.)
    Coal Pneumatic Cleaning Plants. (1930.)
    Inert Gas for Fire and Explosion Prevention, Use of. (1931.)
    Woodworking Plants. (1934.)

202. **Gasoline and Kerosene on the Farm,** Safe use and storage of. *Dept. Agr., Bull.* 1678.

203. **Fire Protective Construction on the Farm.** *Dept. Agr., Bull.*

B. Suggested Municipal Ordinances

1. **Automobile Tank Trucks and Tank Trailers.** Regulating the Construction and Operation of.

1*a*. **Bureau of Fire Prevention,** To establish, provide officers and define their powers and duties.

2. **Chimneys, Flues and Fireplaces,** To provide for the safe construction of.

3. **Explosives,** Suggested Ordinances.

3*a*. **Fireworks Ordinance,** Suggested. (Contained in D-21*a*.)

4. **Flammable Liquids** and the Products. Thereof, To regulate the use, handling, storage and sale of. Including Appendixes on Gasoline Service Stations, and Rooms, Cabinets and Outside Houses for Flammable Liquids.

4*a*. **Fumigation Ordinance, Model.**

5. **Oil Burning Equipments,** and Oil Storage in Connection Therewith; **Stove or Range Oil Burners.**

5*a*. **Oil Burning Equipments** (Short form).

8. **Wooden Shingle Roofs,** Prohibition of.

9. **Fire Prevention Code for Cities.** Advisory Outline.

C. Reference Books

1. **Crosby-Fiske-Forster Handbook of Fire Protection.** A standard reference volume for beginners and experienced men specializing in fire prevention and protection. It is also an invaluable reference work for those occasionally requiring

---

* Reprint from Quarterly.

specific facts of this character. Eighth edition (1936) 1154 pp., 4¼ by 6½ in., 242 illustrations, 161 tables, handsomely bound.

1*a*. **Field Practice. Inspection Manual** designed for the use of property owners, fire departments and inspection offices in safeguarding life and property against fire, 206 pp., 4½ by 6¾ in., bound in flexible "Fabrikoid."

2. **Dust Explosions,** Theory and Nature of Phenomena, Causes and Methods of Prevention. An authoritative and comprehensive work by Price and Brown. 246 pp., 6 by 9 in., large readable type. Forty-eight illustrations, 39 drawings, 25 tables.

3. **Industrial Fire Hazards,** by Dana and Milne. The first section describes the manufacturing processes in the principal industries, dealing with special hazards, construction and protection, giving fire record data and suggestions for safeguarding hazards. The second section, entitled Encyclopedia of Hazardous Materials, describes most of the chemical material, oils, etc., found in the field by inspectors, and notes the hazards. 950 pp., illustrated.

4. **The Boston Fire Waste Survey.** A comprehensive report on the fire waste problem of Boston prepared by the field engineering staff of the N.F.P.A. 310 pp., illustrated.

Note: A paper, "Fire Protection Engineering as Applied to Municipalities," by Akers, describing the Survey, is available in pamphlet form.

5. **Hydraulics of Fire Streams and the Nozzle as an Accurate Water Meter.** Two papers recording original experiments by Freeman, reprinted from Transactions of American Society of Civil Engineers. 216 pp., illustrated.

6. **Fire Tests of Building Columns.** An Experimental Investigation of the Resistance of Columns, Loaded and Exposed to Fire or to Fire and Water with Record of Characteristic Effects. Published by Underwriters' Laboratories, 300 pp.

7. **Prevention and Control of Farm Fires.** A popular handbook on the causes of farm fires and best methods of safeguarding against loss of life and property. An attractively bound 167-page volume.

8. **The Volunteer Fire Company,** by Brinckloe. An attractive 168-page volume with 24 illustrations of volunteer fire department apparatus, equipment and fire fighting methods.

