Section 19

Solid-Solid Operations and Equipment

Kalanadh V. S. Sastry, Ph.D., *Professor, Department of Materials Science and Mineral Engineering, University of California, Berkeley, CA; Member, American Institute of Chemical Engineers, Society for Mining, Metallurgy and Exploration. (Section Editor)*

Harrison Cooper, Ph.D., *Harrison R. Cooper Systems, Inc., Salt Lake City, Utah. (Sampling of Solids and Slurries)*

Richard Hogg, Ph.D., *Professor, Department of Mineral Engineering, The Pennsylvania State University, University Park, PA. (Mixing)*

T. L. P. Jespen, M.S., Min. Proc., *Metallurgical Engineer, Basic, Inc., Gabbs, NV. (Dense Media Separation)*

Frank Knoll, M.S., Min. Proc., *President, Carpco, Inc., Jacksonville, FL. (Super Conducting Magnetic Separation and Electrostatic Separation)*

Bhupendra Parekh, Ph.D., *Associate Director, Center for Applied Energy Research, University of Kentucky, Lexington, KY. (Other Techniques)*

Raj K. Rajamani, Ph.D., *Professor, Department of Metallurgy and Metallurgical Engineering, University of Utah, Salt Lake City, Utah. (Wet Classification)*

Thomas Sorenson, M.B.A., Min. Eng., *President, Galigher Ash (Canada) Ltd. (Flotation)*

Ionel Wechsler, M.S., Min. and Met., *Vice President, Sala Magnetics, Inc., Cambridge, MA. (Magnetic Separation)*

INTRODUCTION

SAMPLING OF DRY SOLIDS AND SLURRIES OF SOLIDS

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INTRODUCTION

GENERAL REFERENCES: *Proceedings of XIX International Mineral Processing Congress,* SME, Littleton, CO, 1995. Gaudin, *Principles of Mineral Dressing,* McGraw-Hill, New York, 1939. Kelly and Spottiswood, *Introduction to Mineral Processing,* Wiley, 1980. Roberts, Stavenger, Bowdersox, Walton, and Mehta, *Chem. Eng.*, **78**(4), 89 (Feb. 15, 1971). Taggart, *Handbook of Mineral Dressing,* 2d ed., Wiley, New York, 1964. Weiss, *SME Mineral Processing Handbook,* SME, Littleton, CO, 1985.

Most of the process industries deal with solid-solid systems which belong to the class of particulate systems. Particulate systems are composed of discrete solids known as particles dispersed in a gaseous or liquid phase. Solids dispersed in liquids are known as slurry systems. Thus, the processing of particulate solids might be carried out in either dry or wet state. Processing of particulate solids involves basically two kinds of operations: *mixing* leading to the generation of a homogeneous product, and *separation* in order to produce valuable solid components and to discard undesired less valuable solids.

The control of processes involving the treatment of solids generally requires means for careful sampling and analysis of solids and slurries at various points in an operation. Unlike liquids, particulate solids are not homogeneous. The composition of individual particles will vary with particle size and particle density. It follows that care must be exercised to take a sample that represents the entire solids mixture at the point of interest in the process. If the solids are not sampled in a representative manner, process and product control will not be reliable. The first subsection presents various aspects of sampling of solids and slurries including the underlying theory and details of different sampling equipment and their selection.

Mixing of solids is an important unit operation in the production of solids with consistent properties. A number of properties of the solid particles influence the mixing process, the design, and selection of mixing equipment. The second subsection elaborates on the theory of mixing, types of mixing equipment, and their operation.

Various techniques are available to separate the different types of particles that may be present in a solid mixture. The choice depends on the physicochemical nature of the solids and on site-specific considerations (for example, wet versus dry methods). A key consideration is the extent of the "liberation" of the individual particles to be separated. Particles attached to each other obviously cannot be separated by direct mechanical means except after the attachment has been broken. In ore processing, the mineral values are generally liberated by size reduction (see Sec. 20). Rarely is liberation complete at any one size, and a physical-separation flow sheet will incorporate a sequence of operations that often are designed first to reject as much

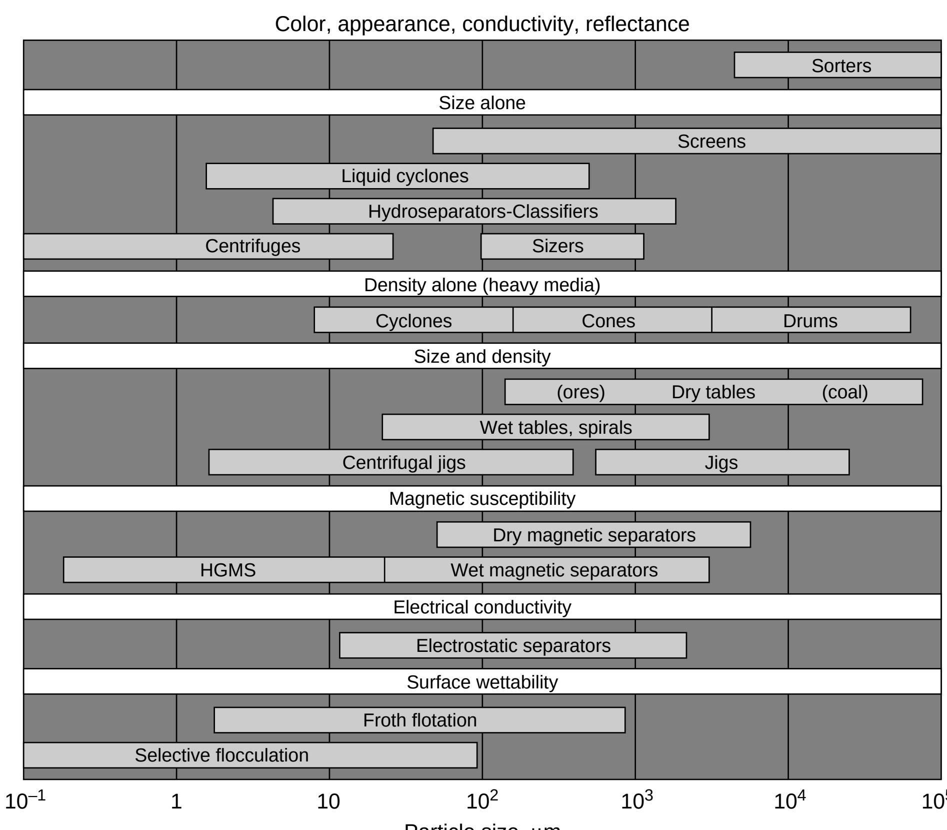

FIG. 19-1 Particle-size range as a guide to the range of applications of various solid-solid operations.

unwanted material as is possible at a coarse size and subsequently to recover the values after further size reduction.

Any difference in physical properties of the individual solids can be used as the basis for separation. Differences in density, size, shape, color, and electrical and magnetic properties are used in successful commercial separation processes. An important factor in determining the techniques that can be practically applied is the particle-size range of the mixture. A convenient guide to the application of different solid-solid separation techniques in relation to the particle-size range is presented in Fig. 19-1, which is a modification of an original illustration by Roberts et al.

The classification of solids by particle size is carried out for a number of reasons. Size classification can facilitate subsequent processing steps. An example is the scalping of tramp oversize material to avoid clogging a piece of processing apparatus. Similarly, better efficiency is achieved by removing fines before size reduction in crushers or ball or rod mills. Finished products generally are required to meet particle-size limits. Size separation is accomplished either in the dry condition or with the solids in suspension as a slurry. Wet classification allows higher process rates, particularly for materials of very fine sizes. Classification often is an integral part of a unit operation, as in closed-circuit grinding. Air classification methods for dry size classification in conjunction with size-reduction operations is covered in Sec. 20.

Gravity concentration is one of the oldest of the solids-separation techniques and the most important mineral-dressing method for obtaining ore concentrates. It is used mainly now for coal cleaning, yet Mills ["Process Design, Scale-Up and Plant Design for Gravity Concentration," in Mular and Bhappu (eds.), *Mineral Processing Plant Design,* 2d ed., Society of Mining Engineers, AIME, New York, 1980] notes that still more tonnage and greater values of material are concentrated by gravity methods than by a method such as froth flotation. The major unit operations which comprise gravity separation are jigging, tabling, spiral concentration, and dense-media separation. For high-capacity treatment of finer-sized low-grade ore materials, particularly the heavy mineral sands, the Reichert cone is becoming an industry standard [Ferree, "An Expanded Role in Minerals Processing Is Seen for the Reichert Cone," *Min. Eng.*, **25**(3), 29 (1973)].

Solids separation based on density loses its effectiveness as the particle size decreases. For particles below 100 microns, separation methods make use of differences in the magnetic susceptibility (magnetic separation), electrical conductivity (electrostatic separation), and in the surface wettability (flotation and selective flocculation). Treatment of ultrafine solids, say smaller than 10 microns can also be achieved by utilizing differences in dielectric and electrophoretic properties of the particles.

Physical separation methods are most widely used for the processing of coal and ore materials, and their basic development was designed for that purpose. Tremendous tonnages of solids are processed routinely at costs often as low at $1 per ton of material separated. The methods are applicable for other than ore processing, and solid-separation technology has become a more integral part of chemical-process operations. Recent requirements to recover values from various solid wastes have emphasized the need to adapt the relatively low-cost physical separation techniques of the ore processor, and as the needs to treat new types of materials and to improve recovery efficiency are constantly increasing, new designs are being developed.

The following subsections discuss the basic considerations involved in various unit operations of solid-solid separation and describe present industrial practice and equipment in general use.

SAMPLING OF DRY SOLIDS AND SLURRIES OF SOLIDS

References: Society of Mining Engineers, *Minerals Processing Handbook,* Norman L. Weiss, ed., chap. 30, "Sampling and Testing", part 2. "Theory and Practice of Incremental Sampling", Littleton, Colorado, 1985. Gy, Pierre M., *Sampling of Particulate Materials—Theory and Practice,* Elsevier Scientific Publishing Co., New York, 1979. Pitard, Francis F., *Pierre Gy's Sampling Theory and Sampling Practice,* CRC Press, Inc., Boca Raton, Florida, 1995. Gayle, G. B., *Theoretical Precision of Screen Analysis,* Report of Investigations No. 4993, U.S. Bureau of Mines, Dept. of Interior, Washington, D.C., 1952.

INTRODUCTION*

Sampling is a statistically derived process—a small amount of material *S* is taken from a large quantity *B* for the purpose of estimating properties of *B*. If *S* is an accurate sample (or stated more correctly, is representative of *B* according to a defined statistical parameter), it is a suitable estimator for the properties of *B*.

Sampling is typically required for quality control, wherein statistical data are compiled using specified procedures for mechanical sample collection and sample testing. Another sampling application is providing data for process control. A key factor in process-control sampling is minimizing time delays in making data available for use. Automatic analysis equipment is often employed, and the role of mechanical sampling becomes presenting samples for analysis by a reliable procedure.

The process of sample taking encompasses several steps, beginning with (1) taking a gross sample *S* from bulk materials *B;* (2) preparation of sample *S* for testing, which typically includes division of the sample and possible further substeps according to whether sampling is for analysis, size distribution, moisture, ash, and so on; and (3) the testing (analysis) step itself to determine properties of interest. Each step of the process contributes statistical error to the final result.

Estimations based on statistics can be made for total accuracy, precision, and reproducibility of results related to the sampling procedure being applied. Statistical error is expressed in terms of variance. Total sampling error is the sum of error variance from each step of the process. However, discussions herein will take into consideration only step (1)—mechanical extraction of samples. Mechanical-extraction accuracy is dependent on design reflecting mechanical and statistical factors in carrying out efficient and practical collection of representative samples *S* from a bulk quantity *B*.

Although mechanical sampling methods are to be the focus of attention, manual sampling methods are also employed for practical sample collection in commerce. Techniques of mechanical sampling should be emulated as closely as possible for best results with sampling by manual procedures.

Approved techniques for manual and mechanical sampling are often documented for various commodities handled in commerce by industry groups. Examples are the International Standards Organization (ISO), British Standards Association (BSA), Japan Institute of Standards (JIS), American Society for Testing Materials (ASTM), and the Fertilizer Institute. Sampling standards developed for use in specified industry applications frequently include instructions for laboratory work in sample preparation and analysis—steps (2) and (3) above.

Sampling techniques are more rigorous for materials with large variations in particle size and density compared to sampling of fine-sized powders. Coarse solids are often comprised of substantially differing mineral and crystalline forms within complex solids matrix. Fine-sized solid materials typically are relatively uniform in terms of chemical and physical characteristics with particle-size distributions and mineral densities usually within narrow ranges. Solids of organic chemical derivation and many commercial chemical materials, such as fertilizers, generally follow patterns of property distributions typical of powdered-mineral solids.

