27.6—SCREENING AND CLASSIFICATION

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27.6.1—SCREENING

The processing of minerals almost invariably involves the reduction in size of the minerals contained in an ore to effect liberation of disseminated values or, in other instances, to effect increased surface area. Separation of solids according to size usually is undertaken to promote maximum production from crushing and grinding equipment. Screens are generally used for making this size separation when coarse material is involved, whereas classifiers and cyclones usually are employed when fine particulate material is being processed. However, special screening devices may be used for separations as fine as 325 mesh. The term "mesh" refers to the number of openings per linear inch on a screen. Mesh number vs. size of screen opening is listed in Table 27-12.

TABLE 27-12—Mesh Number and Screen Opening, U.S. Sieve Series and Tyler Equivalents

U.S. Series	Tyler Equivalent	Size, μ
1.060 in.	1.050 in.	—
0.745 in.	0.742 in.	—
0.530 in.	0.525 in.	—
0.375 in.	0.371 in.	—
0.265 in.	0.263 in.	—
No. 4	4 mesh	4,760
No. 6	6 mesh	3,360
No. 8	8 mesh	2,380
No. 12	10 mesh	1,680
No. 16	14 mesh	1,190
No. 20	20 mesh	841
No. 30	28 mesh	595
No. 40	35 mesh	420
No. 50	48 mesh	297
No. 70	65 mesh	210
No. 100	100 mesh	149
No. 140	150 mesh	105
No. 200	200 mesh	74
No. 270	270 mesh	53
No. 325	325 mesh	44
No. 400	400 mesh	37

Numerous screening devices are available, and they can be classified as stationary, mechanical, high-speed mechanical and electrically vibrated. The advent of modern explosive techniques, which produce finer crushing-plant feed, has caused a trend from stationary grizzles to vibrating multideck mechanical screens (Sec. 28.3.3).

The same type of mechanical screen also is applied to secondary and tertiary crushing stages. The decks are available as alloy castings, bars or rods, punched plate, woven-wire mesh, rubber or rubber-covered steel and even plastics. In some instances, decks are heated electrically to eliminate the buildup of fines on the screening surface. Many operations are improved with the application of rubber or plastic decks.

Deck motions can be straight-line, circular or eliptical in nature. High-speed mechanical screens are used widely for separations of about ¼ in. and finer. Electrically vibrated machines also are used to make separations of ⅛ in. and finer.

The amount of material that can be sized by screening per unit of time is termed the capacity of the screen and can be described in tons of throughput per hour per square foot of screening surface. Capacities of screens can be determined by utilizing techniques described by equipment manufacturers or by utilizing the chart presented in Fig. 27-11. Note that larger capacities are obtained with sand and gravel compared with crushed material of the same specific gravity. This fact results from the more spherical shape of particles and their free-flowing characteristics. Since the bulk density of coal and other solids varies, appropriate adjustments must be made for these variations. The "coal" line shown is for 60 lb per cu ft coal; coke corresponds to 30 lb per cu ft on the chart. Similarly, adjustments for the variation in bulk density of ores should be made using the "stone and ores" line, which corresponds to 100 lb per cu ft.

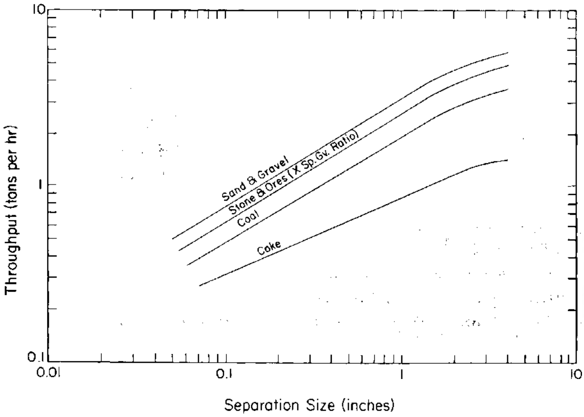

Fig. 27-11—Relationship between screen throughput and separation size for various solids.

Other adjustments for type and location of deck, wet or dry screening, and screen analysis of deck feed are necessary. With respect to use of the chart, some caution is advised, as the screening capacity also is dependent on required efficiency of separation, amenability to separation, speed, slope and amplitude. Further, this chart should not be used for separations finer than about ⅛ in. or coarser than about 3 in.

The capacity of a screen surface making a specific separation can be determined by multiplying the chart reading at this separation size by the factors noted below. That is, Through-Capacity (tph per sq ft) = chart reading $\times E \times D \times F \times W \times T \times B$, where E, D, F, W, T and B are factors adjusting for efficiency, number of decks, % fines, wet screening, type of deck factor and bulk density.

Efficiency Factor (*E*)—This factor accounts for the percent of material in the feed to each deck that actually passes through compared to the amount available to pass through. If a feed contains 50% minus ½ in. material, and it is desired to pass 95% of this material through the screen, then the efficiency factor is 1.0. When screens are used to scalp feed, removal of 80 to 85% of the fine material ahead of the crusher usually will ensure maximum capacity of the crusher. A factor of 1.5 to 1.75 is used under these conditions. Other values of this factor are presented in the following tabulation:

Efficiency Factor (E)	
95 %	1.00
90 %	1.25
85 %	1.50
80 %	1.75
75 %	2.00

Deck Factor (*D*)—This factor is used because the effective screening lengths of second or third decks are smaller than that of a single-deck screen due to the manner in which material is distributed to the lower decks. For lightly loaded decks carrying free-flowing material, or for wet-screening applications, the larger factor of the range should be used, while the smaller factor would apply to more difficultly separated materials.

Deck Factor (D)	
Top	1.00
2nd	0.95–0.85
3d	0.80–0.75

Fines Factor (*F*)—This factor probably is the most significant of all and is used to correct for the quantity of fine material in the feed to each deck. This factor is unity when 40% of the material to be screened is less than one-half the separation size. This criterion is selected because nearly all crushers tend to produce a product that is about 40% finer than one-half the crusher setting. For example, when screening is being conducted at ½ in., the amount of minus ¼-in. material, expressed as a percentage of the total feed to the ½-in. deck, determines the factor. Importantly, the fines factor should be based on the feed to a given deck, not on the feed to the screen unless determination of the top-deck size is being made.

Fines Factor (F)	
10 %	0.55
20 %	0.70
30 %	0.80
40 %	1.00
50 %	1.20
60 %	1.40
70 %	1.80
80 %	2.20
90 %	3.00

Wet Screening Factor (*W*)—Care is suggested in applying this factor, which is used to make proper allowance for the increase in the amount of material which can be made to pass through a square foot of screen surface when ample water is mixed with the material to make a slurry. It usually is better to make the slurry prior to introduction to the screen rather than to attempt to do all of the wetting on the screening surfaces. Subsequent rows of sprays can be applied in various manners to provide an oversize product that usually is more free of contaminating fines. It is recommended that this factor be applied conservatively, since the increased cost of a foot or two in screen length or width is little if size separation is assured without difficulty.

Wet Screening Factor (W)

−20 mesh	1.25
−10 + 20 mesh	2.0 −3.0
−½ in. + 4 mesh	3.0 −3.5
−1 in. + ½ in.	1.5 −2.0
+1 in.	1.25

Note that with the small and coarse particles, the effect of washing is minimal. Further, because a range of factors is presented, when the material is clayey or not free-flowing, the low value should be used. When the material is sandy and/or free-flowing, the high value should apply. Experienced judgment is a prerequisite here.

Type of Deck Factor (T)—The configuration of the screen surface also plays a role in the capacity of a given screen. The openings of rectangular, long-slot and rod-deck surfaces present greater percentages of open area for particle passage than do decks with square openings. Typical configurations are shown in Figs. 27-12, 27-13 and 27-14.

When rods are positioned at right angles to the flow of the material, the material will pass through the surface about 30% faster than with a square-mesh deck. Thus, a factor of 1.30 should be used with this screen configuration. Likewise, factors for other openings approximate the following:

Type of Deck Factor (T)

Square	1.00
Rectangular (ton-cap.)	1.10–1.15
Long-slot (Tyler rod)	1.15–1.20
Rod-Deck (right-angle)	1.30
Rod-Deck (slot parallel)	1.40

Bulk-Density Factor (B)—This factor takes into account variations in weight of material on the basis of bulk density compared to a material weighing 100 lb per cu ft.

The slope of a screen deck affects the velocity at which the material moves along the deck and, hence, the depth of bed and rate of classification for any given condition of amplitude and speed. With a slope of 15 to 18°, most materials will flow at about 60 fpm; 20°, about 90 fpm; 25°, about 120 fpm. These figures are for normal sand, gravel, crushed stone and ores. Usually, material velocity on horizontal machines will be about 40 to 45 fpm.

The relationship between tons per hour of feed to each surface for different weight materials and screen width is shown in Fig. 27-15. The width should be great enough to establish a depth of bed of up to about four particles of the separation size at the discharge end of the deck. For example, for separation at ½ in., the bed should be about 2 in. deep at the discharge end. The screen must be long enough to allow time for fines to classify through the oversize and pass through the deck.