D. Popular Educational and Miscellaneous Pamphlets

1. **Advanced Courses in Firemen's Training.*** Bond.
1*a*. **Automatic Sprinkler Systems by Fire Departments,** Use of.†
3. **Belt Dressings for Eliminating Static Electricity.*** Edwards and Reed.
4. **Carbon Dioxide Fire Protection.*** Moulton.
6. **Certificate of Occupancy.†** Miller.
6*a*. **Chlorine in Industry.*** Stone.
6*b*. **Cigarettes, Fire Hazard Test with.*** Hoffheims.
8. **City Planning and Fire Protection.*** Klein.
8*a*. **City Planning and Zoning in Relation to Fire Prevention and Fire Protection.†**
9. **Decreasing the Fire Hazard.** Wentworth.
12. **Dwelling House Inspections.*** Tierney.
12*a*. **Dwelling Inspection by Fire Departments.***
13. **Explosions, Dust and Smoke.*** Steward.
14. **Fire Alarm Central Stations.*** Revised and reprinted. Carroll.
15. **Fire Department Drill Towers.***
16. **Fire Department Pumpers,** Their Effective Use and Co-ordination with Private Protection Facilities.† Daniel.
16*a*. **Fire Department Records.***
17. **Fire Effects on Fire-Resistive Construction.*** Burton.
17*a*. **Fire Gases.*** Ferguson.
18. **Fire Prevention Exhibits.** Illustrations of fire prevention exhibits and window displays.
19. **Fire Prevention Week Handbook.** Suggestions for guidance in planning the observance of Fire Prevention Week. 48 pages, illustrated.
20. **Fireproofing of Wood.†** Truax.
21. **Fire Test of Brick Joisted Buildings.***
21*a*. **Fireworks Casualties and Regulation.***
22. **Fresno, The Story of.** A Tale of the Redemption of a Fire Department.*

* Reprint from Quarterly.
† Reprint from Proceedings.

23. **Gasoline Transport Truck Fires.***
24. **High Tension Wires and Hose Streams.*** Walker.
25. **Indianapolis, The Story of.** A successful fire prevention campaign.*
26. **Individual Liability Laws** for Fires due to Carelessness or Neglect.
27. **Inspected Electrical Appliances.** Published by Underwriters' Laboratories, Inc. (Revised semiannually.)
28. **Inspected Fire Protection Appliances.** Published by Underwriters' Laboratories, Inc. List includes appliances and materials which have been investigated as to the fire hazard. (Revised semiannually.)
29. **Inspected Gas, Oil and Miscellaneous Appliances.** Published by Underwriters' Laboratories, Inc. List includes devices, materials and systems which have been investigated as to the fire hazard. (Revised semiannually.)

29*a*. **Jute, Heat and Ignition Tests with.**

30. **Loss of Life by Fires in Buildings Equipped with Automatic Sprinklers.***
31. **Menace of the Wooden Shingle, The.**
32. **Metal Working Processes.*** Milne. Including Fire Record of Metal Working Industries.
33. **Modern Fire Fighting in Small Cities.*** McAuliffe.

33*a*. **Municipal Fire Apparatus Budget Programs.*** Bugbee.

34. **Municipal Regulation of Oil Storage, The.*** Moulton.
35. **Mysterious Barn Fires.*** Price.
36. **Oil Tank Fire Tests.*** Joint report of engineers of five oil companies.
37. **Organization Plan for a Fire Prevention Committee.**

37*a*. **Private Fire Brigade Results.***

40. **Railroad Fire Hazards.*** Rapalje.
41. **Record Protection and Office Equipment.*** Ingberg.
43. **Safeguarding the Home against Fire.** Fire Prevention Manual for School Children, Published by National Board of Fire Underwriters.
44. **Salvage.** McAuliffe. Chairman, N.F.P.A. Committee on Salvaging Operations. Published by National Board of Fire Underwriters.

44*a*. **Selected Demonstrations for Use in Fire Safety Education.***

45. **Smoke and Water Damage.*** Roberts.