* Sampling of slurries and solids, differs fundamentally from sampling a completely mixed liquid or gas. A bulk quantity of solids incorporates characteristic heterogenity—that is, a sample S_1 differs inherently from a sample S_2 when both are taken from a thoroughly mixed load of solids as a result of property variances embodied in solids. In contrast, all individual samples from a completely mixed liquid or gas container are statistically identical.

The following discussion centers on sampling applications for powder solids comprised of small particulate sizes and equivalents in dry form or slurries. Sampling applications involving coarser solids (⅜ inch or 10 mm nominal size) as encountered in mineral products, typical ores, coal, and quarry rock for cement manufacture, are given more complete discussion in the *Mineral Processing Handbook* published by the Society of Mining Engineers and in other references (Pitard, Gy). "Nominal" particle size implies 95 percent through-screen particle size.

THEORY OF SAMPLING

Two principal topics are considered under theory of sampling. First is theory accounting for physical properties of material to be sampled. Second is the process of mechanical sample extraction. The theory predicts accuracy of sample taking—how much sample to take and how to take it to meet an accuracy specification.

Theory related to material characteristics states that a minimum quantity of sample is predicated as that amount required to achieve a specified limit of error in the sample-taking process. Theory of sampling in its application acknowledges sample preparation and testing as additional contributions to total error, but these error sources are placed outside consideration of sampling accuracy in theory of sample extraction.

Variations in measurable properties existing in the bulk material being sampled are the underlying basis for sampling theory. For samples that correctly lead to valid analysis results (of chemical composition, ash, or moisture as examples), a fundamental theory of sampling is applied. The fundamental theory as developed by Gy (see references) employs descriptive terms reflecting material properties to calculate a minimum quantity to achieve specified sampling error. Estimates of minimum quantity assumes completely mixed material. Each quantity of equal mass withdrawn provides equivalent representation of the bulk.

The theory enables a reasonable estimate of sample quantity needed to attain specified accuracy of a composition variable. The result is an ideal quantity—not realized in practice. Actual quantities for practical estimation are larger by an appropriate multiple to account for the reality that material is incompletely mixed when stored in stockpiles or carried on conveyors. Sample quantity to accommodate incompletely mixed solids can be specified through evaluating variance by autocorrelation of data derived with a series of stockpile samples, or from multiple sample extractions taken from a moving stream (Gy, Pitard).

In addition to composition factors, a sampling theory is available in sampling for size distribution. Quantity of sample needed to reach a specified error in determining size fraction retained on a designated screen is estimated by application of the binomial theorem (Gayle).

The second topic in theory of sampling pertains to mechanical sample taking. Design of mechanical sampling must conform to established criteria for sample-taking error to be minimal. This ensures error variance introduced by mechanical sample extraction is statistically insignificant compared to physical factors of sampling arising from heterogeneity, sample preparation, and sample testing sources of error.

Estimating Minimum Sample Quantity for Analysis The fundamental theory of sampling error variance can be applied to estimating a minimum quantity required from a completely mixed lot of solids for attaining an objective level of accuracy (Gy):

$$V = \left[\frac{1}{W_S} - \frac{1}{W_B}\right]\left[\frac{1-F}{F}\{(1-F)A_m + FA_g\}\right] fgbd^3$$

where V is the objective sampling-error variance (weight fraction), W_S is weight of the sample, W_B is weight of the bulk-solids lot, F is weight fraction mineral or other measurable quantity in the solids, A_m is density of mineral, and A_g is density of the nonmineral matrix.

Remaining terms to right of the bracket relate to properties to be measured within the matrix. The factor f is adjusted from 0 to 1 in relationship to the purpose of testing. A low value of f is indicated for scarce elements such as precious metals in electronic-source scrap. Moisture content has a high f value. The factor g is adjusted from 0 to 1 according to the degree of particulate classification. A high degree of size classification, as in a case of fine powders from screening, indicates values of 0.5 or higher. Unclassified fine solids from crushing have a value assigned to g of 0.25 or less. The factor b relates to size of elemental or crystal particles in bulk-solids particulate and degree of liberation ranging from 0 to 1. The term d is nominally the largest particle size. Estimated values employed in calculations rely on sampling experience and from solids-property investigation according to development of the theory, as described in related publications (Gy).

Example 1: Sample Quantity for Composition Quality Control Testing An example is sampling for quality control of a 1,000 metric ton (W_B) trainload of ⅜ in (9.4 mm) nominal top-size bentonite. The specification requires silica to be determined with an accuracy of plus or minus three percent for two standard errors (s.e.). With one s.e. of 1.5 percent, V is 0.000225 (one s.e. weight fraction of 0.015 squared). The problem to be solved is thus calculating weight of sample to determine silica with the specified error variance.

Bentonite has expected silica content of 0.5 weight percent (F is 0.005). Silica density (A_m) is 2.4 gm per cu cm, and bentonite (A_g) is 2.6. The calculation requires knowledge of mineral properties described by the factor ($fgbd^3$). Value of the factor can be established from fundamental data (Gy) or be derived from previous experience. In this example, data from testing a shipment of bentonite of 10 mesh top-size screen analysis determined value of the mineral factor to be 0.28. This value is scaled by the cube of diameter to ⅜-in screen size of the example shipment. The mineral factor is scaled from 0.28 to 52 by multiplying 0.28 with the ratio of cubed 9.4 mm (⅜-in screen top-size of the shipment to be tested) and cubed 1.65 mm (equivalent to 10 mesh).

Minimum weight W_S of sample is 110 kg from

$$0.000225 = \left[\frac{1}{W_S} - \frac{1}{10^6}\right]\left[\frac{1-0.005}{0.005}\{(1-0.005)2.6 + (0.005)2.4\}\right] 52$$

noting dimension of d^3 (particle diameter) is cubic mm requiring division by 1000 to rationalize with cubic cm of density. Sample weight in grams (from density) is divided by 1000 in converting to kg.

Estimating Change of Sampling Error with Change in Sample Size Increased accuracy in estimating a quality parameter by sampling through larger sample quantity can be estimated using the simplified Gy sampling equation

$$W_1V_1 = W_2V_2$$

W and V are values for sample weights and variances of parameter measurements at states 1 and 2 respectively.

Example 2: Calculation of Error with Doubled Sample Weight Repeated measurements from a lot of anhydrous alumina for loss on ignition established test standard error of 0.15 percent for sample weight of 500 grams, noting V is the square of s.e. Calculation of variance V and s.e. for a 1000 gram sample is

$$V = \frac{(0.15)^2(500)}{(1000)} = 0.01125\text{; standard error} = 0.11 \text{ percent}$$

Estimating Minimum Sample Quantity for Size Distribution Testing A simplistic approach to specifying minimum sample size for estimating particle distributions within allowed variance is based on a screening process in terms of binomial distribution. Each screening event is an outcome of two possibilities—particles either pass the screen or not. A relationship according to this principle presented by Gayle (loc. cit.) is employed in the example. Further development of sampling concepts for particle-size distribution is provided in the references (Pitard).

Example 3: Calculating Sample Weight for Screen-Size Measurement Weight W of bulk sample for screen analysis is calculated by the Gayle model for percent retained on a specified screen with relative standard error s.e. in percent

$$W = \frac{G(100-G)w}{V}; \quad \text{example } W = \frac{5.5(100-5.5)0.0120}{1.56} = 4.0$$

where G is the weight percent of the sample retained on the given screen either as determined by testing or defined per specification, and w is the weight of a particle of the size retained on that screen.

Sample weight estimated in this example is for two standard errors of 2.5 percent (resulting in V of 1.56) for testing iron ore (hematite) retained on a ½-in screen. Estimate of G is 5.5 for 94.5 percent of weight passing. Particle weight *w* retained on a ½-in opening screen assuming spherical shape and 5.1 specific gravity is 0.0120 lb. The calculation yields 4.0 lb as a minimum sample size, *W*.

Estimating Minimum Sample Quantity for Moisture Measurement Estimates of material quantity for testing moisture content depend on mechanisms of moisture distribution in the material. Moisture is physically retained on particle surfaces, chemically adsorbed on surfaces and within pores of particulate solids, and contained as an internal constituent of solids. Significant internal moisture is most often encountered in organic and agricultural source materials.

Sample quantity to estimate moisture for specific material is influenced to various levels of significance by properties such as particle-size range as well as relative amounts of moisture distributed among denoted forms of retention. Practical sample size estimates require background knowledge of parameters derived from experience for specific materials. More detailed examination of moisture-sampling aspects is provided in reference texts (Pitard).

Example 4: Calculation of Sample Weight for Surface Moisture Content An example is given with reference to material with minimal internal or pore-retained moisture such as mineral concentrates wherein physically adhering moisture is the sole consideration. With this simplification, a moisture coefficient K is employed as multiplier of nominal top-size particle size d taken to the third power to account for surface area. Adapting fundamental sampling theory to moisture sampling, variance is of a minimum sample quantity is expressed as

$$V = \left[\frac{1}{W_S}\right]\left[\frac{1-F}{F}\right]Kd^3;\ \text{example } 0.0000562 = \frac{0.95\ 5\ 0.00633}{W_S\ 0.05\ 1728}$$

where V is variance in weight fraction, W_S is minimum weight of sample, F is nominal weight fraction moisture, and K is a constant with dimension mass per unit volume. In absence of prior knowledge for material surface moisture characteristics, a value of K equal to 5 lb/ft^3 can be used for typical mineral concentrates and other nonabsorbing fine materials. This relationship is applied in an example of a crystalline product—hydrated sodium sulfate (Glaubers salt) with d of minus 4 mesh (0.185 in). Standard material moisture content is 5 percent by weight, with required sampling error of 1.5 percent relative to total weight for two s.e. Variance for this value in weight fraction is 0.0000562 in calculating 6.1 lb as sample weight (1728 converts in^3 to ft^3).

MECHANICAL DELIMITATIONS OF SAMPLING

Sample increment extraction requires a cutter to move through (traverse) a flowing stream being sampled while meeting accepted criteria of design and operation. Two methods of mechanical sampling for materials in flow regime are employed. A preferred first method is sample extraction from material in gravity free fall, such as from trajectory discharge at the head pulley of a conveyor or gravity flow down an enclosed chute. Cutter motion can be linear or rotational with constant speed while taking samples by traversing a gravity free-fall flow stream.

Sampling is required to meet the principle of mechanical sample extraction in maintaining statistical validity. This principle states that the cutter must take through-stream extractions during each traverse of the flow stream being sampled such that each particle in the flow stream at any place in the stream has equal probability of being extracted into sample. The diagram of Fig. 19-2 illustrates a typical arrangement meeting criteria (sampling delimitations) for a linear-traversing cutter installation extracting from a free-fall stream of material.

An alternative method is sampling directly from a moving or stationary conveyor with cutter traverse through the complete material bed carried on the conveyor. The alternative method cannot assure executing complete extractions, or through-stream sampling, because in many applications residual fines from the material stream remain on the conveyor surface.

The alternative method of sample extraction is termed the *cross-stream* sampling method, or *cross-belt* when used in conjunction with a belt conveyor. Sample extraction typically take place with a belt conveyor in motion. However, with a rotary table-feeder conveyor, extractions are made with the table stopped. A cutter can perform extractions by this means from a machined flat surface with negligible

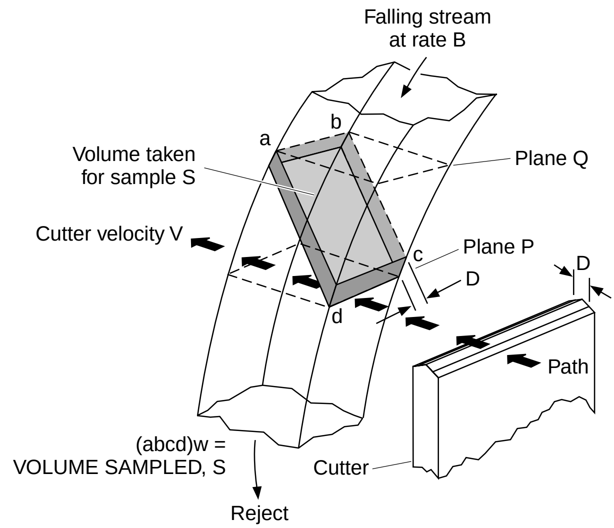

FIG. 19-2 Through-stream linear sampling. (*Courtesy of Harrison R. Cooper Systems, Inc. Salt Lake City, Utah.*)

residual fines left out of the sample. When sampling from a moving belt conveyor, residual fines become more significant resulting in loss of accuracy in extractions. This is due to clearances necessary between cutter edges and the conveyor belt, and also due to belt surface irregularities.