At the feed end, depth is restricted to standard machine designs or other practical considerations. For instance, when scalping minus 6-in. rock, the feed might be about a foot or only two particles deep. When screening at 1½ in., a bed depth of six particles might be used. At 10 mesh and finer, a bed depth of 10 particles is common.

An example of the application of the foregoing factors for determining the size of screen needed for a given size separation is given in the following:

Conditions:

1. Separation at ½ in. on second deck.
2. Feed to second deck contains 60 % minus ¼-in.
3. Specifications call for not more than 10 % minus ½-in. in the oversize from the second deck.

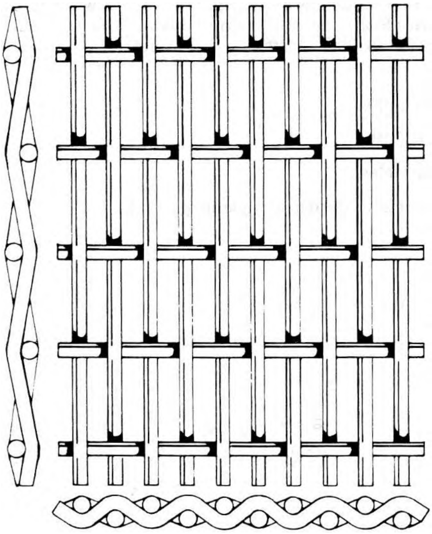

Fig. 27-12—Rectangular screen surface (*courtesy W. S. Tyler Co.*).

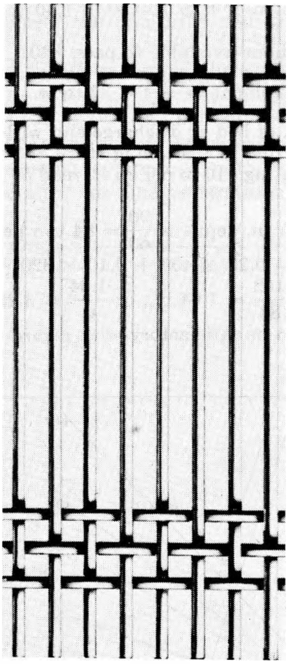

Fig. 27-13—Long slot screen surface (*courtesy W. S. Tyler Co.*).

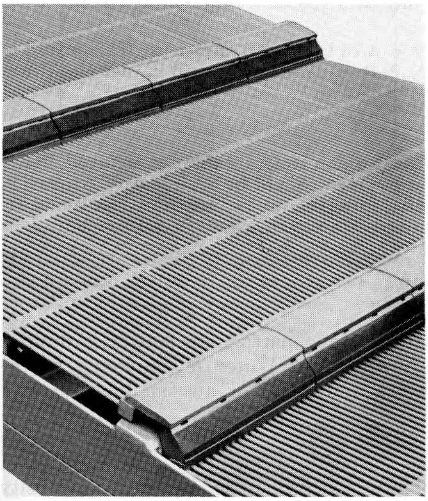

Fig. 27-14—Rod deck screen surface (*courtesy W. S. Tyler Co.*).

4. Cloth will be ½ × 1-in. opening from 0.192 wire.
5. Crushed rock weighs 110 lb per cu ft (compared to chart's basis of 100).
6. Feed rate is 400 tph with 80 % minus ½-in.
7. Ample water is available for wet screening.

The reading from the chart in Fig. 27-11 under these conditions is 1.75 tph per sq ft. The factors for efficiency, location of deck, fines, wet screening, type of deck and bulk density are: 1.25, 0.90, 1.40, 3.00, 1.10 and 1.10, respectively.

Capacity = 1.75 × 1.25 × 0.90 × 1.40 × 3.00 × 1.10 × 1.10 = 10.0 tph per sq ft passing.

Tons per hour available to pass, 320.

Area requirement $= \frac{320}{10} = 32$ sq ft.

For depth of bed at discharge end with screen at 20° slope (Fig. 27-15), allowing 6-in. loss of width due to tensioning strips to secure cloth, try 4-ft-wide screen (3 ft 6 in. net cloth); estimating 110 lb per cu ft, read 56 tph @ 60 fpm. At 20°, assume material velocity to be 90 fpm:

56 tph per in. depth $\times \frac{90}{60} = 84$ tph per in. depth.

Oversize = 0.20 × 400 + 0.10 × 320 = 112 tph.

Depth $= \frac{112}{84} = 1.64$ in., or $\frac{1.64}{\frac{1}{2}} = 3.28$ particles.

This would be satisfactory with regard to particle depth at discharge end.

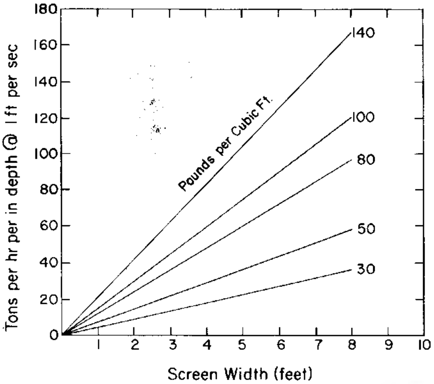

Fig. 27-15—Relationship between throughput per inch of bed depth for solids traversing at 1 fps as a function of screen width.

At the feed end, the depth will be 400 ÷ 84 = 4.76 in., or 9.52 particles. This would be unsatisfactory because the bed is too deep. Try 5-ft-wide screen (4 ft 6 in. net cloth) and read 74 tph @ 60 fpm from Fig. 27-15.

74 tph $\times \frac{90}{60} = 111$ tph per in. depth.

Depth = 400 ÷ 111 = 3.6 in., or 7.2 particles, which is acceptable for the feed end.

Based on these calculations, the length of the screen should be 32 ÷ 4.5 = 7.1 ft. The screen selected would then be a 5 × 8-ft unit. However, allowance must be made for surges and variances, and the next larger standard-size screen would be recommended, or 5 × 10-ft.

27.6.2—CLASSIFICATION

Mechanical Classifiers—When size separation of fine particulate matter is desired, mechanical classifiers or cyclones frequently are used. Water is employed as the suspending medium with mechanical classifiers, and water or air can be used with cyclones.

With both types of units, the separation size is based on the relative velocity with which a particle moves through the suspending medium. Although other factors are important, these velocities are determined principally by the size, density and shape of the particulate matter.

Separation of particles in a mechanical classifier occurs in the settling pool, or trough, of the unit. Solids larger than the separation size report to the bottom of the settling pool and are removed from the tank by a sand-transporting device, which may be in the form of a rake or spiral (Sec. 28.3.3). The effective pool area can be controlled by the shape and slope of the tank, the height of the overflow weir, and the position of the feed apron. The slope of rake classifiers normally is about 2 to 3 in. per ft, while spiral classifiers operate at a slope of 3 to 4 in. per ft. An increase in slope decreases the pool area and thus increases the separation size.

Spiral classifiers are rated on the basis of the diameter of the spiral expressed in inches and vary from about 10 to 100 in. in diam. The recommended speeds vary from 6 rpm for 78-in. spirals to 20 rpm for 12-in. spirals.[115] In addition, these units are characterized according to the position of the spiral in the tank. The high-weir type of spiral classifier is the most common. Another is the submerged-spiral type which provides high capacities for fine separations. The third type is a low-weir classifier designed especially for dewatering granular materials. The capacity for various size separations utilizing a high-weir classifier is shown in Figs. 27-16 and 27-17.[115] Since a duplex unit has two spirals, overflow capacity is doubled when this arrangement is used.

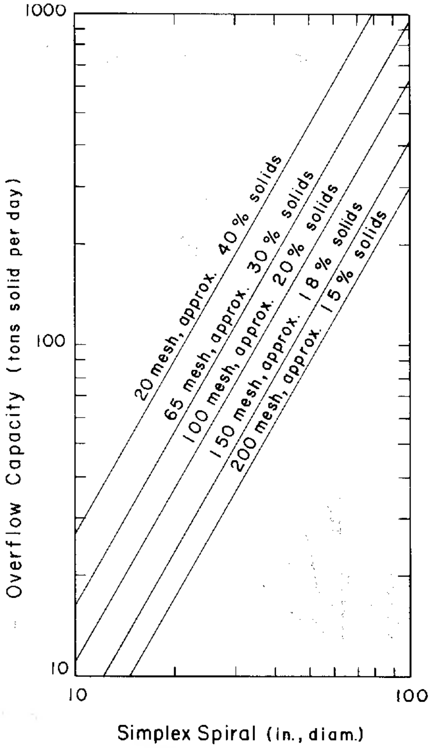

Fig. 27-16—Relationship between overflow capacity and spiral diameter for various separation sizes for spiral classifier; specific gravity of solids, 2.70 (*after Taggart*[115]).