45*a*. **Spontaneous Ignition of Chopped Hay.*** Roethe, Bradshaw, and Hoffman.

46*a*. **Sprinkler Users Win Rate Case.***

46*b*. **Sprinklers in Boston Apartment Houses.*** Wentworth.

46*c*. **Static Electricity as a Fire Cause.*** Turkington.

47. **"Story" of the National Fire Protection Association.**
48. **Syllabus for Public Instruction in Fire Prevention.**
49. **Tests of the Severity of Building Fires.*** Ingberg.

49*a*. **Uninsulated Steel Vault Doors.*** Porter.

50. **Thermostats.†** Hendricks.

50*a*. **Venting Dust Explosions.***

50*b*. **Water Supplies for Rural Fire Protection.*** Charnock.

50*c*. **Use of Water to Extinguish Oil Fires.*** Brown.

50*d*. **Water Damage Following Sprinklered Fires.***

51. **Wired Glass Windows in the Fall River Conflagration.** 14 pp., illustrated.

Special Bulletins

FIRE PROTECTION: Assheton, "History of Explosions," Institute of Makers of Explosives, New York. "Fire Tests of Building Columns," Associated Factory Mutual Fire Insurance Companies, Boston. Crosby, Fiske, and Forster, "Hand Book of Fire Protection," Van Nostrand, New York. Dana, "Automatic Sprinkler Protection," Wiley, New York. Dominge and Lincoln, "Fire Insurance Inspection and Underwriting," Spectator Company, New York. Eichel, "Fire Prevention and Protection for Hospitals," Wiley, New York. Gamble, "Outbreaks of Fire, Their Causes and Means of Prevention," Lippincott, Philadelphia. Hough and Lawson, "Fire Prevention Year Book," Hough-Lawson, Baltimore. Hutson, "Fire Prevention and Protection," Spectator Company, New York. Owen, "Notes on Hydraulics," Insurance Press, New York. Phillips, "The Handling of Dangerous Goods," Crosby, Lockwood, London. Shepperd, "Practical Hydraulics for Firemen," Sheppard, New York. Von Schwartz, "Fire and Explosion Risks," Griffin, London. Wallace, "Fire Losses Locomotive Sparks," Barr-Erhardt Press, New York.

GENERAL SAFETY: Chase, "Men and Machines," Macmillan, New York. DeBlois, "Industrial Safety Organization," McGraw-Hill, New York. Fisher, "Mental Causes of Accidents," Houghton Mifflin, Boston. Heinrich, "Industrial Accident Preven-

* Reprint from Quarterly.

† Reprint from Proceedings.

tion," McGraw-Hill, New York. Lange, "Handbook of Safety and Accident Prevention," McGraw-Hill, New York. Williams, "The Manual of Industrial Safety," McGraw-Hill, New York.

INDUSTRIAL HEALTH HAZARDS AND MEDICAL SERVICE: Hamilton, "Industrial Poisons in the United States," Macmillan, New York. Kober, "Industrial Health," Blakiston, Philadelphia. Rambousek, "Industrial Poisoning From Fumes, Gases and Poisons of Manufacturing Processes," Legge, London. Thompson, "The Occupational Diseases—Their Causation, Symptoms, Treatment and Prevention," Appleton, New York. Underhill, "Toxicology," Philadelphia.

PUBLICATIONS OF VARIOUS BUREAUS: Various organizations and governmental departments have issued bulletins from time to time relating to safety and fire protection. Among these are the bulletins, technical papers, and circulars of the U. S. Department of Labor, U. S. Bureau of Mines, U. S. Public Health Service, U. S. Bureau of Standards, Federal Board for Vocational Education, Interstate Commerce Commission, and others.

PERIODICALS: *Fire Engineering* (Semimonthly), Case, Sheppard, Mann Publishing Corp., New York. *Fire Protection* (Monthly), National Underwriter Co., Indianapolis. *National Safety News* (Monthly), National Safety Council, Chicago. *Safety Engineering* (Monthly), The Safety Magazine Publishing Corp., New York. *Quarterly*, National Fire Protection Association, Boston.