CRITERIA FOR SAMPLER DESIGN

Operation of a traversing sampler for *gravity flow* of material for through-stream sampling is required to meet the following design factors:

1. The cutter moves at constant speed (or constant rotation rate in the case of a rotary-motion sample cutter) such that the entire flow of material is traversed by the cutter, with the further requirement that the stopped position of the cutter at either limit of traverse (out of stream) is at sufficient distance from the stream so that no material from the stream enters the cutter while it is held stationary between traversing operations.
2. The sample cutter opening is set to specified width according to a multiple of the maximum (nominal) size of particulate being sampled and selected speed of the cutter. A minimum width of 10 mm or 0.375 in is recommended unless material is moist or has other properties to induce bridging of the cutter, suggesting need for a wider opening for practical operation. Experiments have determined that a cutter opening of a multiple of three times the nominal largest particle size and an 18-inches-per-second cutter speed (0.46 meters per second) is optimum to minimize sample extraction quantity with negligible delimitation error for fine-sized materials.
3. Cutter blade length extends beyond the material stream width on either side of the stream and volume of the cutter is sufficient to ensure all material taken into sample can be contained in the cutter body. Cutter blades are parallel, and are beveled to a sharp edge in the case of linear-motion traverse. For rotary-motion sample cutters, sharp edges of the cutter blades are radial to the center point of rotation.

Criteria for mechanical delimitations in sampling by the alternative cross-stream method to fulfill through-stream extraction requirements are revised from gravity-sampling criteria in the following respects:

1. The cutter opening is to exceed maximum (nominal) particle size by sufficient clearance to ensure that a large particle will not wedge into the opening. Sampling error due to free-fall deflection is avoided as a factor in setting cutter opening width. A 2 inch minimum cutter opening, required for practical operation, is recommended.

2. The cutter length should be approximately equal to the width of the material load carried on the conveyor.
3. When sampling from moving belt conveyors, the cutter operates in a radial mode with the belt surface contoured at the point of sampling by idlers, fixing radial curvature to the outer radius of the cutter. Clearance is minimized between outer edges of cutter blades and belt surface by cutter-shaft adjustment in the drive-clamping bracket.
4. Cutter speed at the outer radius is recommended at twice the conveyor belt speed for through-stream extractions from moving belts. The cutter is adjusted in a lateral angle to a 30-degree position, matching the cutter extraction path through the material bed on the belt at specified speed.

Cross-stream sampling from flat surfaces with material handled on a linear conveyor or rotary table is best carried out with the conveyor stopped. Sample extraction is then performed by linear traverse.

MECHANICAL SAMPLING EQUIPMENT

Repeating an axiom stated earlier, mechanical samplers are designed to extract increments of sample from a bulk quantity of material *B* in a manner that increments *S* are representative within statistical bounds of the bulk *B*. Further, the sampler is designed and constructed in conformance to criteria stated previously under "Mechanical Delimitations of Sampling" to assure that negligible errors arise from mechanical influence.

Many designs of equipment purported for sample extraction have been offered to industry or placed into service for sampling that fail to meet accepted mechanical standards. Extracted increments often have bias—inaccuracies found from tests on increments showing deviations usually with more or less fixed offset from true median values, or otherwise producing inconsistent and statistically poor test data compared to true values. Extraction increments using nonconforming designs may best be regarded as specimens of bulk *B*, but not samples in the statistical sense.

Mechanical sampling procedures further discussed are limited to sampling of flowing materials. Dry solid flows carried on a conveyor or in chute gravity fall are subject to through-stream sampling designed to extract correctly defined increments. Slurry gravity flows in launders and sloped pipes are sampled at the point of discharge, or slurries are sampled at open discharge from a vertical gravity pipe.

Static sampling methods with mechanical systems to operate thief-pipe sampling of solids taking increments from railroad hopper cars, trucks, or bins are seen in use. These are considered manual sampling methods operated mechanically. Applying criteria of through-stream sample extraction is infeasible, and it is inherently understood that bulk materials to be sampled in this manner are not perfectly mixed. An assured mode of sampling is providing through-stream sample extraction of bulk materials as they are loaded into bins, rail cars, trucks, and so on.

Various static thief or pipe samplers, often including pumps for stream transfers, are employed in slurry flows as well. These lack validity in terms of through-stream extraction capability. A pressure-thief sampler mounted on a pump discharge flange can be an approximation to through-stream sampling with assumption of complete mixing in flow from the pump if time lapse for flow to the thief from a pump is minimal, and pipe bends or other elements inducing classification are absent.

SELECTING A SAMPLER

Mechanical samplers meeting delimitation criteria are available in two basic designs for sampling material in gravity free fall. The basic designs are sampling with linear cutter motion and sampling with radial cutter motion (see Fig. 19-3 and Fig. 19-4 respectively). The net result is the same with either when equipment is properly designed and operated.

Selection of linear or radial (rotary cutter) sampling is made according to mechanical installation factors often on a basis of flow quantity. Smaller flows can be sampled in a cost-effective manner by rotary cutter samplers (frequently termed "vezin" design samplers, see Fig. 19-4).

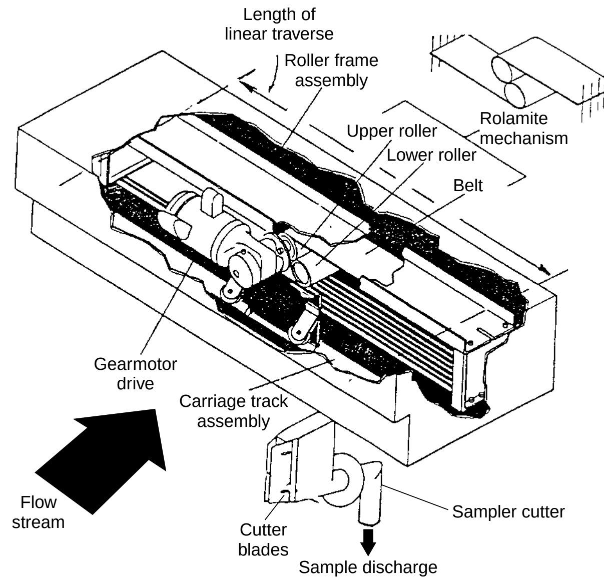

FIG. 19-3 Traversing sampler. (*Courtesy of Harrison R. Cooper Systems, Inc. Salt Lake City, Utah.*)

Sampling directly from material lying on the conveyor using a cross-stream cutter for extracting sample increments is diagrammed in Fig. 19-5 for moving conveyor belts and in Fig. 19-6 for a rotary table application. Cross-stream sampling can frequently be applied with acceptable delimitation error to materials of relatively low particle size and minimal variation, and also to materials with moisture content sufficient to avoid fines classification onto conveyor surfaces. A brush fixed to the cutter trailing edge aids in fines extraction to minimize residual sample remaining on the belt surface following cutter traverse.

In Fig. 19-5, the conveyor belt is radially profiled at the point of sample extraction with contouring idlers set to match the path of the cutter moving from its driveshaft rotation axis. Cutter edges are posi-

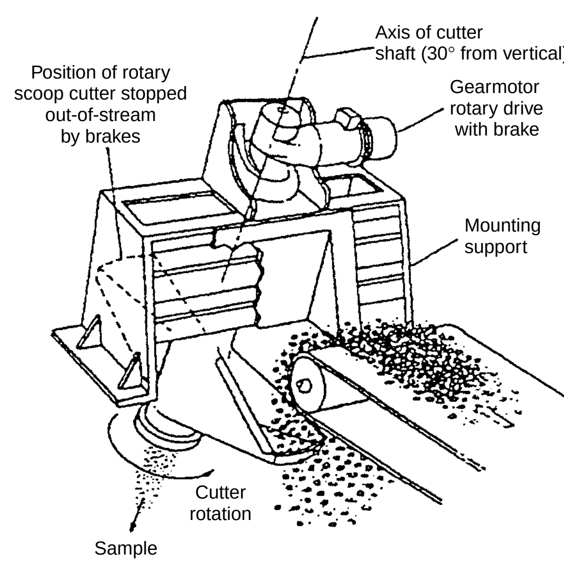

FIG. 19-4 Rotary sampler. (*Courtesy of Harrison R. Cooper Systems, Inc. Salt Lake City, Utah.*)

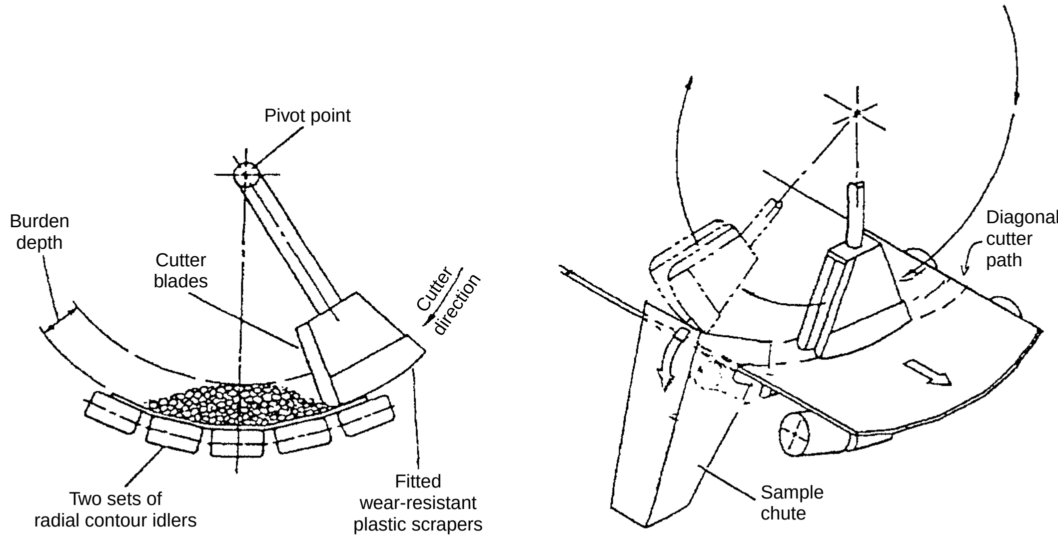

FIG. 19-5 Cross-belt sampler. (*Courtesy of Harrison R. Cooper Systems, Inc. Salt Lake City, Utah.*)

tioned with minimum clearance from the belt surface as is reasonably accomplished without contact of the cutter with the belt surface. Cutter blades are angled 30 degrees from the conveyor belt direction in positioning the cutter to its path through the conveyor belt load for cutter speed twice conveyor speed.

Extractions performed with the conveyor stopped allow more assured accuracy by the certainty of including fines in the sample increment. Sampler design to extract increments from a flat belt or rotary table sampler while the conveyor is stopped minimizes potential for residual fine particles remaining on the conveyor surface in carrying out extractions. See Fig. 19-6 for rotary table sampler extraction diagram.

Composite Samples Obtained by Multiple Sample Extractions Material flow streams are sampled in practice by combining extractions taken at successive time intervals into a composite sample. Multiple increment collection to obtain representative composite samples for specified bulk-material flows is performed according to a designated process in accommodating the presence of material property variations.

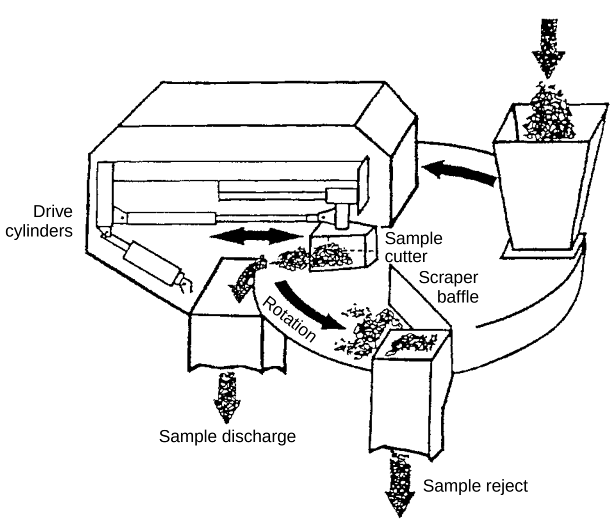

FIG. 19-6 Rotary-table sampler. (*Courtesy of Harrison R. Cooper Systems, Inc. Salt Lake City, Utah.*)

The requirement is to obtain proportional samples from the flowing material. This is accomplished in a technically accurate procedure by extractions taken on fixed time intervals. Variable time intervals with intervals determined from random selection are optionally employed to avoid bias error in sampling when characteristic periodic effects are known to be present in the stream of material. Possibilities for fixed sampling intervals to systematically coincide with periodicities are avoided by random time interval selection. Setting sampling intervals to material flow quantity, as in using belt weigh scale readings, opens potential for nonproportionalities and error in the composite sample.