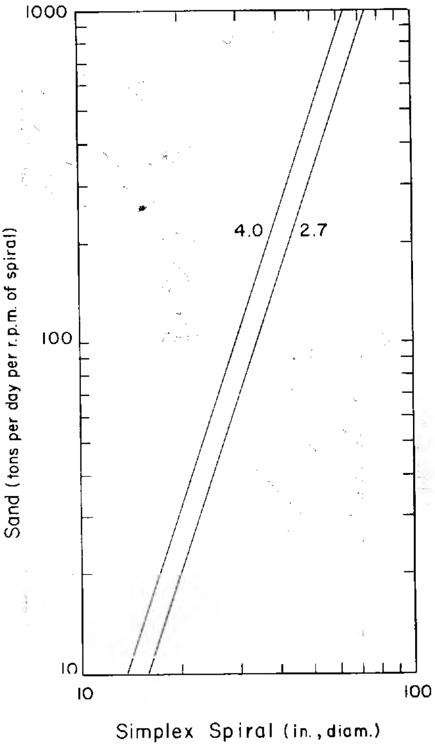

Fig. 27-17—Relationship between sand capacity and spiral diameter for solids of two specific gravities for spiral classifier (*after Taggart*[115]).

Assuming that an overflow discharge of 50 tpd is required at a separation size of 100 mesh and 20% solids by weight, a 24-in. Simplex spiral could be used. Dilution of the pulp overflowing the classifier also is an important factor, and if a pulp dilution greater than 20% solids is desired, the overflow capacity will be reduced.

The amount of classified sand that can be produced with this unit is 35 tpd per rpm of the spiral (Fig. 27-17). For example, operating the spiral at 10 rpm, 350 tpd of sand will be discharged.

Similar data for capacities obtainable with a rake classifier are presented in Figs. 27-18 and 27-19.[115]

Hydrocyclones—As a sizing device, the hydrocyclone generally is preferred today over a mechanical classifier. This is due to the saving in floor space for a given classifying capacity and a considerable saving in capital cost.

The hydrocyclone is a conical-cylindrical apparatus which operates under pressure. Hydrocyclones usually are constructed of fabricated steel or cast metal and may be lined with rubber or ceramic material. Their size is designated by the diameter of the cylindrical section or the cone base, D_c. The slurry is pumped into the unit tangentially and, after some residence time, leaves via the apex, the coarse product discharge, D_u, and the vortex finder, the fine product discharge, D_o. A schematic representation of the cyclone is shown in Fig. 27-20.[116]

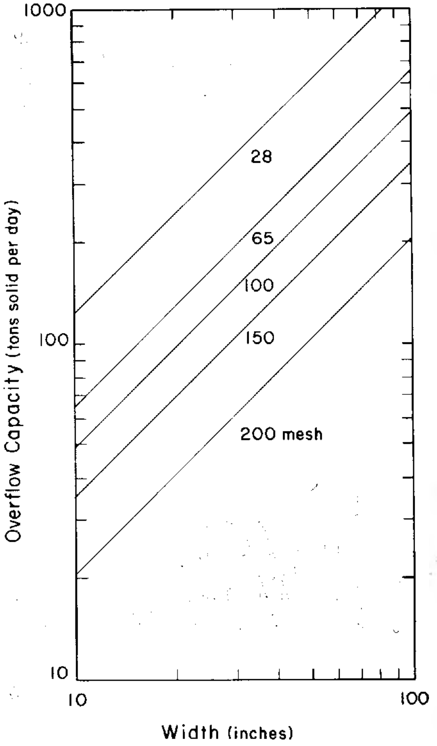

Fig. 27-18—Relationships between overflow capacity for various separation sizes for rake classifier; specific gravity of solids, 2.70 (*after Taggart*[115]).

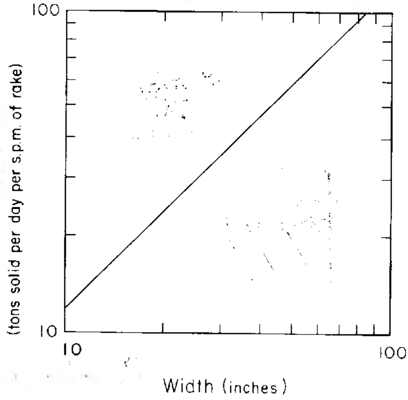

Fig. 27-19—Relationship between sand capacity and weir width for rake classifier (*after Taggart*[115]).

The hydrocyclone operates at pressures usually exceeding 5 psi and converts this energy into rotational fluid-solid motion. As a consequence, particles are separated according to their mass, but unlike mechanical classifiers the force acting on particles in a hydrocyclone is much greater than the normal gravitational force responsible for sedimentation in the mechanical classifier.

The efficiency of a cyclone is determined by both design variables, such as size, vortex-finder diameter, length of vortex finder, cone angle, and design and area of feed port, and by operating variables such as pressure, pulp density of feed, particle-size distribution and apex diameter.

The relation between cyclone size and geometry for a given classification operation is complex. As a result the relationships between size of separation and such parameters as capacity, inlet and vortex-finder diameters, and pressure have been established empirically. Dahlstrom[117] has presented the following equations for systems containing less than about 25% solids by weight:

$$d_{50} = \frac{81(D_oD_i)^{0.68}}{(Gpm)_f^{0.53}}\left[\frac{1.73}{\rho_s - \rho_o}\right]^{0.5}$$

and

$$(Gpm)_f = 6(D_oD_i)^{0.9}P^{0.5},$$

where d_{50} is separation size, which is that feed size which distributes itself equally between the overflow and underflow products, in microns; D_o is vortex finder diameter,

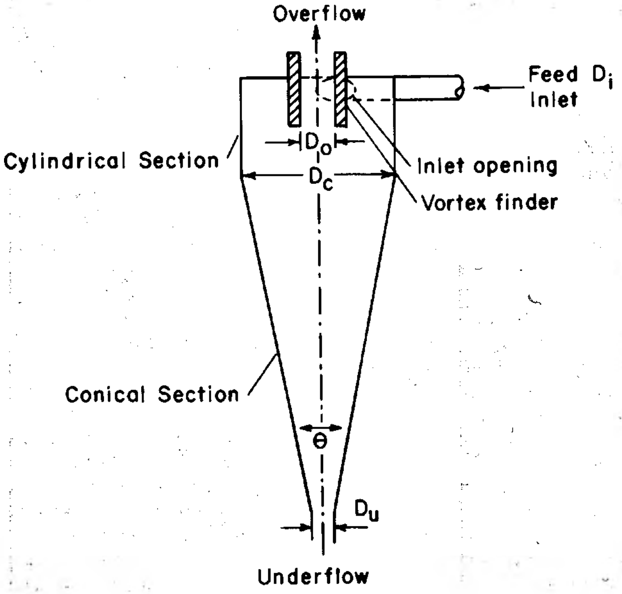

Fig. 27-20—Principle features of a hydrocyclone (*after Bradley*[116]).

in.,; D_i is feed inlet diameter, in.,; D_c is cyclone diameter, in.,; Gpm_f is cyclone capacity, gpm water in the feed; ρ_s is specific gravity of solid; ρ_o is specific gravity of liquid; P is pressure, ft water.

When these relationships are combined with two commonly used design criteria,[118] i.e., $\frac{D_c}{3} \geq D_o \geq \frac{D_c}{4}$ and $D_o = 1.4D_i$, capacity and separation size can be calculated. For example, for solids of specific gravity of 2.70 in water and with an operating pressure of 10 psig, the following relationships are obtained.

For $D_o = D_c/3$:

$$Gpm_f = 2.87D_c^{1.8}$$
$$d_{50} = 8.26D_c^{0.4}.$$

For $D_o = D_c/4$:

$$Gpm_f = 1.71D_c^{1.8}$$
$$d_{50} = 7.34D_c^{0.4}.$$

Cyclone capacity is presented as a function of cyclone diameter in Fig. 27-21. Because of the limits in the relationship between vortex-finder and the feed-inlet diameters, the relationship between capacity and cyclone diameter is represented by a band rather than a single line.

With regard to separation size and cyclone diameter, the separation size, d_{95-99}, frequently is used as the design criterion rather than d_{50}. The quantity d_{95-99} represents the range in size of particles of which 95 to 99% are recovered in the underflow. Assuming a typical efficiency curve, the separation size calculated for d_{50} should be multiplied by a factor of 2.75 to convert d_{50} to d_{95-99}. A plot of d_{95-99} as a function of cyclone diameter for solids of specific gravity 2.70 and an operating pressure of 10 psig is given in Fig. 27-22. If a size separation is desired at 325 mesh (44μ) for solids of this density, a 6-in. cyclone should be used at 10 psig. The capacity would be 50 to 60 gpm of feed water contained in the pulp.

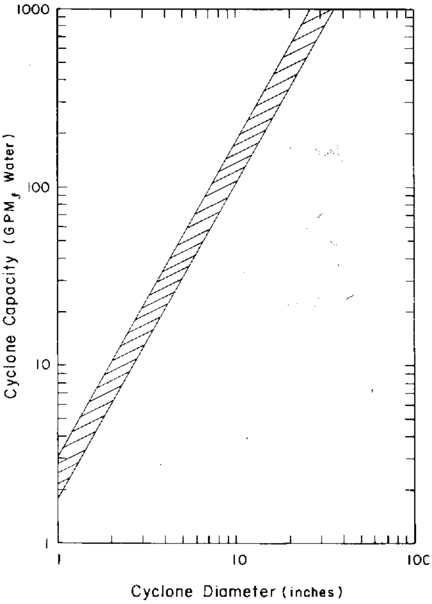

Fig. 27-21—Relationship between cyclone capacity in gpm of feed water as a function of cyclone diameter; pressure, 10 psig.