INDEX

SECTION 29

REPORTS AND REPORT WRITING

BY

Fred C. Zeisberg, deceased; formerly Technical Investigator, Development Department, E. I. du Pont de Nemours & Co.

CONTENTS

29

REPORTS AND REPORT WRITING

BY FRED C. ZEISBERG

REFERENCES: Allbutt, "Notes on the Composition of Scientific Papers," Macmillan, London, 1923. Baker, "The Preparation of Reports," Ronald Press, New York, 1924. Earle, "The Theory and Practice of Technical Writing," Macmillan, New York, 1911. Harbarger, "English for Engineers," McGraw-Hill, New York, 1923. Hubbell, The Chemist and His Message, *Chem. Ind.*, **47**, 1110–1113 (1928). Norton, The Art of Writing Scientific Reports, *Sci. Monthly*, **11** (6), 648–654 (1920). Rickard, "Technical Writing," Wiley, New York, 1923. Sypherd and Brown, "The Engineer's Manual of English," Scott, Foresman, Chicago and New York, 1933. S. M., On Scientific Writing, *Chem. Ind.*, **47**, 1114–1115 (1928). Trelease and Yule, "Preparation of Scientific and Technical Papers," Williams & Wilkins, Baltimore, 1925. T. W. J., The Titles of Chemical Papers, *Chem. Age* (*London*), **11**, 237 (1924).

There are two general classes of reports in which the chemical engineer is most likely to be interested: (1) form reports, used in the control of some plant operation or recording of a research experiment, and (2) general reports, varying from routine weekly reports on experimental work to veritable volumes covering a detailed analysis of a whole industry. Financial reports, of course, have their place in many chemical engineering studies, but discussion of these will be omitted from the following.

FORM REPORTS

Daily Reports. These are generally designed for the purpose of recording pertinent data in connection with some plant operation. A ruled form is made up, containing blank spaces in which the significant figures can be inserted, usually by the foreman in charge of the operation. The size must be chosen with respect to the number of entries required, but it is well to bear in mind that if these reports are to be filed there are certain standard-size cabinets or drawers, such as 4 by 6 in., 5 by 8 in., $8\frac{1}{2}$ by 11 in. and $8\frac{1}{2}$ by 13 in., in which the reports can be filed neatly and without wasted space if the form is chosen of one of the standard sizes instead of being made up to any odd size.

Once the form has been made up, it can be multigraphed or printed and is then ready for use. Such reports show the weights of materials fed to an operation and products delivered therefrom, together with figures showing temperatures, pressures, chemical tests, or any similar matter important in the control of the operation. Even when recording thermometers or pressure gages are part of the regular equipment of a process, the use of a daily report form requiring the periodic entry of significant figures by the operator in charge insures better control of the process.

For continuous processes the daily report form usually provides for a 24-hr. operation, and, since most chemical operations are on an 8-hr. shift basis, three separate foremen or operators fill in one form. It is hence customary to provide spaces designated shift 1, shift 2, and shift 3 for the signatures of the three men in question, so that responsibility for irregularities on any shift can be easily placed.

The daily reports are sent daily to the supervisor in charge of the operation and are kept on file in his office, for use in compiling the monthly report. They serve, also, to bring to his attention quickly any abnormal condition in the operation in his charge.

Only the most general directions can be given regarding the designing of a daily report form. Common sense must be used liberally. Spaces should be provided for all figures needed in the chemical accounting of the process, and only such additional figures should be required as are necessary to an intelligent control of the process. On forms used for continuous processes, the length of time a process is interrupted should be noted. A space, usually the reverse of the form, should be reserved for comments by the operator, and he should be encouraged to make them. It is well, from time to time, to look over the form critically to see whether figures once regarded as important cannot be omitted and to make sure that no other figures which have become important fail in being reported because no spaces are provided for them.