Sampling of specified material flows to obtain representative composite quantities is a common practice for material accounting and quality control. A typical case is composite sampling of a shipload or trainload cargo transfer for either receiving or delivering materials. Another frequently used specification is eight-hour shift production quantities to be sampled to generate composite samples for testing.

Industry standards are frequently applicable in designing sampling procedures for many commodities in commerce transferred by ship cargoes or trainloads. Standards for iron ore, coal, metallurgical concentrates, and similar materials are often to be observed. Standards are likely to give details on sampling specifications necessary for acceptance-based material characteristics and lot size to mandate minimum number and weights of increments, gross (combined) sample weights, and other factors.

Selection of appropriate time intervals for increment extractions relates to property variation (inhomogeneity) within material flow streams. Ten minute extraction intervals are generally adequate to obtain suitably representative samples from material flows under practical circumstances. Precise determination of extraction intervals consistent with individual applications can be calculated through autocorrelation of historical sampling data, a statistical method described in references (Gy, Pitard).

Sample Quantity Reduction As sample increments are accumulated by multiple extractions from a bulk flow of material, according to the parameters of sampling to accommodate material stratification and nonhomogeneous composition, gross sample quantities (primary sample) often become quite extensive. Large primary sample volumes are subject to mechanical resampling to obtain final samples of practical, reduced quantities for testing. The same principles of sampling applied to primary sampling are used to design resampling to accomplish sample reduction without loss of sample statistical validity.

Sample reduction in successive stages—primary to secondary, secondary to tertiary, etc.—can be fulfilled using automatic sampling equipment while observing design principles of statistical sampling. Alternatively, sample quantity reduction may be carried out in a laboratory.

Sample reduction by mechanical procedures in automatic on-line mode encompasses (1) particle-size reduction preceding a following stage of resampling, and (2) multiple secondary increments taken for each primary increment when resampling without particle-size reduction. Particle-size reduction implies crushing or grinding the sample before resampling. A sampling-unit design incorporating primary and successive stages of sampling, with particle-size reduction and controlled flow of sample through intermediate stages, is developed in accord with application requirements while maintaining specified standards of sample accuracy.

Calculation of Sample Extraction Increments Sample quantities taken in an extraction increment are calculated in accord with the mechanical sampler employed. The following three examples illustrate calculations for three commonly used sampling methods.

Example 5: Solids Sampling by Linear Traversing Trajectory Cutter Increment weight S by a linear traversing cutter from bulk material flow of fine powder B expressed in unit weight per unit time is calculated by

$$S = \frac{B \times D}{V} \quad \text{example } S = 1.38 = \frac{120 \times 2000 \times 0.375}{3600 \times 18}$$

where V is cutter velocity and D is cutter opening. For S given in 120 short tons per hour converted to lbs, 0.375-in cutter opening, and 18-inches-per-second cutter speed, each increment is 1.38 lb. For consistent units, tons per hour is multiplied by 2,000 for lbs per hour, and divided by 3,600 for lbs per second.

Example 6: Slurry Sampling by Rotary Traverse of Gravity Flow Increment volume, quantity of slurry extracted by one cutter rotation, is S from bulk slurry flow B expressed in volume-per-unit time. R is cutter rotation per minute. D is cutter angle opening, with $D/360$ extraction ratio for continuous cutter rotation.

$$S = \frac{D \times B}{360 \times R} \quad \text{example } S = 0.055 = \frac{2.5 \times 200}{360 \times 25}$$

with S gallons per extraction for 200 gallons per minute, 2.5 degree cutter opening, and 25 RPM cutter rotation rate.

Example 7: Cross-Belt Sampling of Solids from Conveyors Increment weight is S from bulk material flow B expressed in unit weight per unit time. J is belt speed in length-per-unit time. $J \times 2$ is cutter speed. Therefore, cutter angle determining path length through material loads on the conveyor belt is 30 deg for traversing perpendicular to direction of conveyor movement. Extraction weight is corrected by csc(30 deg) to account for diagonal path of the cutter. Solids weight-per-unit length of conveyor belt is B/J. With cutter width D of 50 mm, minimum recommended for fine powders, increment weight S is

$$S = \frac{B \times D \times 1.16}{J} \quad \text{example } S = 1.93 = \frac{120 \times 1000 \times 50 \times 1.16}{3600 \times 1 \times 1000}$$

for S given in kg at 120 metric tons per hour and 1 meter per second conveyor belt speed. Consistent units require tons per hour be multiplied by 1,000 for kg per hour, and divided by 3,600 for kg per second. Cutter opening is divided by 1,000 for meters.

Sampling Trajectory Stream from Conveyor-Belt Discharge Conveyor-belt speeds above approximately 300 ft per minute (1.5 meters per second) impart sufficient momentum to material discharging at its head pulley to cause lifting of material streams in a trajectory from the head pulley. A trajectory is illustrated in Fig. 19-7. Blades of the sample cutter are positioned to intersect the trajectory. See Fig. 19-7 for an example of a linear-traversing bottom-dump cutter installation. Calculation of trajectory profiles are described in the Conveyor Equipment Manufacturers Association publications and similar references.

SAMPLING EQUIPMENT COST DATA

The cost of an electric-drive rotary-cutter sample of the smallest size manufactured—suitable for gravity sampling of fine particulate solids or slurry flow—including timer and control unit was approximately $5,000 in 1996.

An electric-drive linear-traversing sampler of minimum standard manufactured size with cutter and controls will range upwards of $8,000.

Pneumatic as well as electric-drive samplers are available. Generally, pneumatic-drive samplers are lower in cost.

Cross-belt samplers of minimum size for 24-in (600-mm) conveyors cost approximately $15,000 with controls using an electric drive, and about $12,500 with pneumatic drive.

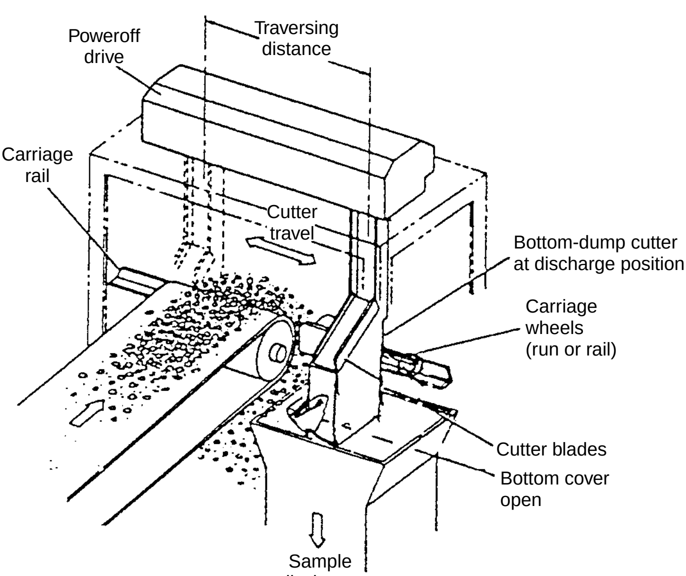

FIG. 19-7 Traversing linear bottom-dump sampler. (*Courtesy of Harrison R. Cooper Systems, Inc. Salt Lake City, Utah.*)

Hydraulic-drive samplers are also available, but cost factors tend to be substantially greater than electromechanical units. Recent use of hydraulic-drive systems has diminished with the availability of increased strength and durability electric-motor linear-drive units capable of reliable operation in high-capacity applications.

Sampling systems for multiple-stage sample reduction incorporating components such as crushing units, interstage feeders, reject handling, and others range up to several hundred thousand dollars in cost. A requirement would be rarely encountered in fine-powder applications.

SOLID-SOLID SYSTEMS

MIXING

General References

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A comprehensive bibliography is available in Ref. 9. A more recent update can be found in Ref. 18. Equipment photographs and details are available in Refs. 2, 4, 5, 7, 8, and 10. References 3 and 6 give excellent theoretical work. Reference 5 gives a tabulation and summary of many mixer types and applications. References 8 and 9 are book chapters dealing with mixing of solids and cover both the theoretical and the equipment aspects. Interpretive summaries of the literature in various areas (state of mixedness, theoretical frequency distributions, rate equations, and equipment) are included in Refs. 9 and 18. Reference 1 gives a procedure for testing solids-mixing equipment.

Fundamentals

Objectives Equipment in which solid materials are mixed may be used for a number of operations. Blending of ingredients may be the main objective, as, for example, in the preparation of feeds, insecticides, fertilizer, glass batches, packaged foods, and cosmetics. Other objectives may include cooling or heating such as in the cooling of limestone or sugar or the preheating of plastic prior to calendering. Drying or roasting of the solids is sometimes desired. In some applications, such as polymerization of plastics, catalyst manufacture, or the preparation of cereal products, the solids mixture may be reacted. Coating is desired in some cases, as in the manufacture of pigments, dyes, minerals, candy, and other food products and in the preparation of feeds. In certain of these cases, small amounts of liquid may be added, but the end product is a solids mixture. Sometimes agglomerates are desired, as in the preparation of food products, pharmaceuticals, detergents, and fertilizer. Often size reduction is desired while solids are being mixed. In all cases, the mixing of solids occurs. However, in some of these operations, the details of the equipment to accomplish operations other than pure blending may become a major problem. This portion of Sec. 19 will deal with equipment whose major function is to give a thorough mixture of solids. Specialized equipment to perform the other functions is discussed in other sections of the *Handbook* and will not be dealt with here. Thus, for example, Sec. 8 is devoted to size reduction and enlargement, although equipment mentioned there may also accomplish mixing.

Properties Affecting Solids Mixing Wide differences among properties such as particle-size distribution, density, shape, and surface characteristics (such as electrostatic charge) may make blending very difficult. In fact, the properties of the ingredients dominate the mixing operation. The most commonly observed characteristics of solids are as follows:

1. *Particle-size distribution.* This tells the percentages of the material in different size ranges.
2. *Bulk density.* This is the weight per unit of volume of a quantity of solid particles, usually expressed in kilograms per cubic meter (pounds per cubic foot). It is not a constant and can be decreased by aeration and increased by vibration or mechanical packing.
3. *True density.* The true density of the solid material is usually expressed in kilograms per cubic meter (pounds per cubic foot). This, divided by the density of water, equals specific gravity.
4. *Particle shape.* Some types are pellets, egg shapes, blocks, spheres, flakes, chips, rods, filaments, crystals, or irregular shapes.
5. *Surface characteristics.* These include surface area and tendency to hold a static charge.
6. *Flow characteristics.* Angle of repose and flowability are measurable characteristics for which standard tests are available (e.g., ASTM Test B213-48, Flow Rate of Metal Powders, etc.). A steeper angle of repose would indicate less flowability. The term "lubricity" has sometimes been used for solid particles to correspond roughly to viscosity of a fluid.
7. *Friability.* (Also see "Grindability," Sec. 8.) This is the tendency of the material to break into smaller sizes in the course of handling. There are quantitative tests specially devised for certain materials such as coal which can be used to estimate this property. Abrasiveness of one ingredient upon another should also be considered.
8. *State of agglomeration.* This refers to whether the particles exist independently or adhere to one another in clusters. The kind and degree of energy employed during mixing and the friability of the agglomerates will affect the extent of agglomerate breakdown and particle dispersion.
9. *Moisture or liquid content of solids.* Often a small amount of liquid is added for dust reduction or special requirements (such as oils for cosmetics). The resultant material may still have the appearance of a dry solid rather than a paste.
10. *Density, viscosity, and surface tension.* These are properties at operating temperature of any liquid added.
11. *Temperature limitations of ingredients.* Any unusual effects due to temperature changes which might occur (such as heat of reaction) should be noted.

A look at these properties for the ingredients to be mixed is a first step toward selecting mixing equipment.

Measuring Uniformity Except for cases in which a coating of one ingredient with another takes place, the theoretical end result of mixing will not be an arrangement in which one type of particle is directly next to a different type. Rather, the theoretical end result when random tumbling takes place will be a random mixture along the lines shown in Fig. 19-8.

The variation among spot samples of known size can be predicted theoretically for a random mixture and used as a guide to determine how closely random blending of the ingredients has been approached. Various types of analyses can be made on spot samples to determine batch uniformity. These could include x-ray fluorescence, flame spectrometry, polarography, emission spectroscopy, and so on, depending on the powder being examined. Radio-tracing techniques may also be appropriate. As many spot samples as possible should be analyzed. These should be taken at random from different locations in the batch. Sample size is an important consideration and is discussed below.