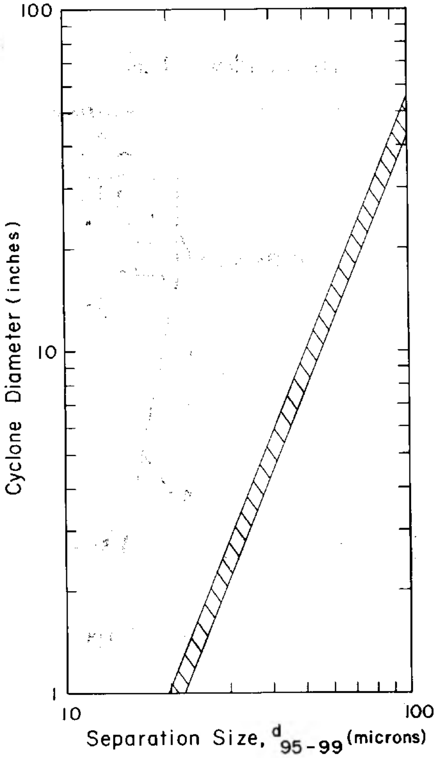

Fig. 27-22—Relationship between separation size, d_{95-99}, and cyclone diameter; specific gravity of solids, 2.70; pressure, 10 psig.

27.7—CONCENTRATION

CARL RAMPACEK

Mineral deposits usually consist of mixtures of minerals more or less intimately associated. They vary in size, scope and quality, depending upon the geologic origin and history of the deposit and upon the rock types present. Mineral deposits are ores only when it is economically possible to extract the agglomerate mixtures from the earth, recover the desired minerals, and convert them into a form useful to the consumer. As a general rule, except for some nonmetallic minerals and fossil fuels which are mined and used without further processing, most raw ores must be subjected to some form of concentration—physical beneficiation, or chemical and pyrometallurgical treatment methods—to produce marketable-quality metal or mineral products. One or more concentrating steps usually are required. The choice of processes is dictated chiefly by the types of minerals present and the complexity of their association in the ore.

The following subsections identify some of the processes most commonly used, their fields of applicability, and references for more detailed study.

27.7.1—FROTH FLOTATION

Froth flotation is the most widely used method of beneficiating complex and low-grade ores. Flotation is a complex physicochemical process taking place in an ore pulped with water, by which the surfaces of one or more minerals in the finely ground pulp are made water-repellent and responsive to attachment with air bubbles. Beneficiation is accomplished as the mineral-laden bubbles rise to the surface, leaving behind minerals or ore particles which have not responded to the treatment. Under proper conditions, some degree of flotability can be imparted to almost all minerals, but success is dependent on the ability selectively to differentiate between those minerals for which flotation is desired and those minerals which are to remain in the tailings. In most instances, the minerals are caused either to float or not to float by judicious use of flotation chemicals or reagents which become attached to mineral surfaces through physical or chemical sorption. The nature and extent of this sorption are dependent upon surface charges assumed by the mineral in the aqueous pulp.

Collectors—Success in froth flotation applications often is dependent upon a balancing of reagents with different specific functions. Collectors (or promoters) are employed to coat selectively the particles to be floated with a water-repellent surface that will adhere to air bubbles. In general, collectors are heteropolar compounds, are ionizable, and are at least partially miscible with or soluble in water. At an air-water interface the polar end of the compound, being water soluble, is oriented toward the water phase and the nonpolar end toward the air phase. Attraction between the polar grouping of the reagent and the polar mineral surface results in adsorption, leaving the long-chain nonpolar hydrocarbon end generally oriented away from the mineral surface. Bubble attachment occurs at the nonpolar segment of the collector ion, thereby completing the mechanism by which the mineral particle may be rafted to the surface of the pulp. Collectors are classified as anionic if the polar end carries a negative charge and cationic if the polar end carries a positive charge.

Modifiers—Selectivity in mineral separations by flotation is achieved by using various modifying reagents. Such chemicals are classified into four major groups:

Regulators (of pH): NaOH, CaO, Na_2CO_3, H_2SO_4 and H_2SO_3. Regulation of pH is important in that ionization of collectors occurs within a specific pH range. Further, many minerals best respond to flotation within a specific pH range with a given collector, and this pH range is bracketed by ranges in which flotation proceeds poorly, if at all.

Activators: Reagents which promote the collection of minerals when added in conjunction with a regular collector. Examples are Cu ion for flotation of sphalerite with xanthates, or Ca ion to float quartz with fatty acid collectors.

Depressants: Reagents which are used to prevent or suppress flotation of one mineral without impairing the flotation of another. Depressants are used when the flotabilities of constituent minerals with a given collector are similar, and are the basis for differential or selective flotation. Examples of depressants are lime, sodium cyanide, sodium silicate, chromates, sodium fluoride, tannin, starch and other natural colloids, phosphates, and a wide variety of proprietary compounds.

Dispersants: Reagents used to deflocculate and clean mineral surfaces. Dispersants also frequently function as pH regulators and depressants and include such reagents as sodium carbonate, sodium silicates, tannins and the complex phosphates.

Frothers—Frothers are heteropolar organic substances with surface-active properties. Being surface-active, frothers tend to be adsorbed at the air-water interface in greater concentrations than in the bulk liquid and to lower the interfacial surface tension. A practical frother should be inexpensive, promote the formation of froth having the desired characteristics of stiffness or fragility, and have little or no collecting property, except for minerals classified as natural floaters. Examples of frothers are pine oil, aliphatic alcohols, alkyl ethers of polypropylene glycol, and cresylic acid.

Oils—Included in this category are kerosene, various fuel oils and wood-tar and coal-tar oils. They frequently are used as extenders to heteropolar collectors and as froth modifiers.

The amount of the various reagents used varies with the requirements of the ore being treated and its economic value. Also, with the total cost of all flotation reagents averaging about $0.112 per ton (1965 basis), it is obvious that the unit cost of the reagent must be considered in determining the permissible levels of use. Reagents vary in value from about $0.008 per lb for lime to more than $1 per lb for some of the aliphatic amines. Table 27-13 gives the range of normal reagent usage.

TABLE 27-13—Normal Amounts of Reagents Used in Flotation

Class of Reagents	Lb of Reagents per Ton of Ore
Collectors	0.02 to 2.00
pH regulators	0.02 to 5.00
Depressants, dispersants	0.02 to 2.00
Activators	0.50 to 4.00
Frothers	0.01 to 0.50
Hydrocarbon oils	0.50 to 5.00

Applications—Froth flotation is the dominant mineral dressing process in current use, by which over 90% of the world's copper, lead, zinc, molybdenum, antimony and nickel is recovered. A substantial proportion of phosphate and potash also is obtained through this process. In the United States, more than 200 million tons of sulfide ore, 50 million tons of nonmetallic ores, 15 million tons of iron ore, and 10 million tons of coal are processed by flotation each year. In terms of total tonnages undergoing some form of froth flotation treatment, all commodities may be expected to show a continued growth, but especially the latecomers, such as iron and coal.

Preparation of the Ore—For effective flotation to take place, the ore must be ground (a) to a size that will insure an adequate degree of liberation between mineral constituents, and/or (b) to a size that can be accommodated by the froth flotation process. Maximum particle size will vary with the ore treated, being about 48 to 65 mesh for metallic sulfides, but flotation of pebble phosphate as coarse as 14 mesh has been reported.

Inadequate grinding of the ore can lead to a loss of coarse mineral to the nonfloat fraction. Conversely, overgrinding, besides representing an unnecessary cost, may create fine-size fractions (slimes) which are reluctant to float, consume excessive amounts of reagents and reduce selectivity. In some instances, effective flotation may require desliming prior to flotation. Desliming may reject minus 150-mesh particles, as in the phosphate industry, but usually is carried out to discard minus 10μ material.

After grinding and sizing to within the desired range, the ore usually is routed to mixers or conditioners where reagents are added in a predetermined sequence.

In the conditioning step, pulp density, time, temperature and intensity (power input) are among the variables which influence subsequent flotation. Density of the pulp undergoing conditioning is frequently as high as 70% solids for nonmetallics and sulfides. In other circumstances, conditioning in more dilute pulps may be favored.

Flotation Separation—Separation of prepared pulps takes place in flotation machines, which essentially are open troughs with means of introducing and dispersing air into the pulp. The mineralized air bubbles or froth collected in the rougher separation customarily are cleaned by routing through additional flotation machines to drop the misplaced, poorly liberated or poorly reagentized minerals. These middlings are routed within the circuit to reclaim additional values. The cleaned products are the *concentrates,* and the waste products are contained in the *tailings.*

Two general types of machines—pneumatic and mechanical—are used for mineral flotation.