Monthly Reports. Most chemical manufacturing companies require monthly reports on the various chemical operations conducted by them. These are designed to give information not contained in the usual cost sheets and permit a comparison of operating efficiency month by month, or among plants. The daily report is the source of information used in compiling the monthly report.

The same points important in the compilation of daily reports apply equally well to monthly reports, except that monthly reports are usually permanent records and should hence be on a better grade of paper than daily reports, which may be destroyed after the monthly reports are compiled.

Research Form Reports. While research data are preferably, and for the most part, recorded in blank books, it is often more convenient to use loose-leaf forms. This is particularly the case in continuous experiments on the semiworks scale, in which the responsibility for recording the data is passed on from one shift to another until the run is completed. Since the correlation of the data is usually in the hands of someone other than those who record them, it is obvious that a blank book, which might be needed for the recording of a succeeding run, would not be so handy as a pad of loose-leaf report forms.

The forms, of course, will have to be designed for the particular experiment. Probably the most convenient size is 8½ by 11 in., as this can be filed in standard letter files or placed in ring binders for future reference. For runs lasting several days, and particularly in those cases in which data on one form must be compared with those on another or with the record from some continuous recording instrument, it is very convenient to use 24-hr. time, *i.e.*, to call midnight 24:00 o'clock. Thus errors arising from comparing data for, say, 3:00 A.M. on one sheet with those for 3:00 P.M. on another sheet, are avoided.

GENERAL REPORTS

In this category fall such papers as research reports, descriptions of plants, industry surveys, analyses of newly proposed or going processes, and many other similar writings, varying in size and scope from one-page memoranda to monographs extending into hundreds of pages. At least two cardinal points must be borne in mind in the preparation of all of these: absolute clarity, and the training and experience of the reader for whom the report is meant. Even when discussing the various subheadings below, it is obvious that only generalities can be touched upon, each particular type of report being subject to great flexibility in method of presentation, depending upon the individuality of the author, the points he wishes to emphasize, and the limitations of the readers whom he wishes to reach.

Research Reports. These are nearly always prepared for a group of readers of the same general experience and training as the author. There is hence no restriction on the fullest use of technical language, except that terms very special to the subject under discussion should be defined early in the report, if this report is meant for technical men who have not specialized in the subject as fully as the author. In the following discussion, it should be remembered that commercial research laboratories, in which the researchers are part of an organization and not individualists, have been kept in mind.

Various systems of reporting the results of research work are in vogue. Several systems used in research laboratories connected with the chemical industry will be briefly described and commented upon.

In one it is the practice for each man to prepare a brief, separate, weekly statement of progress on each subject. This has the advantage of keeping his superiors up-to-date, but the disadvantage that a considerable stenographic and filing effort is required. Moreover, the information being in homeopathic doses, it is doubtful whether the superior for whom the report is meant, especially when he is handling a number of studies, can properly integrate the disjointed weekly reports into a well-rounded picture of the status of the work on each study. The same laboratory, in which this weekly report system is the practice, requires monthly reports on each subject, regardless of whether the research has reached a logical point for a report or not. This monthly report is much fuller than the weekly report and attempts to review and coordinate the information developed since the previous monthly report was issued.

In another research laboratory no weekly reports are required, but monthly sections are written on each subject, the whole being combined in a single monthly report. No final reports are required, a series of sections in the monthly report serving as the history of the research.

The advantages of requiring reports at stated intervals are various. There unfortunately exist a number of research workers who approach report writing as a distinct task, and who consequently postpone writing up their results until, in the enthusiasm of pursuing some new line of research, they neglect entirely to write up properly their past work. For this class the monthly report, though a bore, is a necessity. Further, in writing up a research in monthly installments, the opportunity is provided for suggestions from readers of the reports to be tried before the research is terminated. Finally, monthly reports establish dates, at least within definite time limits, which may become exceedingly valuable in future establishment of priority in patent litigation.