Evaluation Statistical tests can be used to evaluate relative homogeneity based on observed variations in spot sample composition. For a simple binary mixture such as that shown in Fig. 19-8, it can be shown (see Ref. 9) that the expected variance among samples containing n particles each is given by

$$\sigma^2 = \frac{p(1-p)}{n} \tag{19-1}$$

where p is the overall fraction of black (or white) particles in the mixture. The observed sample variance can be computed using

$$S^2 = \frac{1}{m-1}\left[\sum_{i=1}^{m} p_i^2 - \frac{1}{m}\left(\sum_{i=1}^{m} p_i\right)^2\right] \tag{19-2}$$

where p_i is the fraction of black (or white) in the i^{th} sample and m is the total number of samples taken. The expected and observed variances can be compared using the statistical F-test (see Sec. 2 or any standard reference on statistics) which determines the likelihood that the F-ratio (S^2/σ^2) could be obtained from a random mixture, purely by chance.

The procedure can be readily extended to multicomponent systems by applying the test to each component in turn. In real systems, it is generally convenient to take samples of fixed volume or mass rather than fixed number of particles. In such cases, the expected variance can be computed using (see Refs. 19 and 20)

$$\sigma^2 = \frac{f_{ij}(1-f_{ij})w_{ij} + f_{ij}^2(\overline{w} - w_{ij})}{M} \tag{19-3}$$

where f_{ij} is the overall mass fraction of size i composition j material in the mixture, M is the sample mass, w_{ij} is the mass of a single particle of size i composition j and $\overline{w}$ is the mean particle mass:

$$\overline{w} = \sum_i \sum_j f_{ij} w_{ij} \tag{19-4}$$

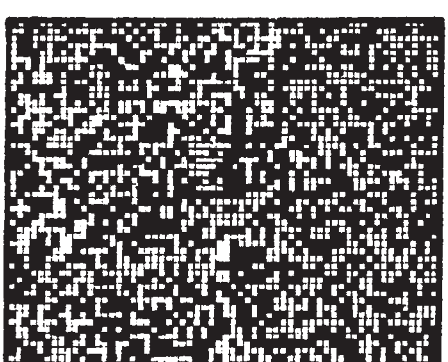


FIG. 19-8 Random arrangement of black and white particles. [*Lacey,* Trans. Inst. Chem. Eng. (London), ***21,*** 52 (*1943*).]

The test for homogeneity is based on the probability of including different kinds of particles in a sample. For large samples, containing many particles, the expected variance given by Eq. (19-3) becomes extremely small and will often be exceeded by the variance due to experimental (analytical) error. The approach described above is, therefore, appropriate only for evaluating homogeneity at a scale approaching the size of the individual particles. If information at that scale is needed, it is necessary to use extremely small samples, containing no more than some hundreds of particles each. For very fine powders, this may seriously limit the choice of analytical techniques.

The use of very small samples to evaluate fine-scale homogeneity will often tend to mask long-range but small variations in composition. The use of somewhat larger samples is appropriate for detecting and quantifying such variations. In such cases, the sample variance can be compared, using the F-test, with an experimental variance S_E^2 obtained from replicate testing of the analytical procedure used to determine sample composition.

In general, a two-level procedure is recommended in which very small samples are used to evaluate microhomogeneity at the individual particle scale and larger samples are employed to investigate longer range variability. The actual sample sizes should be chosen such that microhomogeneity is evaluated from samples for which σ^2, as calculated using Eq. (19-3), is substantially less than the experimental (analytical) variance S_E^2 while macrohomogeneity is tested using samples with $\sigma^2 >> S_E^2$.

Whether the desired end product is satisfactory can also be used as a practical criterion of the adequacy of the solids mixture. A further consideration is the effect of the solids mixture on the overall economics of the manufacturing process. Studies of the type mentioned in the preceding subsection *may* be part of such an evaluation. When the solids mixture is made directly into a product, as in the case of feed pellets or pharmaceutical tablets, uniformity tests on these items will speak for themselves. If the solids mixture must be further processed, as in the manufacture of glass or plastics, the efficiency and costs of the subsequent operations can often be related to the starting solids mixture. In such cases, knowledge of the homogeneity of the solids mixture is needed to determine its effect on the manufacturing process.

Regardless of the method of evaluating the solids mixture, the sampling procedure is vital. Often a sampling thief, or other special device, is used to remove samples from the mixture without excessive disturbance of the batch. If an easier method of sampling is obvious and will bring less contamination to the batch, it should be used.

Method of sampling, location, size and number of samples, method of sample analysis, and fraction of the batch removed for sampling all contribute to how well the sampling study reflects the actual conditions.

A standard testing procedure for solids-mixing equipment is available (Ref. 1). This contains details and references pertaining to sampling from solids mixtures for both batch and continuous mixing.

Segregation Problems Previously it was pointed out that wide differences among properties may make blending very difficult. For example, natural segregating tendencies will be observed with extreme differences in specific gravity, size, or shape. The heavier, smaller, or smoother and rounder particles tend to sink through the lighter, larger, or jagged ones respectively. In some cases, preparation of the materials to avoid extreme differences in such ingredient properties can avoid segregation problems.

There are also other factors which can cause segregation.

Electrostatic charges may cause particles to repel each other. When continued blending may cause such charges to build up, it is important to determine the precise blending time required and not to overblend.

Loss of material as dust must be considered as a possible means of segregation and should not be aggravated by too strong suction in the dust-collection apparatus.

If there are smeary particles which have an almost pastelike behavior and barely flow (high angle of repose), frictional anchorage of these onto the other particles in the mixture may be necessary in order to achieve good mixing.

If a batch ingredient is in agglomerate form, some device to break up the agglomerates should be used to prevent them from segregating from the rest of the mixture and to ensure the intimate dispersion of this ingredient throughout the mixture.

The use of a liquid such as water (possibly with a surface-active agent) can have remarkable effects in overcoming segregation which may appear inevitable otherwise.

Although these statements apply to the actual solids-mixing operation, thought must also be given to the subsequent processing steps. Thus, the solids-mixing operation must be checked from the point of view of delivering a well-mixed batch to a certain point. The system must be scrutinized for possible segregating points such as transfer points, long drops, flow through silos, and vibratory equipment. Where a liquid is used, the amount that can be added without getting into caking problems which may upset the later processing of the solids mixture should be determined.

Equipment

Mixing Mechanisms There are several basic mechanisms by which solid particles are mixed. These include small-scale random motion (diffusion), large-scale random motion (convection), and shear.

Motions which increase the mobility of the individual particles will promote diffusive mixing. If there are no opposing segregating effects, this diffusive mixing will in time lead to a high degree of homogeneity. Diffusive mixing occurs when particles are distributed over a freshly developed surface and when individual particles are given increased internal mobility. A plain tumbler gives the former, while an impact mill gives the latter.

For most rapid mixing, in addition to diffusive (fine-scale) mixing, there should be a means by which large groups of particles are intermixed. This can be accomplished by either the convective or the shear mechanism. A ribbon mixer illustrates the former, whereas a plain tumbler gives the latter.

The diffusion mechanism occurs readily for free-flowing powders in which individual particles are highly mobile, but is inhibited by cohesion among particles. It follows that cohesive powders, containing fine material or liquid phases, are relatively difficult to mix. At the same time, reduced particle mobility inhibits demixing so that once mixed, cohesive powders tend to remain so. Free-flowing powders, on the other hand are prone to demixing during any transport/handling operation. The beneficial effects, noted above, of liquid addition presumably result from increased cohesion.

Types of Solids-Mixing Machines There are several types of solids-mixing machines. In some machines the container moves. In others a device rotates within a stationary container. In some cases, a combination of rotating container and rotating internal device is used. Sometimes baffles or blades are present in the mixer. Most types can be quite effective for free-flowing powders, bearing in mind that segregation may also be favored. Highly cohesive powders generally require high shear (velocity gradient) to achieve a high degree of microhomogeneity. Table 19-1 classifies solids-mixing machines via the characteristics given in the column headings. Illustrations of several of the machines listed there are shown in Fig. 19-9. The various types listed in Table 19-1 will be briefly discussed, with paragraph numbers referring to the columns.

1. *Tumbler.* Suitable for gentle blending; capable of handling large volumes; easily cleaned; suitable for dense powders and abrasive materials. Not for breaking up agglomerates.

Figure 19-9*a* and *b* (without broken-line portions) shows some unbaffled tumblers.

Figure 19-9*c* and *d* shows some baffled tumblers.

2. *Tumbler with agglomerate breaker.* See Sec. 20: "Tumbling Mills," for ball mill, rod mill, and vibratory pebble mill which will accomplish mixing along with size reduction.

Several tumblers are available with separately driven internal rotating devices for breaking up agglomerates. The tumbler itself can be used for gentle blending if agglomerate breakdown is not required.

The broken-line portions of Fig. 19-9*a* and *b* show some types of agglomerate-breaking devices for tumblers.

Table 19-2 includes impact velocities for some internal rotating devices in tumblers as well as other mixers. Contamination and wear problems of internal rotating devices are discussed under "Performance Characteristics."

3. *Stationary shell or trough.* There are a number of different types of mixers in which the container is stationary and material displacement is accomplished by single or multiple rotating inner mixing devices.

a. Ribbon mixer (Fig. 19-9*e*). Within this subgroup there are several types. Ribbon cross section and pitch, clearances between outer ribbon and shell, and number of spirals on the ribbon are some features which can be varied to accommodate materials ranging from low-density finely divided materials that aerate rapidly to fibrous or sticky materials that require positive discharge aid. Other construction variations are center or end discharge and the mounting of paddles or cutting blades on the center shaft. A broad ribbon can be used for lifting as well as for conveying, while a narrow one will cut through the material while conveying. The ribbon is adaptable to batch or continuous mixing.

b. Vertical screw mixer. This subgroup also has several variations. One type is shown in Fig. 19-9*f*. In this type, the screw rotates

TABLE 19-1 Types of Solids-Mixing Machines*

Tumbler (1)	Tumbler with internal agglomerate breaker (2)	Stationary shell or trough (3)	Both shell and internal device rotate (4)	Impact mixing (5)	Process steps which can affect solids mixing‡ (6)
Without baffles:					
Drum, either horizontal or inclined	Ball mill	Ribbon	Countercurrent, muller turret and pan rotate in opposite directions	Hammer mill	Filling of hoppers
	Pebble mill				
Double cone	Rod mill	Stationary pan, rotating muller turret†		Impact mill	Fluidization
Twin shell	Vibratory pebble mill	Vertical screw	Planetary types	Cage mill	Screw feeders
Cube					
Mushroom type	Double cone	Single rotor		Jet mill	Conveyor-belt loading
	Twin shell	Twin rotor		Attrition mill	Elevator loading
	Cube				
		Turbine			Pneumatic conveying
		Paddle mixer			Vibrating
With baffles:					
Horizontal drum		Sifter (turbosifter)			
Double cone revolving around long axis					

*Diagrammatic sketches of many of these machines are shown in Fig. 19-9.
†There is also a muller in which the turret is stationary but the pan rotates.
‡Although these steps, when carefully selected, can aid mixing, caution must be exercised with pneumatic conveying and vibrating, as they may tend to separate materials.

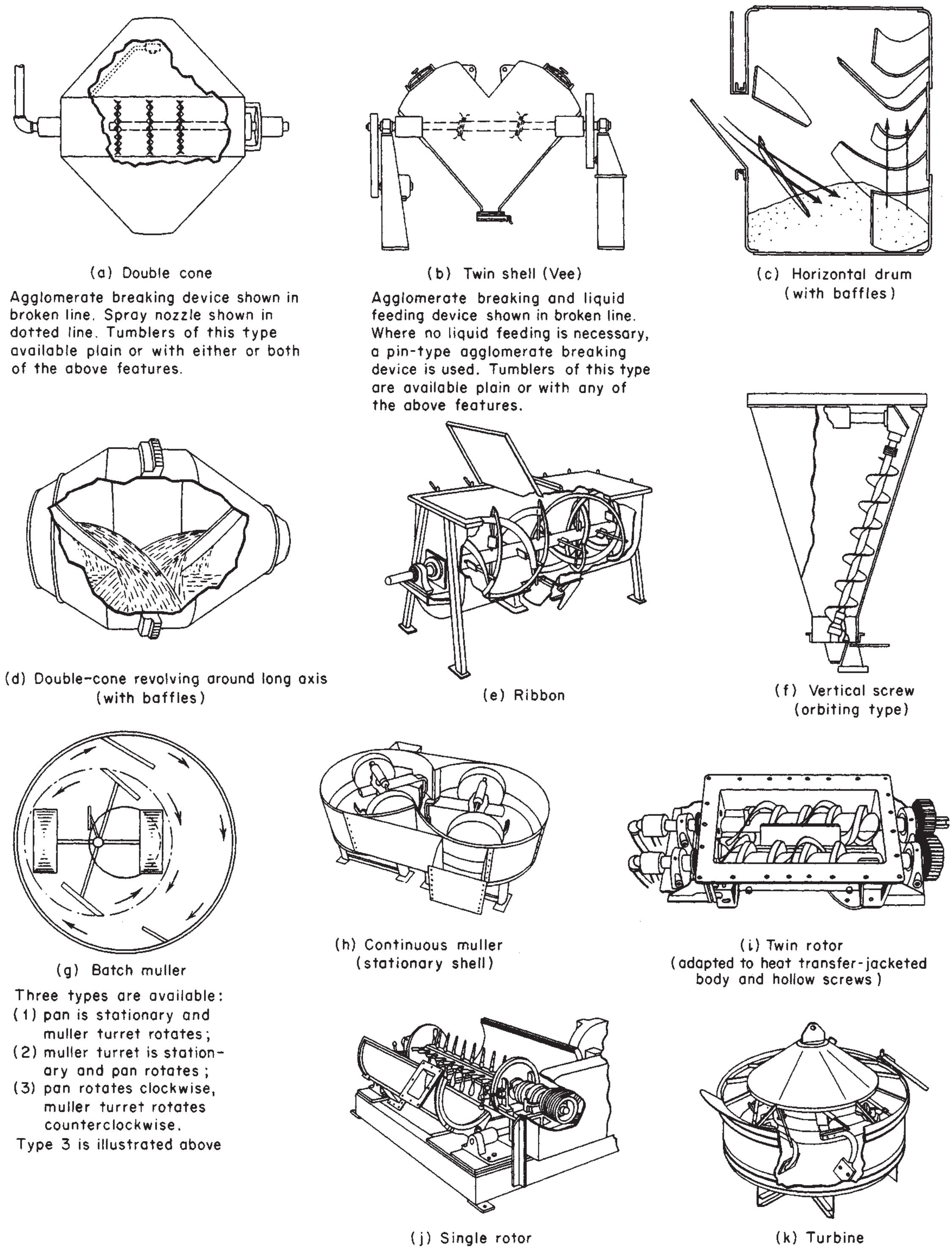


FIG. 19-9 Several types of solids-mixing machines. (See Table 19-1).