Pneumatic, or air, cells are the simplest, consisting of a long open trough through which pulped ore flows. Gas bubbles are introduced into the medium through the porous bottom (Callow type) or through regularly spaced air lances (Forrester type). In pneumatic cells, the air flow also provides agitation of pulp. While the actual number of plants with air cells is relatively small, the majority of these units are used in concentration of copper ores and handle fairly large tonnages. The cell components usually are specially designed and made for a specific plant operation or application. Air cells feature low power costs and often are favored for cleaning operations. Cost of maintenance varies. Cells of the air-lance type should be relatively free of the need for maintenance; Callow cells, on the other hand, suffer from blinding of the porous bottom from slimes and chemicals—a condition which calls for extensive maintenance. The trend appears to be away from cells of this type and toward large-volume mechanical cells.

The more popular mechanical flotation cells (Sec. 28.3.4) differ from pneumatic cells in that agitation is provided by a rotating impeller through which air is introduced into the suspension. Air requirements may be supplied from the atmosphere, being drawn into the vortex created by the mechanical action of the impeller, or low-pressure compressed air may be used to supply or increase the pulp aeration. The three most prominent makes of mechanical flotation machines are the Agitair, Galigher Co.; Denver, Joy Mfg. Co.; and Wemco-Fagergren, Envirotech Systems, Inc.

Prior to about 1965 the size of the flotation cells ranged from 40 to 100 cu ft, but today cells from 200 to 400 cu ft are common and units up to 500 cu ft are being manufactured. In addition to the advantages of lower power cost and the ability to float coarser material, large cells offer lower maintenance costs, simpler flowsheets, lower labor costs, and new opportunities for automation and close control.

Another type of flotation mechanism is the Cyclo-Cell, manufactured by Heyl & Patterson, Inc., being used in the coal industry. Agitation is achieved by submerged vortex chambers which discharge a high-velocity jet of water in the form of a hollow cone. Air is piped to the center of this cone, where it is sheared into a multitude of uniformly minute bubbles and released to be dispersed throughout the cell. Simplicity of design, high capacity, and low operating and maintenance costs are claimed for the machine.

27.7.2—GRAVITY CONCENTRATION

Gravity concentration is a method of separating solids of different specific gravities in a fluid medium—often water or air. Mineral mixtures susceptible to separation by gravity methods are those in which valuable mineral and gangue differ appreciably in specific gravity. For simple methods with water as the medium, a specific-gravity differential of at least 1.5 is desirable. Sizing of feed is important because when two particles of the same specific gravity are placed in a fluid the larger sinks faster. Thus a small particle of high specific gravity may have the same settling rate as a larger particle of lower specific gravity. Shape, too, affects separation because round grains settle faster than flat or long narrow grains.

Simple Sluice—One of the most elementary systems of concentration—sluicing—may be seen in most mineralized areas after a rain. The surface waters have washed down whatever sand they can shift, leaving grains of heavy minerals concentrated in natural depressions lying along their course. Sluicing methods have long been used to recover high-value heavy minerals, such as gold and silver. The simplest form consists of a stationary inclined plane, sloping downward with just sufficient inclination for a film of water and mineral particles to flow gently. The light mineral particles roll down, while the heavier are stopped in free-fall traps, riffles, or removable cloth surface.

Pinched Sluice—Basically, the pinched sluice is an inclined launder 2 to 3 ft long, narrowing from about 9 in. in width at the feed end to 1 in. at discharge. The pulp enters gently and stratifies as it descends. At the discharge end, these strata are separated by splitters.

Humphrey's Spiral—This gravity concentration device essentially is a helical sluice consisting of four to six turns of troughing. The separating action is complex. It combines centrifugal action with multiplane sluicing in a partially controlled medium. The descending ribbon of pulp can be seen to spread, the heaviest and coarsest particles remaining nearest the center and on the lowest part of the cross section, while the lightest and finest material climbs well up the outer side. The innermost part of the stream is diverted by splitters for removal of the heavy concentrate. A series of splitters is arranged along the entire six turns. In spite of the complexity of the principles involved, the device is very simple to operate and has no moving parts.

Sink-Float—When insoluble solids of different specific gravities are placed in a liquid whose specific gravity is between those of the solids, a separation is made readily. Minerals have long been separated in laboratories, using heavy organic liquids, such as mixtures of acetylene tetrabromide and carbon tetrachloride, as the separation media. Toxicity, high cost and difficulty of media recovery have precluded industrial acceptance of organic heavy-liquid separation. The first commercial application of the principle was to remove shale from coal using a saturated calcium chloride solution. About 1930, it was discovered that under certain conditions water suspensions of heavy minerals behave like a true heavy liquid. Sand, barite and galena were all used but have been replaced by magnetite and ferrosilicon. Either of these solids in a water suspension can give specific gravities ranging from 1.3 to 3.4.

For open-vessel heavy-media separation, ore is crushed only to the degree that the specific gravity difference between the "sink" and "float" products becomes evident. Pieces of ore as large as 12 in. have been used. Fine material is removed by screening, usually at 10 mesh, since fine material makes medium recovery more difficult and slimes tend to make the slurry viscous. Ore is fed to the medium vessel. The "float" material is flowed over a weir, and the "sink" is continuously removed from the bottom. Both products are washed on screens, and the diluted slurry is treated by magnetic separation to recover the magnetite or ferrosilicon.

In recent years hydraulic cyclones have been used as the separation vessels. The centrifugal and centripetal forces of the cyclone action permit magnetite slurry with specific gravity of 2.1 to 2.5 to make a separation equivalent of 2.8 to 3.0 gravity. Feed normally is crushed to ⅜ in. or finer. Minimum size can be extended to 28 mesh, although separation efficiency at less than 10 mesh is low.

Jig—The jig, earliest of the mechanized concentrating devices, consists essentially of a submerged screen that supports a bed of ore. The ore is stratified by the action of two currents of water, an upward and a downward, alternating with each other in quick succession.

Particles of different specific gravities arrange themselves according to size and specific gravity during the pulsating movement of the jig. The products of the jig are the tailing which forms the top layer, the coarse concentrate which forms the heavy layer on the screen, and the fine concentrate which goes through the screen to form a "hutch product."

Jigs are used extensively in coal cleaning on material as coarse as 7 in. For concentrating iron ore and barite, however, the maximum size usually is ¾ in. High-speed diaphragm jigs frequently are used in mill grinding circuits to recover gold or other heavy minerals.

Jigs are economical and easy to operate and can handle a wide size range of feed, but they require large amounts of water for clean mineral separations. The trend over the last 40 yr has been to replace jigs by shaking tables for treating minus 10-mesh ore, and by sink-float systems for treating larger sizes.

Shaking Table—A shaking table is a tilted quadrilateral deck that is vibrated longitudinally with a slow forward motion and a rapid return. The deck is riffled with cleats that taper in thickness from ½ in. at the feed end to a thin edge near the other end. A film of water flows over and down the sloped deck, and slurried feed (25 to 30% solids) is introduced at the corner near the head-motion mechanism. The particles fan out as they move toward the concentrate end. Small heavy particles ride highest on the table. Light particles are washed off the lower side. Unsized feed sometimes is used, but tables operate much more efficiently with classified feeds between 8 and 100 mesh.

Because of limited capacity, the table has been replaced in many instances by the Humphrey's spiral. The table now is used chiefly to concentrate fine coal and ores of tungsten, tin and chromium.

Dry Concentrators—Various forms of dry concentration have been devised for use in arid areas. In the pneumatic jig, air takes the place of water as the pulsating medium used to dilate the bed of ore and promote stratification. Pneumatic tables use a throwing motion to move the feed along a flattish riffled deck, and blow air continuously up through a porous bed. The general principles of separation are similar to those applied in wet tabling. The use of dry separation methods is rather limited.

27.7.3—MAGNETIC SEPARATION

Magnetic separation comprises sorting one solid from another by means of a magnetic field and is based upon the principle that particles placed in a magnetic field are either attracted (paramagnetic) or repelled (diamagnetic) by it. For practical purposes, paramagnetic solids are considered in three broad categories: highly magnetic, weakly magnetic and nonmagnetic. The only important highly magnetic mineral is magnetite. Many other minerals have measurable magnetic susceptibilities, but fewer than 20 are amenable to magnetic separation, and these are classed as weakly magnetic. The remainder are considered to be nonmagnetic.

Magnetic separation is conducted dry or wet. Either may use low- or high-intensity magnets. Although definitions vary, low intensity usually is considered to be below 2,000 Gauss. High intensity may extend to the saturation of iron—about 20,000 gauss. Low-intensity magnetic separators, both wet and dry, contain either permanent magnets or electromagnets. Permanent-magnet types are favored because of lower operating costs and higher dependability. Low-intensity dry magnetic separators are simplest, and generally are used only for removal of tramp iron or for "cobbing" coarse iron ore.

Tramp iron, commonly defined as metallic iron coarser than ⅛ in. can cause dangerous and costly equipment damage if permitted to pass through crushers, grinders and pulverizers. Consequently, magnets often are used in concentration plants to remove tramp iron from the ore stream. One kind in common use is suspended flat above a conveyor belt or at an angle over the conveyor head pulley. Selection of size and type is influenced by the size of the tramp iron, belt speed and average burden depth. Magnetic head pulleys also are common. These may be electromagnetic, but recent installations have favored radial-pole permanent-magnet pulleys for pulley diameters and belt widths up to 5 ft. Belt speeds of as much as 600 fpm are allowable with either type. Plate magnets, or grate magnets, sometimes are used to remove tramp iron from the ore by trapping it in a chute or on the bars of a grizzly. Both are accumulating-type magnets, and the flow of ore must be interrupted periodically to permit removal of the material collected.