The disadvantage of the monthly report, at least when a final report reviewing the whole research is not required or not prepared, is that the researcher's viewpoint may change considerably as the work proceeds, and statements of fact or theory made early in the course of the work and subsequently found in error may remain in the record uncorrected. Also, with the best intentions in the world on the part of the author, such a series of reports is bound to present a disjointed and unfinished appearance, which would be absent in a full report prepared on the completion of the work.

Some laboratories follow the practice of reporting research work only on the completion of the study on a particular subject. If the study is not too long, this practice results in the best type of report, but it is not without its disadvantages. If the study is a long one, especially if the researcher does not make copious daily notes, points of considerable importance may not be reported. Even on short studies there are men whose memories will not marshal all the desired facts when the report is being written. For these, periodic reports provide the best medium.

There probably is no hard-and-fast rule which is applicable to reporting the results of all kinds of research at all laboratories. The length of the research must determine, to a large extent, the manner in which it is reported. The peculiarities of the researcher must also be considered. The ideal way to produce the best and most readable report is to prepare this report at the conclusion of the research from copious dated notes taken during the progress of the work. Unfortunately, this happy end can hardly ever be attained in the commercial research laboratory. For the sake of keeping those at the head of the laboratory reasonably promptly informed as the work progresses, and to assure some literary activity on the part of those men who dislike writing, the monthly report is perhaps the best system. To make it complete, however, there should be a final report in which are incorporated all of the findings of the preceding monthly reports properly coordinated.

1. Form of Research Report. Instead of leaving the form of the report to the caprice of the author, experience has shown that it is well to prescribe it. There are, of course, several possible forms, but one which has been adopted as standard in a good many laboratories is described below.

First comes the **introduction,** in which is stated the reason for making the study and the end it is hoped to attain. Next comes a section devoted to the **state of the art,** which may also be called the **historical section.** Then comes the body of the report in which the research is described and, finally, comes the **summary** and/or **conclusions.** At some laboratories it is the practice to put the summary immediately following the introduction, so that a busy executive need not leaf over the whole body of the report, in which he has no particular interest, to find out what the results of the research were.

It is often the practice to attach an **appendix** in which are shown illustrations, tables, or references to the literature. It is a nuisance, however, to have to turn over to an appendix to refer to something which must be consulted to obtain a clear understanding of the text. For this reason small tables are best incorporated in the text, and for the same reason references to the literature or to patents, if they are not too numerous, are best shown in parentheses in the text itself. If there are many such references it is better to give reference numerals in the text and to show the reference itself in footnotes or in a bibliography in the appendix. In giving literature references, it is well to adopt the abbreviations standardized by the American Chemical Society and used in *Chemical Abstracts*.

Since illustrations, unless they are small, cannot be conveniently shown in the body of typewritten text, and then only if they are separately made and pasted into a space left for them, it is common practice to show these on separate sheets. Such separate sheets, particularly if they carry curves or diagrams which must be repeatedly consulted to get the sense of the text, are best made with throw-outs, *i.e.*, on folded sheets which can be opened to expose the figure beyond the edge of the report. The eye can then easily pass back and forth between the text and the figure.

Research reports have commonly been bound at the top, but it is much more convenient to bind them along the left margin so that they open like a book. Bound thus, they are easier to read and they file more readily, either in loose-leaf ring binders, or in file cabinets. The above is true for the standard report, on 8½- by 11-in. paper. In some companies, reports for the higher executives, and all reports for use in legal cases, must be written on 8½- by 13-in., or 8½- by 14-in., paper, which is more conveniently bound at the top.

2. What to Include in a Research Report. The sole product of a piece of research, except for the experience gained by the researcher, which may be lost owing to his death, or to his leaving the employ of the organization for which he is working, is the reports produced in its prosecution. It is consequently of the utmost importance to make these reports complete. It is much better to err on the side of including too much rather than too little.