TABLE 19-2 Approximate Impact Velocities of Some Rotating Internal Devices in Mixers*

Type of mixer (see Table 19-1)	Tip speed, ft/min
Ribbon	280
Turbine	600
Twin-shell tumbler with	
Pin-type intensifier	1700
Liquid-feed bar	3300
Twin rotor	Up to 1300
Single rotor	6000–9000
Mills of various types	2500–20,000

*To convert feet per minute to meters per second, multiply by 0.00508.

about its own axis while also orbiting around the center axis of the conical tank. In another variation, the screw does not orbit but remains in the center of the conical tank and is tapered so that the swept area steadily increases with increasing height. In another type, the central screw is contained in an inner cylindrical casing. This type of mixer is primarily suitable for free-flowing dry solids.

c. Muller mixer. The stationary-pan muller with rotating turret is one of several types. Other muller types are the countercurrent type, in which the pan and muller turret rotate in opposite directions, and the rotating-pan type, in which the muller turret is stationary.

The heavy, wide roller rides over the material. There is some skidding action where the rollers engage the mass of materials. This gives local shearing plus coarse-scale mixing which is aided by the plows and scrapers.

The muller is useful for mixing problems requiring certain types of aggregate breakdown, frictional anchorage of particles to one another, and densification of the final mix. Materials which are excessively fluid or sticky should be avoided. The muller mixer is generally used for batch operations (Fig. 19-9*g*), although Fig. 19-9*h* shows a continuous muller.

d. Twin rotor (Fig. 19-9*i*). This consists of two shafts with either paddles or screws encased in a cylindrical shell. There are various types available with shaft speeds ranging from moderately low to relatively high (see Table 19-2). The twin rotor is useful for continuously mixing non-free-flowing solids; liquids can be added, there is minor product attrition, and materials can be added beyond the inlet. It is easily adaptable to heating or cooling. Some machines are specifically designed for heat transfer during mixing. The pug mill is one type of twin rotor.

e. Single rotor (Fig. 19-9*j*). This consists of a single shaft with paddles encased in a cylindrical shell. This type is available with relatively high speeds (see Table 19-2), although in certain cases lower speeds are used. A high-speed single rotor gives the maximum impact short of a grinding mill. It is used for intensive dispersion and disintegration. The type is available with split casing and is suitable for heating or cooling and for small amounts of liquid addition.

f. Turbine mixer (Fig. 19-9*k*). This is a circular trough with a housing in the center around which revolves a spider or a series of legs with plowshares or moldboards on each leg. The moldboards spin around through the circular trough. This mixer is suitable for free-flowing dry materials or semiwet materials which do not flow well and is also adaptable to liquid-solid mixing and coating problems.

4. *Shell and internal device rotate.* The countercurrent muller (Fig. 19-9*g*), which is in this category, is mentioned under "Muller mixer." This machine has a clockwise rotating mixing pan with a counterclockwise rotating mixing tool head mounted off center of the pan, thus providing a planetary mixing pattern. For the mixing of free-flowing solids not requiring the shearing and compressive action of mullers, plows are sometimes used alone. When used with mullers, plows deflect material into their path. Special mixing tools are also available.

5. *Impact mixing.* This process, which includes size reduction, is covered in Sec. 20.

The process steps listed in Table 19-1 can sometimes be used to promote mixing. However, they are primarily for functions other than solids mixing. (Note precautions for pneumatic conveying and vibrating in Table 19-1.)

Since paste mixing is not within the scope of this section, such widely used paste mixers as the sigma blade and banbury types will not be covered here but instead are discussed in Sec. 18.

Performance Characteristics Before selecting solids-mixing equipment, a careful study should be made of various performance characteristics. These are given here.

Uniformity of Mixture The proper type of mixer should be chosen to assure the desired degree of batch homogeneity. This cannot be compromised for other conveniences. Information is given under "Types of Solids-Mixing Machines" about the special abilities of various kinds of machines to blend different types of materials.

Care should be taken to avoid mixing too long, as in some cases this will result in a poorer blend. A graph of degree of mixing versus time should be made to select the proper mixing time quantitatively.

Mixing Time The actual time during which the batch is being mixed is usually less than 15 min if the proper type of machine and working capacity have been chosen. In some cases much more lengthy mixing times are tolerated so as to avoid the cost of purchasing more efficient equipment. However, there is usually a machine that can properly homogenize almost any type of mixture in less than 15 min provided one is willing to pay the price. In fact, proper mixer design in most instances will produce the desired blend in a few minutes.

Besides actual mixing time, however, the total cycle time should be optimized.

Charging and Discharging The total handling system must be considered in order to obtain optimum charging and discharging conditions. This includes the efficient use of weigh hoppers and surge bins, minor-ingredient premixing, location of discharge gates, and so on.

Power In general, power requirements are not a major consideration in choosing a solids mixer since other requirements usually predominate. However, sufficient power must be supplied to handle the maximum needs should there be changes during the mixing operation. Also, when a variety of mixes may be required, power must be sufficient for the heaviest bulk-density materials. If the loaded mixer is to be started from rest, there should be sufficient power for this. When speed variation may be desirable, this should be taken into account in planning power requirements.

Horsepower requirements of several types of mixers are listed in Table 19-3.

Cleaning The ease, frequency, and thoroughness of cleaning may be crucial considerations when incompatible batches are to be mixed at different times in the same machine. Plain tumbling vessels are easy to clean provided that adequate openings are available. Areas that may present cleaning problems are (1) seals or stuffing boxes, (2) crevices at baffle supports, (3) any corners, and (4) discharge arrangement. If cleaning between different batches may be time-consuming, several small mixers should be considered. Special sanitary construction can usually be provided at extra expense.

Agglomerate Breakdown and Attrition The two methods of producing agglomerate breakdown and attrition are as follows:

1. *Impact.* The major factor is the peripheral speed of the rotating internal device. Table 19-2 gives impact-velocity data for various mixers.

2. *Shearing and compressive action.* In mullers this depends upon the clearance between muller and pan and the muller weight or spring load respectively.

When an attrition device is necessary to break down aggregates but may also produce too much size reduction on other batch ingredients, tolerable attrition should be determined by tests.

Dust Formation Loss of dust can seriously affect batch composition, particularly when vital minor ingredients are lost. Methods of minimizing dust formation are: (1) Use of less dusty but equally satisfactory batch ingredients. Sometimes a pelletized form of an extremely dusty material is available. (2) Proper venting so as to enable filtering of displaced air rather than unregulated loss of dust-laden air. (3) Dust-tight arrangements for loading and unloading the mixer. (4) Addition of liquids if tolerable. Not only is water effective in minimizing dust upon discharging from the mixer, but if properly added it will also render the batch less dusty in subsequent handling steps. The addition of a small quantity of surface-active agent will improve the penetration of the water throughout the batch and enable

TABLE 19-3 Horsepower Requirements and Speeds of Rotation for Some Commercial Solids Mixers
[Approximately 1.5 m^3 (50 ft^3) Working Capacity]

Type of solids-mixing machine	Approximate working capacity, ft^3	Horsepower, hp		Rotational speed, r/min		Comments
		Shell	Internal device	Shell	Internal-device shaft speed	
1. Tumbler						
Without baffles						
Double cone	54	7½		18		Based on 100-lb/ft^3 material.
Twin shell	50	5		13.7		Maximum bulk density of material = 55 lb/ft^3.
With baffles						
Horizontal drum						
Manufacturer E	50	20		11.1		Heavy-duty (material 100 lb/ft^3). For extremely heavy duty (150–200-lb/ft^3 material), the maximum working capacity with 20-hp motor is 35 ft^3.
Manufacturer F	50	10		14		For material of 40-lb/ft^3 maximum bulk density.
Double cone revolving about horizontal axis	56	25		11.5		Mixer can be tilted. Rear end charger. Capacity based on mixed concrete.
2. Tumbler with agglomerate breaker						
Double cone	54	7½	See Comments.	18	See Comments.	Horsepower requirement for internal device depends on character of material, type, and speed of agitator. These are to be determined by adequate testing.
Twin shell	50	5	5 (pin-type intensifier bar) 7½ (liquid-solids intensifier bar)	13.7	945 (1730-ft/min tip speed) 1055 (3320-ft/min tip speed)	Maximum bulk density of material = 55 lb/ft^3.
3. Stationary shell or trough						
Ribbon						
Manufacturer C	50		12		28	Horsepower required based on material of 50–60-lb/ft^3 bulk density, medium free-flowing, using 10 hp/ton for average mix cycle of 3–10 min (depending on material, range can be 3–18 hp/ton).
Manufacturer A	46		10		37	Based on material of 30-lb/ft^3 bulk density.
Manufacturer D	50		15		45	Based on material of 40–50-lb/ft^3 bulk density.
Three-shaft ribbon	50		Blender shaft 20 Feeder shaft 7½ (total)		Variable-speed drives on all shafts	This blender is rated at 300 ft^3/h on batch-mixing basis; 900 ft^3/h on continuous-mixing basis. Materials rated at 70-lb/ft^3 bulk density.
Vertical screw	52.9		5		Screw, 64.4 Orbit, 2.2	Horsepower based on 37-lb/ft^3 bulk density. This may vary with different materials. Maximum hp = 10, maximum weight = 4410 lb.
Muller:						
Batch; stationary pan, rotating turret	40		60		24 (turret speed)	Based on material of 60–75-lb/ft^3 bulk density.
Continuous; stationary pan, rotating turret	Basically, the continuous mullers are merely two-batch mullers joined together at the cribs, making a figure-8 design. Thus, the 40-ft^3 batch muller rated at 60 hp becomes an 80-ft^3 working-capacity continuous muller requiring 125 hp. This would give 125 tons/h with a 2½-min residence time. Turret speeds are 24 r/min.					
Single rotor		See Comments.				In this *continuous* unit, output can range from 25–600 lb/min with hp from 5 to 100 and r/min of 500 to 4000, depending on the materials mixed.
Double rotor		See Comments.				In this *continuous* unit the output can range from 200–500 lb/min with hp from 5 to 40 and r/min from 200 to 300, depending on the materials mixed.
Twin-rotor heat-exchanger mixer	49.2		5–15		20–100	Amount of conveying and mixing action affected by amount of pitch and type of ribbons mounted on exterior of hollow screws.
Turbine	50		50		Peripheral speed of 600 ft/min	
4. Both shell and internal device rotate						
Countercurrent muller	45 60–90†	* 20	* 25	6.75–8.75 6.65	28–35 20	

*One 25-hp motor drives both the shell (mixing pan) and the internal device (mixing star).
†Batch-capacity range depends on nature of materials to be mixed.
NOTE: To convert cubic feet to cubic meters, multiply by 0.02832; to convert horsepower to kilowatts, multiply by 0.7457; to convert pounds per cubic foot to kilograms per cubic meter, multiply by 16.02; to convert tons per hour to kilograms per second, multiply by 0.252; to convert revolutions per minute to radians per second, multiply by 0.1047; to convert pounds per minute to kilograms per minute, multiply by 0.4535; and to convert horsepower per ton to kilowatts per metric ton, multiply by 0.8352.

it to wet even such materials as coal dust. The method of adding water is important (see "Method of Adding Liquids").