Cobbing is the treatment of lump ore or coarse sands to concentrate particles with high magnetic response. Dry cobbing, common until a few years ago, now is practiced at only two plants in the United States. Magnetic drums are used in dry cobbing, giving a magnetic field pattern similar to that of radial magnetic pulleys. A more complex type of drum separator to recover a rough concentrate from magnetite ore makes use of alternate polarity magnets of moderate intensity. The pulsing action of the magnet imparts sufficient agitation to shake out nonmagnetic particles, thus producing a higher-grade concentrate. Wet cobbing is restricted to treatment of ore finer than ¼ in. This method has gained in popularity since technology has been developed to exploit the fine-grained taconite iron ores of Minnesota and Michigan. Currently, it is used to treat millions of tons annually since it can treat large tonnages at low cost. Concurrent drum separators are used to lift the magnetic particles from a stream of ore pulped with water. The nonmagnetic particles are washed out as tailings.

Whether the cobbing is done wet or dry, final concentration of magnetite ores is effected in wet-drum separators. A slurry of finely ground ore and water is passed through the machine. Magnetic material adheres to the revolving drum and is removed by sprays. Finer grinding and reseparation are conducted until the desired grade of concentrate is attained. Usually, a concentrate is made that contains 62 to 65% iron.

Another type of wet low-intensity separator is the magnetic filter. Unlike drum separators, these are not designed for continuous discharge of magnetics. Their use, therefore, is restricted to removal of relatively small amounts of fine magnetic particles from a large amount of nonmagnetic feed material.

There are two main varieties of high-intensity dry magnetic separators. In the crossbelt type, weakly magnetic minerals, such as chromite and wolframite, are lifted from the moving stream of ore and moved from the field by a fast-moving magnetic crossbelt. Control is accomplished by varying the distance between the magnet poles and by changing the amount of current to the magnet windings. High-intensity dry magnetic separators of the induced roll type are used in Florida and abroad to separate the valuable components of beach sand, such as rutile, ilmenite and zircon. They also are used to concentrate specular hematite ores in Canada.

Several new developments have been introduced since the mid-1950s. Increases in field intensities have been achieved by replacing Alnico V magnets with lighter and stronger magnets of barium-ferrite ceramic. New high-speed high-intensity dry magnetic separators now are being used to concentrate minus 100-mesh magnetite, but generally are not considered effective for material smaller than 200 mesh. Significant progress has been made toward development of practical high-intensity wet magnetic separators. At least four types have been proved feasible on a laboratory scale, but few plant-scale tests have been tried.

27.7.4—ELECTROSTATIC SEPARATION

Electrostatic separation of mineral grains is an integral part of the treatment of beach sands all over the world. Dry particles subjected to a surface electrical charge, on or before entering an electrostatic field, behave in accordance with their ability to conduct electricity. Conductive particles are repelled by the active electrode submitting the charge. Dielectric (nonconductive) particles will be attracted by it. The dielectric particles become charged to different degrees and therefore are attracted to the electrode at varied rates. Concentration of the component minerals is effected by proper placement of splitters in the falling grain stream.

Electrification, or charging, mechanisms are of three general types: contact or frictional, conductive induction and ionic bombardment. Ionic bombardment, sometimes known as "high tension," is very effective for charging solid particles. Very good separations can be achieved, especially when "high tension" is combined with conductive induction.

Commercial electrostatic separators are based on a grounded rotor revolving in an electric field. The electric field is controlled by the voltage applied to one or more electrodes in the vicinity of the rotor but not in contact with it, or by positioning the electrode. Particles fed onto the surface of the rotor either receive the charge (dielectric) or are grounded to the rotor (conductive) and are separated accordingly. Best particle size range is between 8 and 70 mesh. Dry particle surfaces are mandatory, and extremely fine particles impair clean separations. Pretreatment, therefore, may include dedusting, desliming and, in some cases, drying and heat-treating prior to electrification.

Electrostatic separation is used not only for recovery of ilmenite, rutile and zircon from beach sand concentrates but also for removing feldspar and mica from quartz, iron oxide from silica sand and abrasives from grindings.

27.7.5—EXTRACTIVE METALLURGICAL PROCESSING OF ORES AND CONCENTRATES

Extractive metallurgy is the branch of chemical engineering concerned with recovering metals and metallic compounds of usable purity from ores and mineral concentrates. Pyrometallurgy, hydrometallurgy and electrometallurgy are the interrelated and usually integrated subdivisions of extractive metallurgy, whereby the mineral constituents of ore and concentrates are chemically altered to produce marketable-quality metals and metallic compounds.

Pyrometallurgy—Pyrometallurgy encompasses all processing operations using refractory-lined furnaces and high temperatures created by electrical energy or burning fuels to produce refined metals from ores and concentrates. Drying, roasting, sintering, distilling, smelting and fire refining techniques are the major unit processes employed. Effective separation and recovery of refined metals by pyrometallurgical methods depends upon the relative reactivity of the component minerals to fluxes, oxidants and reductants, and upon the differences in volatility, fusibility and density of the products formed.

Drying, roasting and sintering are subsidiary unit operations that are widely employed to prepare ores and concentrates for smelting or leaching. Drying is a low-temperature operation done in furnaces designed to remove excess water efficiently. Roasting employs heat in hearth, fluid-bed, flash and rotary-kiln-type furnaces to alter, without fusion, the chemical form of minerals in the raw materials in oxidizing, reducing or neutral atmospheres. Sintering and closely related nodulizing differ from roasting in that higher temperatures are used to effect agglomeration of fine material particles by incipient fusion as well as by chemical change. Stationary and moving pallet hearths, employing up or down draft firing, are used in sintering. Countercurrent- or concurrent-fired rotary kilns are used for the nodulizing process.

Because of their relatively high vapor pressure at elevated temperatures, some metals are separated and recovered from ores, fumes and slags by distillation or fuming. Nearly all the world's primary mercury is recovered and refined by relatively low-temperature distillation of low-grade cinnabar ores using rotary kilns, hearth furnaces and retorts. Distillation also is widely employed to recover zinc from oxidized zinc concentrates and ores by retorting. Retorting practice in horizontal batch or vertical continuous retorts involves dead roasting of the concentrate to remove sulfur. The calcine, mostly zinc oxide, is mixed with coal and heated. Metallic zinc is condensed in the cool end of the retort. Zinc also is distilled in Imperial Smelting-type furnaces. Imperial Smelting practice comprises sintering of zinc-lead concentrates to remove sulfur, blast furnace smelting at a high temperature to volatilize the zinc and recover a lead bullion, and condensation of the zinc in a spray of molten lead followed by cooling the zinc-saturated molten lead to enable separation of zinc by skimming. Other metals and metal oxides recovered and refined by fuming, volatilization and distillation techniques include minor tonnages of arsenic, antimony, cadmium and lead.

Liquation is a process whereby metals and metallic compounds are separated by using differences in melting points. Stibnite, which has a low melting point, is recovered as relatively pure antimony sulfide for subsequent reduction in blast furnaces by heating coarsely crushed stibnite ores. Liquidation techniques are important in refining metals, notably lead. As produced by blast-furnace smelting, lead bullion contains copper, arsenic, antimony, gold and silver that must be removed as impurities or recovered as salable byproducts. Copper, arsenic and antimony are removed by cooling the lead bullion, adding elemental sulfur and sparging the bullion with air. When sulfidized and/or oxidized, these impurities become insoluble and can be removed as a dross floating on the molten lead. Gold and silver are removed by adding metallic zinc to form insoluble alloys that can be skimmed from the surface of the melt. Liquation also is an important unit process employed to separate copper-rich matte from heavier nickel-rich matte prior to subsequent processing to recover copper and nickel.

Smelting is by far the most important pyrometallurgical process by which metals are recovered from ores and concentrates in semirefined form. Drying, roasting, calcining, sintering, agglomeration and/or pelletizing commonly are used to prepare the feed for smelting. Either fire or electrolytic methods subsequently are used to remove residual impurities and to recover byproduct metals.

Smelting refers specifically to those high-temperature processes whereby ore and gangue minerals are chemically altered, fluxed and reduced to form a low-density molten slag and one or more heavier liquid metals or metallic compounds. Only relatively high-grade ores and concentrates can be effectively smelted, because smelting of low-grade materials entails use of excess amounts of fuel and flux to produce relatively little valuable metal and large amounts of valueless slag containing much of the metal that was in the ore.

In general, the smelting of metal oxide ores, concentrates and calcines involves reduction of the oxide to metal with coke, carbon monoxide formed by burning coke and, less frequently, iron. Matte smelting of sulfide ores and concentrates differs in that neutral or slightly oxidizing conditions are employed to form matte, an alloy of several metal sulfides. Reduction smelting is performed in blast furnaces and, less commonly, in reverberatory and electric arc furnaces using a mixture of coarse ore, coke and fluxes, and/or a sinter prepared from fine ore, coke, and flux. Matte smelting is done almost entirely in reverberatory-type furnaces using fine-sized ores, concentrates and fluxes. The high temperatures employed are achieved by burning natural gas, fuel oil or pulverized coal above the molten charge in the furnace.