There is a temptation to leave out details, or little side experiments, which at the moment appear to have no bearing on the main subject under consideration. Research reports are permanent records, however, and something apparently of no importance at the time may assume considerable significance in the light of later knowledge.

If it is not desired to clutter up the body of the report with apparently irrelevant matter of this kind, the appendix always provides a convenient place to record it.

On the basis of the facts recorded the deduction of a theory to account for what has happened may not be justified. However, there are individuals gifted with an intuition which can deduce theories on very slight evidence. There is no objection to the inclusion of such theories or explanations if it is made clear that they do not purport to be more than unsupported hypotheses.

It is also important to give credit for suggestions used in the prosecution of research work. Modern industrial research is nearly always the product of team work, and, as far as possible, each team-member's contribution should be noted in the reports. This is important not only to maintain esprit de corps but may be of great value in determining the inventor, in case of patent application.

Descriptions of Plants. In this type of report the form is much simpler. There may or may not be an **introduction,** and a **summary** may or may not be necessary. The body of the report in this case is the important thing and, for convenience, may be divided into a number of headed paragraphs, especially when the description goes into considerable detail. Some such headings might be **location, raw materials, labor supply, water supply, shipping facilities,** etc. If at all possible, a flow sheet to show the flow of materials through the plant should be included, because it makes it unnecessary to describe the process so fully and also makes the process much clearer to the reader. If drawings or photographs are available they should be included; if not, there should be included at least a diagrammatic sketch of the plant.

In describing a process it is well to follow some material, from the point where it enters, in its course through the plant. If there are several materials of equal importance, they should be followed through one after another.

After the process has been described in general, the description can go on to the individual steps in detail, taking up the apparatus used in the step, the process occurring therein, and a discussion or criticism of the step. The author's opinions should be clearly differentiated from facts, however.

In this type of report the class of reader should be particularly carefully borne in mind. Many such reports are destined to be used by non-technical business men to assist in deciding whether the plant in question should be purchased. In that case it is necessary to be careful to make the description as non-technical as possible and to bring out clearly in the summary the important facts developed by the inspection covered in the report.

Industry Surveys. It is frequently necessary to report on an industry, in order to provide information upon which the erection of new plants is to

INDEX

be based. In that case the report must cover the consumption of the commodity it is purposed to manufacture, segregating this consumption into uses and into as small territories as is possible. The probable future consumption must then be estimated and reported upon. Raw-material supplies must be looked into, from the standpoint of location and abundance. Freight rates on raw materials and on the finished product must be examined, in order to determine in which locality, supplying the possible markets, minimum freight charges would be incurred. Such a report may be brief or detailed—the purpose for which it is to be used must determine this point.

To even a greater extent than is true for process reports, industry surveys are nearly always made to be used by non-technical men. The chemical engineer must hence be doubly on guard when handling technical subject matter to make this intelligible to the lay reader.

General Remarks. The preparation of a good report presupposes the ability to write. There is an enormous variation in ability in this respect, however, among otherwise able men. As mentioned before, the primary consideration is clarity. If a report is clear, errors of construction or grammar can be overlooked. But no technical man likes to think that his report, though correct as to facts, and free from ambiguity, is not couched in the best of English.

For those who have difficulty in expressing themselves, or who are not satisfied with their best effort, perhaps no advice will help more than that to read good authors. Newspapers, unfortunately, are not generally written in the best English, if some editorials are excepted. In general, philosophers write good English. Huxley and Spencer are shining examples. Darwin is good, though not so precise in his use of words as Spencer. Thackeray uses English marvelously. Dickens is inclined to be careless. Robert Louis Stevenson is famous for the care he took in his writing. Mark Twain's use of English is excellent.