Care should be taken to avoid powerful suction or air flow on the mixer or the weigh hopper from which the ingredients feed into the mixer. If the dust-collection suction on the mixer is too strong, vital ingredients may be sucked out. If the dust-collection suction on the weighing system is too strong, errors in weighing may result.

Electrostatic Charge Certain batch materials such as plastics tend to accumulate a charge easily. Work input will affect the charge on the batch. Coating of the inside of the mixer shell or rotating elements may occasionally result because of electrostatic charge. This can present a cleaning problem. Possible aids in overcoming this are (1) addition of special solid materials with very high surface area to weight ratios, (2) addition of liquids (see "Dust Formation" and "Method of Adding Liquids"), (3) proper choice of material of construction of the mixer, (4) controlling humidity, (5) preparation of the batch ingredients so as to minimize accumulated charge.

Equipment Wear Simple tumbling mixers give the least wear. Attrition devices in tumblers may present serious abrasion problems with certain materials such as sand and abrasive grinding-wheel grains. Abrasion-resistant coating such as rubber coating, special alloys, or platings should be considered for these cases. An internal agitator device may wear even though its speed is low. Particularly when highly abrasive materials are to be mixed, the benefits of an agglomerate-breaking device must be weighed against potential contamination and replacement and maintenance costs.

Contamination of Product This has been partially covered under "Cleaning" and "Equipment Wear." Other sources of contamination are lubricants and repair materials. Types which are not compatible with the batches to be mixed should be avoided.

Heating or Cooling Nearly all commercial mixers can be heated or cooled. Some can be provided with heated or cooled agitators. If temperature rise during mixing is detrimental, cooling facilities should be provided. The various manufacturers can provide details on the means of heating their machines. Most common heating means are (1) water or steam in the jacket and in hollow-screw or paddle-type internal agitator, (2) hot oil, (3) Dowtherm liquid or vapor, (4) electric heaters, contact or radiant, (5) hot air in direct contact with product (suitable only for revolving-drum-type mixers), (6) exterior heating of drum by direct or indirect firing. For cooling, the most common means are (1) water or refrigerated fluid in the jacket and in hollow-screw or paddle-type internal agitator, (2) an evaporant such as liquid ammonia, (3) direct air contact (for rotating-shell mixers), and (4) oil or Dowtherm (or its equivalent) for cooling high-temperature materials.

Flexibility When batches of widely different size must be mixed, flexibility of operating capacity may enable use of fewer mixers. Certain features may necessitate a nonflexible capacity requirement. For example, ordinarily an internal agitating device in a tumbling mixer does not function effectively unless the batch is loaded to a certain level. The need for such features must be weighed against the limitations imposed by a narrow operating-capacity range when choosing equipment for an operation in which batch size will vary considerably.

In general, the effect of percentage of mixer volume occupied by the batch on the adequacy of mixing should be borne in mind, particularly when any change from the recommended volume percent is considered.

Vacuum or Pressure Most tumbling mixers can have provision for vacuum or pressure. Mixers which cannot be adapted to these conditions are mullers with rotating pans. Continuous mixers introduce problems of sealing the charge and discharge ends.

Method of Adding Liquids When the addition of liquids may be desirable (see "Dust Formation" and "Electrostatic Charge"), this should be considered when designing the mixing system rather than hastily improvised. The purpose of the liquid should be considered, whether for (1) dust suppression, (2) product, or (3) heating and cooling. If a viscous liquid must be well distributed, this requirement should be considered when choosing the mixer.

Liquid should be directed into the batch materials and not onto bare mixer surface since this could cause buildup. Nozzle spray pressure should be sufficient to penetrate the batch but not so high as to cause heavy splashing. The liquid should be added to the well-mixed batch. In particular, when premature addition of liquid could impair the adequacy of blending, both the time during which it is added in the mixing cycle and the time taken to add the liquid are important.

Automated equipment for the addition of liquids can be worked into the overall mixing plant when necessary. For dust-reduction purposes, a volumetric method of metering is satisfactory. However, should a critical batch ingredient be added in liquid form, a more precise method of metering may be necessary.

Other considerations are (1) proper ventilation and discharge enclosures, (2) provision for relief of internal explosion, (3) vibration isolation (shock mounts), (4) remote operation of charge and discharge, (5) noise during operation.

Equipment Selection Types of mixers and performance characteristics have been given. Segregating tendencies among solid materials have also been described. A sound approach to solids-mixer selection starts with a careful examination of these areas. However, mixer selection should also involve consideration of the mixer's place in the overall process. Possible consolidation of many solids-processing steps or the opposite (splitting one operation into several) deserves scrutiny at this time. If no one standard machine has all the necessary requirements, thought should be given to which machine can best be modified to achieve the most desirable combination of features. One should look at the overall process objectives as well as at equipment details when selecting a solids mixer.

Pilot Tests In some cases, it is possible to perform pilot tests on a small-scale version of the equipment to be used in production. Much useful information can be found here but the following must be borne in mind:

1. In general, the larger the pilot unit, the more reliable the prediction of large-scale performance. The pilot unit should be a prototype with all dimensions properly scaled down.
2. Published solids-mixing scale-up data are rare. Equipment suppliers can provide scale-up information for their particular types of equipment on the basis of experience. With geometrically similar tumblers, if the speeds are adjusted to give comparable motion and the mixer volume fraction occupied by the charge is the same, scale-up of results will be straightforward. The presence of a rotating internal device presents problems in the scaling up of clearances, blade area to mixture volume, and sizes and speeds of the rotating devices. For agglomerate breakers, the key factor in scaling up is impact velocity. Scale-up in cylinders is discussed on pages 290–292 of Ref. 9. Solids-processing scale-up is discussed in a paper by Sterret (*Chem. Eng.*, Sept. 21, 1959).
3. The actual process materials should be used if possible. If substitute materials must be used, they should have the same mixing characteristics. Tests with differently colored but otherwise identical beads can be misleading, and so can tracers. The reason is that the flow properties of the specific materials to be mixed in the plant may not be the same as these demonstration materials. Regardless of how the mixer contents appear to be moved around, the properties of the actual batch ingredients may cause segregation or other problems.
4. Differences in materials of construction between the pilot unit and the production unit should be considered. These may have a bearing on caking, abrasion, and electrostatic effects.

Continuous Mixing Although batch mixing has been the predominant method of mixing solids, consideration is being given to the use of continuous mixing in many industries. There are two types of continuous-mixing operations. The first type has a low holdup volume and will provide fine-scale blending of the particles via impact and shear elements such as are used in grinding machines. Some machines of this type are hammer, impact, cage, and jet mills. It is essential that the feed to these machines be properly proportioned and premixed to achieve a uniform product.

The second type of continuous mixer involves high holdup machines which contain agitating and conveying mechanisms. These rearrange the individual particles and also displace large volumes of material and move the batch through the machine. Mixers of this type can produce both fine-scale and coarse-scale blending. The ribbon-type mixer is frequently used for continuous mixing, although this is also used for batch mixing. A continuous muller mixer has been developed as shown in Fig. 19-9*h*.

The average composition of the stream leaving a continuous mixer is the same as the average of the added entering streams. Variations in proportions of the entering streams will be damped out by the mixing action of a continuous mixer. These effluent-stream variations will become smaller as average solids residence time is increased and the frequency of the variations increases.

Certain general criteria can be used to determine whether continuous flow will be beneficial. Continuous flow is worth consideration if (1) a single formulation can be run for an extended period, (2) the fluctuations of the outgoing product are within process requirements, (3) sufficiently accurate metering of ingredients can be achieved, (4) the rest of the process warrants continuous mixing. Continuous flow is of doubtful benefit if (1) frequent changes of formulations are anticipated, (2) fluctuations of product composition will be outside the permitted range, (3) the ingredients cannot be metered with the necessary level of accuracy, (4) complex temperature or pressure cycles are involved.

Sometimes a system of mixing and dispersing is composed of one or more batch units providing a feed to a continuous intensive dispersion unit. Another possibility would be a batch mixer and surge bin which provide a continuous feed to a final dispersion unit. Various combinations of this type with adequate sampling at the proper points may be used when continuous flow would be beneficial provided that certain features could be overcome.

OPTICAL SORTING

Difference in optical properties can be used as the basis to separate solids in a mixture. Optical properties include color, light reflectance, opacity, and fluorescence excited by ultraviolet rays or x-rays. Differences in electrical conductance can also be used for separation. With appropriate sensing, the particles in a moving stream can be sorted by using an air jet or other means to deflect certain particles away from the mainstream (Fig. 19-10). The lower limit of particle size is about 0.003 m (⅛ in); below this limit the process rate would be slow and equipment costs become exceedingly high.

In a typical **optical sorting** installation, the mixture of particles is fed from a hopper onto a vibrating feeder. Dust may be removed by dry screening or by water spray. The solids then enter a troughed conveyor belt and align the flow and cause the particles to be projected in a continuous stream along their free-fall trajectory. They are viewed in midair during fall through an optical chamber by a series of cameras arranged to view the entire surface of each particle. The color or reflectivity of the surface of each particle sets up a characteristic voltage pattern in the output circuit of a light-sensing photomultiplier. The patterns are analyzed electronically and compared with a preset reflectance level. When appropriate, a reject signal delayed electronically will activate on air jet to deflect a particle from the main stream.

An example of throughput is given in Fig. 19-11. For the machine referenced in Fig. 19-10, electronic sensing is capable of inspecting 80 particles per second.

Color sorting has been used for the recovery of glass from nontransparent materials in municipal solid waste. The separation of mixed glasses from opaques is illustrated in Figs. 19-12 and 19-13. As the throughput is increased, the glass recovery falls; and for any given installation there will be a breakeven point for the optimum number of machines against the amount of glass recovered. Color sorting can also separate mixed colored glasses to produce amber and green products that meet a specification of 10 percent contamination of one type of glass in the other.

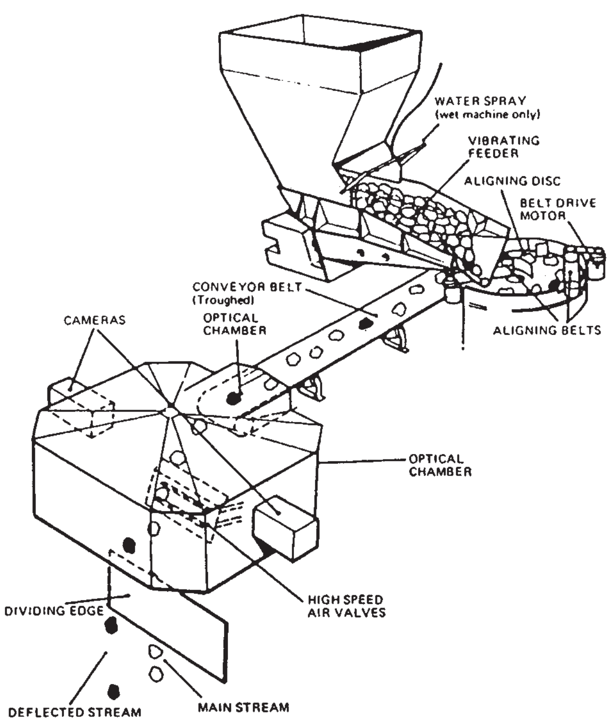


FIG. 19-10 Sortex 711M optical separator. (*Courtesy Gunson's Sortex Ltd.*)

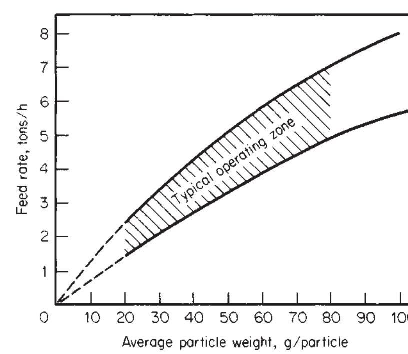


FIG. 19-11 Sortex 711M feed-rate characteristics. (*Courtesy Gunson's Sortex Ltd.*)

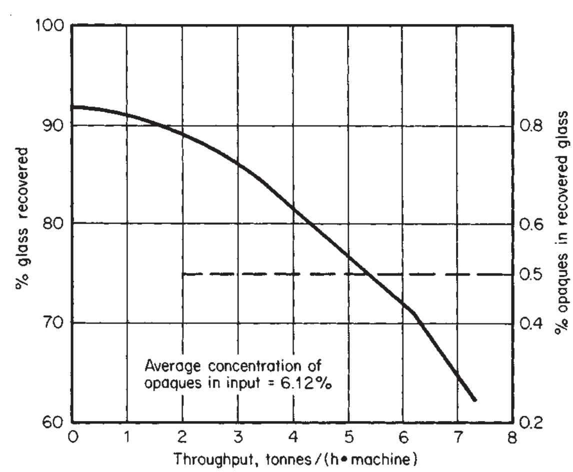


FIG. 19-12 Optical separation of mixed glasses: separation of opaques from glasses. (*Courtesy Gunson's Sortex Ltd.*)

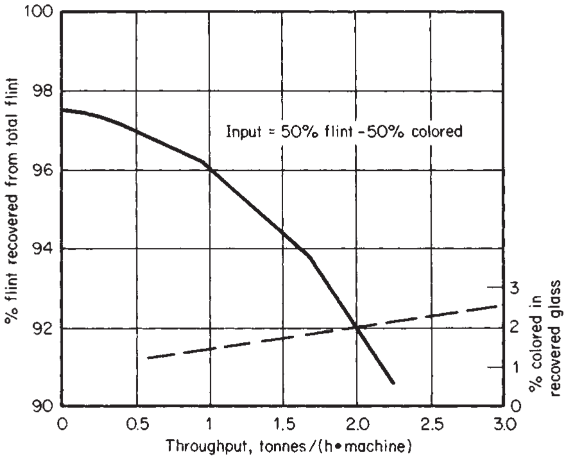


FIG. 19-13 Optical separation of mixed glasses: separation of flint from colored. (*Courtesy Gunson's Sortex Ltd*.)