With the exception of ferroalloys made for additives in steelmaking, all the metals and alloys produced by either reduction or matte smelting require further treatment and refining to produce marketable products. The making of steel from pig iron, commonly produced in blast furnaces, requires removal of carbon, sulfur and phosphorus by oxidation smelting with steel scrap, fluxes and air in converters or in reverberatory or basic oxygen furnaces. Lead bullion, also produced in blast furnaces, must be drossed and softened. The impurities must be removed by oxidation, by alloying or by electrolysis. Copper and copper-nickel mattes produced in reverberatory-type furnaces are converted to blister copper and copper-rich and nickel-rich metals by oxidation of sulfur and iron and slagging of the iron with silica flux. Fire-refining methods commonly are employed to purify aluminum, magnesium, zinc, tin, cadmium, cobalt, gold, silver and a variety of other minor metals produced by methods other than smelting.

All pyrometallurgical operations produce large volumes of gas containing a wide variety of vaporized metals, dust and fumes. Such processes often generate numerous byproduct slags, drosses and metal spills that usually are recycled for reprocessing or sold to other reduction plants for recovery. Major facets of pyrometallurgy include the efficient utilization of heat energy; the design of many different types of high-temperature furnaces, roasters, and kettles; the production of fluid discardable slags; and the selection of refractory furnace linings resistant to corrosion and erosion. Detailed information on these ancillary operations and technology is given in general metallurgy textbooks and references.

Hydrometallurgy—Hydrometallurgy includes a diversity of processes featuring the selective dissolution of metals from ores and concentrates and the subsequent recovery of relatively pure metal compounds or metals. The unit operations comprising hydrometallurgical processes invariably include preparation of the feed material, leaching with aqueous solution of an acidic or alkaline solvent, separation of the metal-bearing solution from the leach residue, and purification of solution followed by recovery of the purified metal compound.

Hydrometallurgical processing of ores and concentrates, except for the rare application of in-place leaching, requires preparation of the raw materials to assure effective leaching and product recovery. Ores must be crushed—and usually ground and sized—prior to leaching to permit effective contact between the ore minerals and the solvent. Although the oxide and carbonate minerals of many of the important metals are readily dissolved sulfide and silicate minerals must be treated to convert them to soluble form. Sulfide minerals may be roasted using air and/or chlorides to produce soluble oxides, sulfates or chlorides. Silicate and insoluble oxide minerals are roasted with reductants, chlorides, sulfur compounds or soda ash.

Dissolution of minerals in any of a variety of lixiviants is called leaching. Dilute low-cost sulfuric acid is the most important solvent used for leaching ores and concentrates. The chief alkaline solvents are the hydroxides and carbonates of sodium or ammonium. Ammonium carbonate solution is used for leaching certain copper and nickel minerals. Sodium and calcium cyanide solutions, containing enough dissolved lime for protective alkalinity, are used to extract gold and oxidized silver minerals from their ores. Use of acidified brines, chloride solution, or sulfurous acid is uncommon. Use of hydrochloric, nitric or other high-cost mineral acids is rare.

The most important of the various leaching methods is agitation of finely ground material in open vessels at atmospheric pressure. It is widely used to leach oxidized copper, uranium, vanadium, beryllium, manganese, gold, silver and nickel ores; roasted zinc and beryllium concentrates; and calcined bauxite. Agitation leaching in closed vessels, or autoclaves, at moderate-to-high pressure and temperature is of lesser importance and is only used on vanadium and uranium ores, tungsten and rare earth metal concentrates, and on complex sulfide concentrates containing copper, cobalt, and nickel.

Vat, heap and dump leaching comprise the other methods used to extract metals from ores. Vat leaching encompasses downward or upward percolation leaching of crushed or ground and deslimed ores that are carefully bedded in round or rectangular vats. The method is used to extract gold with cyanide solution and to win copper from oxidized copper ores with sulfuric acid solutions. Currently, heap and dump leaching techniques are used only in extracting copper from oxidized ores and low-grade porphyry mine strip wastes. Heap leaching involves carefully placing coarsely crushed ore over an impervious pad and distributing sulfuric acid solution on the heap to dissolve the copper by downward percolation. Dump leaching is widely used to extract copper slowly from porphyry mine strip wastes with dilute sulfuric acid-ferric sulfate solutions. Because the copper sulfide minerals, particularly chalcopyrite, in strip wastes dissolve very slowly, the dump is alternately leached and permitted to drain and oxidize, thus promoting faster extraction of the metal.

In general, the degree of extraction that can be attained is dependent on the physical and chemical nature of the raw material, the length and temperature of the leach, and the type and strength of solvent used. Agitation leaching, using dilute acid on finely ground amenable ore, usually will achieve extractions of over 95%, whereas vat and heap leaching of coarser ore requires a much longer leach period to attain extraction of no more than 80%. Similarly, use of less potent alkaline solvents requires prolonged leaching, a higher concentration of solvent, and/or leaching at higher temperature to effect extraction of 90%.

The pregnant solution derived by leaching ores and concentrates must be separated from the residue, which subsequently is thoroughly washed to achieve adequate recovery of the metal-bearing solutions. Thickeners, filters and, less commonly, centrifuges are used. The amount of fresh wash water is carefully controlled to prevent excessive dilution of the pregnant solution. In processing finely ground ores containing clays or other slime-producing minerals, solution recovery and residue washing often are expensive and inefficient.

Because no solvent is completely selective, pregnant solutions contain impurities that must be removed to make a metal product of acceptable quality. Solution purification and product recovery usually require a combination of techniques. These may include evaporation or cooling to promote selective crystallization, use of chemicals to complex and selectively precipitate impurities, application of ion exchange and solvent extraction, or reduction of dissolved metal salts to metal powder using hydrogen or other reductants.

Except as noted, the end product of hydrometallurgical processes is a purified metal salt that commonly is converted to metal by electrolytic or pyrometallurgical processing. Many leach processes feature the recovery, regeneration and recycling of the solvent solution to reduce costs, to conserve water and reagents, and to minimize liquid waste disposal problems.

Electrometallurgy—Electrometallurgy comprises all metal recovery or processing methods that are based primarily on the use of electric current. These processes can be grouped into two main subdivisions, depending on how the electric current is used. Electrothermic processes use electric current as the source of heat, and electrolytic processes use electric current to transport metal ions from anodes and/or electrolytes for deposition on cathodes.

Most of the metal processing operations performed in fuel-fired furnaces also can be performed in electrically heated furnaces. In the United States, where fuel energy is much cheaper than electrical energy, electrothermic processes are limited to applications wherein the several advantages of electric heating outweigh the disadvantages of higher cost. Electric heating permits better control of higher temperatures, the nature of the furnace atmosphere, and the purity of the metal end product.

Electric furnace processes are widely used domestically to melt and refine steel and produce alloy steels; to melt iron, steel and nonferrous metals in foundry practice; and to smelt high-grade chromium, molybdenum, silicon, manganese, tungsten and vanadium ores and their concentrates in producing ferroalloys. A small part of the total zinc produced in the United States is derived by an electrothermic process involving carbon reduction in an electrically heated furnace.

Electrolytic processes are widely employed for recovering and producing pure metals from specially prepared electrolytes, and for refining the impure metals produced by pyrometallurgical processes. These processes fall into two groups: those using an aqueous electrolyte, and those using a fused salt electrolyte maintained at a high temperature. In both, the electrolyte contains a compound that dissociates into metallic ions that are positively charged and therefore move to, and are deposited on, the cathode. The composition of the electrolyte is maintained at near optimum level by using consumable anodes or by adding raw feed material to the electrolyte in proportion to the amount of metal deposited on the cathode. Because impurities build up during electrolysis, part or all of the electrolyte must be removed periodically from the cell and purified.

Fused salt electrolysis is the principal method for producing aluminum, magnesium, calcium, sodium and rare earth metals from refined salts and compounds of those metals. Aqueous electrolysis is employed to refine most of the copper recovered by smelting and to produce lead, gold, silver and several less common metals in exceptionally pure form. Electrolytic processes are used to win zinc, cadmium, nickel, cobalt and antimony from sulfate solutions obtained from leaching ore and concentrates of these metals.

27.7.6—AGGLOMERATION

Agglomeration is the act of forming masses or clusters from fine particles. Most mineral products are agglomerated by four basic processes: sintering, pelletizing, briquetting and nodulizing. Each process has certain advantages, and its application is dictated by these criteria. Sintering and pelletizing make the best feed products for blast-furnace smelting and therefore are predominantly used to agglomerate metallic ores and concentrates.

Briquetting has been employed on soft minerals where uniformity and appearance of the product are of prime importance. Briquetting also is used for agglomerating electric furnace raw materials. Nodulizing is best suited for making cement clinker, which must be pulverized after fusion for the production of portland cement.