One of the most important points in writing good English is the exact use of words. While much can be learned about the exact meanings of words by consulting a dictionary, the finer shades of meaning can be learned only through reading, to see how words are related to other words by the best writers. Much can be learned by consulting books prepared on the subject of report writing. These go much more fully into the subject than is possible here. For those who want to pursue the matter further, the references at the beginning of this section will be of service.

Regarding the more mechanical features of report writing, much can be learned from the style sheets issued by large publishers and by engineering societies, such, for example, as the American Society of Mechanical Engineers.

In closing, there is one piece of advice to bear in mind—write for the sake of the matter and not for the sake of the writing.

INDEX

International Atomic Weights, 1941

Substance	Sym.	At. Wt.	At. No.	Substance	Sym.	At. Wt.	At. No.
Aluminum	Al	26.97	13	Molybdenum	Mo	95.95	42
Antimony	Sb	121.76	51	Neodymium	Nd	144.27	60
Argon	A	39.944	18	Neon	Ne	20.183	10
Arsenic	As	74.91	33	Nickel	Ni	58.69	28
Barium	Ba	137.36	56	Nitrogen	N	14.008	7
Beryllium	Be	9.02	4	Osmium	Os	190.2	76
Bismuth	Bi	209.00	83	Oxygen	O	16.0000	8
Boron	B	10.82	5	Palladium	Pd	106.7	46
Bromine	Br	79.916	35	Phosphorus	P	30.98	15
Cadmium	Cd	112.41	48	Platinum	Pt	195.23	78
Calcium	Ca	40.08	20	Potassium	K	39.096	19
Carbon	C	12.010	6	Praseodymium	Pr	140.92	59
Cerium	Ce	140.13	58	Protactinium	Pa	231	91
Cesium	Cs	132.91	55	Radium	Ra	226.05	88
Chlorine	Cl	35.457	17	Radon	Rn	222	86
Chromium	Cr	52.01	24	Rhenium	Re	186.31	75
Cobalt	Co	58.94	27	Rhodium	Rh	102.91	45
Columbium	Cb	92.91	41	Rubidium	Rb	85.48	37
Copper	Cu	63.57	29	Ruthenium	Ru	101.7	44
Dysprosium	Dy	162.46	66	Samarium	Sm	150.43	62
Erbium	Er	167.2	68	Scandium	Sc	45.10	21
Europium	Eu	152.0	63	Selenium	Se	78.96	34
Fluorine	F	19.00	9	Silicon	Si	28.06	14
Gadolinium	Gd	156.9	64	Silver	Ag	107.880	47
Gallium	Ga	69.72	31	Sodium	Na	22.997	11
Germanium	Ge	72.60	32	Strontium	Sr	87.63	38
Gold	Au	197.2	79	Sulfur	S	32.06	16
Hafnium	Hf	178.6	72	Tantalum	Ta	180.88	73
Helium	He	4.003	2	Tellurium	Te	127.61	52
Holmium	Ho	164.94	67	Terbium	Tb	159.2	65
Hydrogen	H	1.0080	1	Thallium	Tl	204.39	81
Indium	In	114.76	49	Thorium	Th	232.12	90
Iodine	I	126.92	53	Thulium	Tm	169.4	69
Iridium	Ir	193.1	77	Tin	Sn	118.70	50
Iron	Fe	55.85	26	Titanium	Ti	47.90	22
Krypton	Kr	83.7	36	Tungsten	W	183.92	74
Lanthanum	La	138.92	57	Uranium	U	238.07	92
Lead	Pb	207.21	82	Vanadium	V	50.95	23
Lithium	Li	6.940	3	Xenon	Xe	131.3	54
Lutecium	Lu	174.99	71	Ytterbium	Yb	173.04	70
Magnesium	Mg	24.32	12	Yttrium	Y	88.92	39
Manganese	Mn	54.93	25	Zinc	Zn	65.38	30
Mercury	Hg	200.61	80	Zirconium	Zr	91.22	40

66.—

INDEX TO SECTIONS FOR QUICK REFERENCE

USE THE THUMB TABS