Sintering—Sintering originated in the nonferrous metal industry to agglomerate and roast fine sulfide concentrates simultaneously prior to smelting. Subsequently, it was adopted by the ferrous metal industry to agglomerate blast furnace flue dusts, mill scale and iron ore fines. In essence, the sintering process is one in which a bed of small ore particles is bonded into a clinkerlike aggregate by high-temperature fuel combustion. The rate of burning is controlled by drawing or blowing air through the bed. Temperatures approaching 1,400°C in the narrow combustion zone that progresses slowly through the bed promote the physical and chemical changes necessary to form the sintered structure.

Generally, fine ores or concentrates receive considerable preparation prior to actual sintering. Mixing, moisture adjustment, fluffing, and even balling of the raw feed often is necessary. Raw material larger than ½ in. is undesirable. Best results are obtained at ⅛ in. or less. Excessive quantities of fine material must be avoided by balling to maintain bed permeability at an acceptable level. If the raw ore lacks sufficient fuel, 3 to 5% coke breeze is blended into the mix to provide the necessary heat. The moisture content of the feed is carefully controlled to give the desired bed permeability and the proper temperature profile. Normally, 8 to 14% water is best.

When smelting is to be done, fluxes frequently are blended with the ore prior to sintering. The hydrates and carbonates are decomposed, and the flux and gangue are partially fused during sintering, lessening the energy required for smelting.

Essentially, all sintering is performed on traveling grates of the Dwight-Lloyd type. Ignition is provided by burner hoods located over the grate, and combustion is completed as the material nears the end of the grate. Hot sinter cake is cooled, and may be screened.

Sintering in the iron ore industry reached a peak in the late 1950s when it was the principal form of agglomeration. Development of self-fluxing sinter and sized sinter resulted in major advances in blast furnace production. However, sinter decrepitates badly when shipped and so must be prepared near the blast furnace. Since most ore sources are remote from steel mills, there had been a trend away from sintering and towards pelletizing at the mine.

Sintering in the nonferrous metals industry is limited primarily to lead and zinc ores where it is used to desulfurize and agglomerate flotation concentrates. Because of the high fuel content (sulfur) in these concentrates, a high percentage of return sinter is blended with the new feed for temperature control. Several passes over the grate normally are required before a sinter sufficiently low in sulfur is produced. The sinters are then reduced in nonferrous blast furnaces, or, if only zinc is present, in retorts or electrothermic furnaces.

Pelletizing—Pelletizing is begun by forming green (wet) balls, then drying and indurating them to bond the particles into a cohesive unit. A balling unit (drum, disk or cone) is fed moist ore or concentrate, which aggregates into small spheres by rolling action. The small pellets grow as they roll over the unagglomerated material. By control of moisture and residence time, pellets can be produced in sizes from ⅛ to 1 in. in dia. Sizing, if required, is accomplished by screens in closed circuit with the balling unit.

Feed preparation for pelletizing is critical. The raw material should be at least 60% minus 325 mesh and have a size range from about 100 mesh down to about 5μ. Grinding or blending with other materials may be necessary to adjust the size distribution. A binder usually is incorporated into the mix to provide sufficient pellet strength for handling until final induration has been completed. Binders may be an organic product, such as starch, sugar or pitch, or an inorganic substance like bentonite, cement or sodium silicate. The moisture of the feed must be controlled for high production and good pellet quality. Thorough blending of the binder and feed ore is mandatory to insure correct ball growth and adequate pellet strength. Drying and induration impart sufficient strength and hardness to the green balls for subsequent handling, shipment or smelting. Drying is done at about 220°F. Induration temperatures may range from 300 to 2,450°F, depending on the material and binder being used. High-temperature induration processes usually incorporate heat exchangers to lessen costs.

Pelletizing is the key to commercial large-scale treatment of fine iron-ore concentrates, agglomerating them into rock-hard pellets which can withstand shipment to the steel mills. One-half-inch balls are made in rotating drums or disks capable of treating up to 80 tons of iron concentrate per hour. About 20 lb of bentonite per ton of concentrate is used as a binder. Drying and induration are carried out in traveling grates, grate-kiln combinations or shaft furnaces. Temperatures in the 2,450°F range bond the hematite grains, producing balls that will support a load of several hundred pounds without fracturing. Iron ore pellets are readily reduced and smelted to iron in the blast furnace. Their high porosity and uniformity makes them a better raw material than either sinter or lump ore.

Fluorspar flotation concentrates have been pelletized since 1964 for use in steelmaking. The sized free-flowing pellets are particularly well adapted for use in oxygen steelmaking plants where speed and ease of handling are advantageous. Pellets are formed on a balling disk using sodium silicate and bentonite as binders. Induration is accomplished by passing the pellets through an oven at 600°F.

Fertilizers are granulated in equipment similar to that used for iron ore, except that ammonia frequently is added to the balling machines. The granules are indurated in dryers at relatively low temperatures.

Because of an ever-decreasing supply of naturally occurring coal acceptably low in ash and sulfur, agglomeration is being used to reconstitute fine coal particles. Coal fines customarily have been sent to waste, but to avoid stream pollution and to conserve natural resources, much can be recovered and put to use. Minus 28-mesh coal, recovered by froth flotation, is balled into ½-in. pellets using a lignin-liquor binder. The pellets are dried, indurated and cooled as they are conveyed through a tunnel kiln. Coal pellets meeting the requirements of the metallurgical market have been made. When they were added to mixtures of high-volatile and low-volatile coking coals, in amounts up to 5.25% by weight, a quality coke was produced.

Briquetting—Briquetting differs from sintering and pelletizing in that high pressures are applied to the materials being agglomerated. Temperatures of operation range from ambient to the softening point of ores and concentrates. Lubricants (usually water or oil) and binders (pitch, starches, clays, etc.) frequently are used to improve briquetting conditions and produce strong acceptable agglomerates. Briquetting in the mineral industry has been successful in agglomerating coal, charcoal, potash and salt. These relatively soft materials do not cause excessive wear of the forming rolls. More recently, improved alloys and manufacturing techniques have made it possible to reduce roll wear to the extent it is now practical to briquet preheated ores. Briquetting may have a distinct advantage in the agglomeration of dusts and fumes which are too fine, or lack the proper size distribution, for pelletizing.

Bonding in briquets is accomplished in several ways. For coal or charcoal, pitch is added to glue the material together. For soluble substances like salt, a small addition of moisture takes some of the salt into solution while, upon drying and recrystallization, acts as a cement. For a material like iron ore, briquetting at high temperatures and pressures softens the grains and forms intergranular bonds similar to those formed in heat-hardened iron ore pellets.

Briquets are produced as flakes, pellets or pillows. Properly prepared, they are dust-free and uniform.

Nodulizing—Nodulizing is done in a direct-fired rotary kiln, with raw materials fed as a slurry or as a premixed, sometimes balled, solid. As the feed passes from the cool end of the kiln to the hot end, the material begins to soften. Rotation of the kiln causes the sticky material to roll into lumps that may range in size from several feet down to a fraction of an inch. Because the charge is sticky, there is a generous buildup of material on the sidewalls of the kiln, requiring a boring bar for removal. Nodulizing is particularly well adapted to cement manufacturing, where the charge must be fused for formation of hydrophytic compounds.

Phosphate rock presently is being nodulized with the fluxes required for electric furnace phosphorus production. The prefluxing and intimate mixing in the nodulizing kiln increase electric furnace production sufficiently to defray the added costs of nodulizing, crushing and screening. Iron ore concentrates have been nodulized, but the costs generally are higher than those of pelletizing, and the resulting clinker is inferior to pellets as a blast-furnace raw material.

27.8—DEWATERING–AND THERMAL DRYING

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27.8.1—MECHANICAL DEWATERING

Definitions—Mechanical dewatering is the removal of water by means of gravity and centrifugal forces through screens or by sedimentation. Sedimentation is used either for clarification or thickening. Thickening increases the concentration of solids of a slurry while clarification is designed to produce a solids-free slurry.

Screens—The capacity of dewatering screens is affected by material weight per cubic foot, hardness, particle shape, size consist, solid content of the feed and delivered moisture content of the product.

There are no formulas to determine screen dewatering capacity because of the large number of variables. The procedure in sizing a dewatering screen is to determine the variables and rely on laboratory tests and field data. Table 27-14 shows the comparison between centrifugal and vibrating screens. The amount

TABLE 27-14—Capacity Comparison of Centrifugal and Gravity-Type Vibrating Screen[119]

	Tph Through per Sq Ft	
U.S. Fine Series No.	Centrifugal Screen	Gravity-Type Vibrating Screens
8	1.10*	0.40
12	0.833*	0.322
16	0.622*	0.300
20	0.486*	0.276
30	0.417*	0.242
40	0.347*	0.197
60	0.165–0.20†	—
80	0.095–0.14†	—
100	0.076–0.12†	—

* Based on dry crushed feed weighing 100 lb per cu ft with 50 % oversize and 25 % less than half-size of screen aperture; 95 % efficiency for centrifugal screen; 90 % efficiency for vibrating screen; with square-opening screen cloth with over 45 % open area.

† 90 % efficiency for centrifugal screen with square-opening screen cloth with over 30 % open area.