When **electrical conductivity** is used as the basis of the sorting process, contact of the particles is made by a brush type of electrode to generate the signal for analysis. Materials having a resistance difference of 2000 kΩ can be readily separated from material of 100 kΩ resistance.

However, separation between resistance levels of 1000 to 300 kΩ may be marginal. A typical application of conductivity sorting is the separation of massive ilmenite from anorthosite. Both are compact rocks, but ilmenite is a good electrical conductor, whereas anorthosite is an insulator. The dimensions and operating information for the Sortex CS-03 conductivity sorter, which is capable of processing up to about 25,000 kg/h (27.5 tons/h) of 0.05- to 0.15-in mesh size (2 to 6 in), are given in Table 19-4.

The differences in absorptivity of radiant energy by different substances can also be used to separate materials. If a mixture of materials is exposed to radiant energy, for example, infrared, depending on the properties of the material involved, some particles will become heated more than others. The more opaque particles will be heated more by infrared heating than clear particles. Thus, the more opaque will become hotter. By spreading the irradiated material onto a surface coated with a low-melting thermoplastic or a heat-sensitive polymer, the higher-temperature particles will adhere, while the cooler particles will not. This is the basis for the **thermoadhesive-separation** method of Brison ("Separation of Materials," U.S. Patent 2,907,456, 1959) used by the International Salt Co. for removing impurities from mined rock salt. The heat-sensitive resin employed is a mixture of polymerized styrene resins, Piccolastic A-25 and Piccolastic A-50. The proportion of each was adjusted to give the required softening point to achieve the desired results. In practice, the resin can be continuously applied to a moving belt by brush or hot spray. Periodic scrapping and redressing are required to maintain belt performance.

Use of specific forms of radiant energy, infrared, ultraviolet, dielectric heating, etc., can allow specific separations to be made. The separation of clear and colored grains of glass and the separation of different metals are possible applications of the thermoadhesive method being considered in the field of solid-waste processing.

SCREENING

GENERAL REFERENCES: Beddow, "Dry Separation Techniques," *Chem. Eng.*, **88,** 70 (Aug. 10, 1981). Colman, "Selection Guidelines for Size and Type of Vibrating Screens in Ore Crushing Plants," in Mular and Bhappu (eds.), *Mineral Processing Plant Design,* 2d ed., Society of Mining Engineers, AIME, New York, 1980. Kuenhold, "Factors to Consider in Vibrating Screen Installations," *Min. Eng.*, 650–653 (June 1957). Matthews, *Chem. Eng.*, deskbook issue, **99** (Feb. 15, 1971). Matthews, "Screening," *Chem. Eng.,* **79,** 76 (July 10, 1972). Moir, "Recent Developments in Mineral Processing and Their Implications," *Economics of Mineral Engineering Mining Journal,* Books Ltd., London, 1976, p. 125. Mular, *Mineral Processing Equipment Costs and Preliminary Capital Cost Estimations,* spec. vol. 18, Canadian Institute of Mining and Metallurgy, Montreal, 1978. Pryor, *Mineral Processing,* 3d ed., Elsevier, New York, 1965. Reed, "The Story behind the New Sieve Specifications," *Test. World,* October 1959. Taggart, *Handbook of Mineral Dressing,* 2d ed., Wiley, New York, 1945.

TABLE 19-4 Specifications for Sortex Conductivity Sorter CS-03*

Dimensions	Height (excluding feeder), 1.7 m (70 in) Width, 1.9 m (76 in) Length, 2.5 m (100 in) Diameter of disk, 1.50 m (49 in)
Electric power	380–440 V, three-phase 50/60 Hz; consumption, 2 kW (excluding vibrating feeder)
Air consumption (depending on rejection rate)	7 m^3/min at 6 bar (250 ft^3/min at 80 lbf/in^2)
Water supply	4 L/min (1 gal/min) for cleansing the light source
Net weight	560 kg (22 cwt) approximately

*Courtesy of Gunson's Sortex Ltd.

Definitions

Screening Screening is the separation of a mixture of various sizes of grains into two or more portions by means of a screening surface, the screening surface acting as a multiple go-no-go gauge and the final portions consisting of grains of more uniform size than those of the original mixture.

Material that remains on a given screening surface is the oversize or plus material, material passing through the screening surface is the undersize or minus material, and material passing one screening surface and retained on a subsequent surface is the intermediate material.

The screening surface may consist of woven-wire, silk, or plastic cloth, perforated or punched plate, grizzly bars, or wedge wire sections.

Classification of screening operations and the range of separations that can be attained with various screens were given in concise form by Matthews (op. cit., 1971). See Table 19-5. Further details are given under "Equipment." Figure 19-14 indicates the size-range applicability of various screen types.

Mesh and Space Cloth Wire cloth is generally specified by "mesh," which is the number of openings per linear inch counting from the center of any wire to a point exactly 25.4 mm (1 in) distant, or by an opening specified in inches or millimeters, which is understood to be the clear opening or space between the wires. Mesh is generally favored for cloth 2 mesh and finer and clear opening for space cloth of 12.7-mm (½-in) opening and coarser.

Aperture Aperture, or screen-size opening, is the minimum clear space between the edges of the opening in the screening surface and is usually given in inches or millimeters.

Open Area The open area of a screen is the percentage of actual openings versus total screen area and can be determined by the formulas given in Fig. 19-22.

Particle-Size Distribution This is defined as the relative percentage by weight of grains of each of the different size fractions represented in the sample. It is one of the most important factors in evaluating a screening operation and is best determined by a complete size analysis using testing sieves.

Sieve Scale A sieve scale is a series of testing sieves having openings in a fixed succession; for example, in the original basic Tyler standard sieve scale the widths of the successive openings have a constant ratio of the square root of 2, or 1.414, while the areas of the successive openings have a constant ratio of 2. The Tyler scale has been enlarged to include intermediate openings so that the entire scale has successive openings according to the fourth root of 2, or 1.189. The sieve series adopted by the National Bureau of Standards, American Society for Testing and Materials, American National Standards Institute, and many countries applies the fourth-root-of-2 principle, and the openings are fully compatible with the Tyler standard scale even though the sieve designations may vary (Table 19-6).

TABLE 19-5 Types of Screening Operations

Operation and description	Type of screen commonly employed
Scalping—Strictly, the removing of a small amount of oversize from a feed which is predominantly fines. Typically, the removal of oversize from a feed with approximately a maximum of 5% oversize and a minimum of 50% half-size.	Coarse (grizzly); fine, same as fine separation; ultrafine, same as ultrafine separation.
Separation (*coarse*)—Making a size separation at 4 mesh and larger.	Vibrating screen, horizontal or inclined.
Separation (*fine*)—Making a size separation smaller than 4 mesh and larger than 48 mesh.	Vibrating screen, horizontal or inclined; high-speed low-amplitude vibrating screens; sifter screens; static sieves; centrifugal screens.
Separation (*ultrafine*)—Making a size separation smaller than 48 mesh.	High-speed low-amplitude vibrating screen; sifter screens; static sieves; centrifugal screens.
Dewatering—Removal of free water from a solids-water mixture. Generally limited to 4 mesh and above.	Horizontal vibrating screen; inclined vibrating screens (about 10°); centrifugal screen.
Trash removal—Removal of extraneous foreign matter from a processed material. Essentially a form of scalping operation. Type of screen employed will depend on size range of processed material—coarse, fine, or ultrafine.	Vibrating screen, horizontal or inclined; sifter screens; static sieves; centrifugal screen.
Other applications: Desliming—Removal of extremely fine particles from a wet material by passing it over a screening surface. *Conveying*—In some instances transport of the material may be as important as the operation. *Media recovery*—A combination washing and dewatering operation.	Vibrating screens, inclined and horizontal; oscillating screens; centrifugal screens.

Testing Sieves Many product specifications require size of material in terms of given percentages passing or retained on specified test sieves. Test sieves are also generally used to determine the efficiency of screening devices and the work of crushing and grinding machinery.

It is essential that standard sieves, with standard-size openings be used for sieve analyses. The time of screening and the method of agitating the material on the sieve should also be standard, and in many industries the practice of specifying the test-sieve designation and the time and method of sieving is followed. An excellent overview of the theory and use of standard testing sieves is given in the *Testing Sieve Handbook,* No. 53, published by W. S. Tyler, Inc., Mentor, Ohio.

U.S. Sieve Series The American Society for Testing and Materials in cooperation with the National Bureau of Standards and the American National Standards Institute has further refined the U.S. sieve series, combining the former coarse and fine series into a single series series with a fourth-root-of-2 ratio (Table 19-6). The openings in the individual sieves have remained unchanged except for minor adjustments in sieves coarser than 0.00673 m (6.73 mm). In the revised series, sieves 1 mm and coarser are identified by opening in millimeters, and those finer than 1 mm by their openings in microns.

Tyler Standard Sieve Series Many users base their tests on Tyler standard testing sieves (Table 19-6). The only difference between the U.S. sieves and the Tyler screen scale sieves is the identification method. Tyler screen scale sieves are identified by nominal meshes per linear inch while the U.S. sieves are identified by millimeters or micrometers or by an arbitrary number which does not necessarily mean the mesh count. The Tyler standard sieve scale series has as its base a 200-mesh screen in which the opening is 7.37×10^{-5} m (0.0029 in) and the wire diameter 5.35×10^{-5} m (0.0021 in).

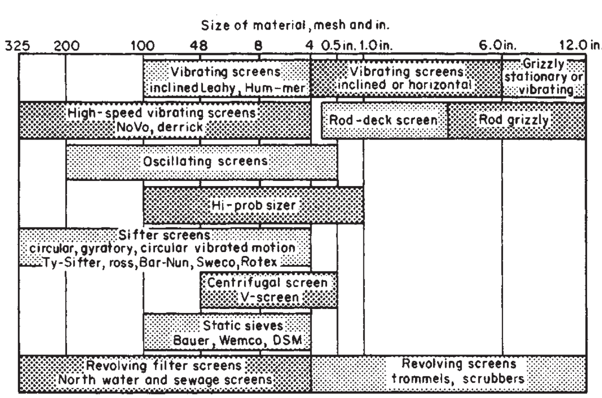


FIG. 19-14 Range of separations that can be obtained with various kinds of screens. To convert inches to meters, multiply by 0.0254. [*Matthews,* Chem. Eng. *(Feb. 15, 1971).*]

International Test Sieve Series The International Organization for Standardization (ISO) has been intensifying its efforts to establish an international test sieve series. At a meeting held at The Hague in October 1959, the ISO provisionally recommended for adoption as an international standard 19 sieves as shown by the data in Table 19-6. These sieves correspond to every alternating sieve in the fourth-root-of-2 U.S. sieve series from 0.022-m (8/8-in) opening to 325 mesh. The ISO has prepared a manual of sieving procedures that is available through the American Society for Testing and Materials.

The Ro-Tap testing sieve shaker (Fig. 19-15) manufactured by the W. S. Tyler, Inc., is the standard machine for automatically carrying out sieve-test procedures with accuracy and dependability. This device is built to hold a series of 0.203-m- (8-in-) diameter Tyler standard scale testing sieves and imparts to the sieves both a circular and a tapping motion. In effect, it reproduces the circular and tapping motion given testing sieves in hand sieving but does it with a uniform mechanical action. An important feature of the Ro-Tap is that both speed and stroke are fixed and not adjustable. This ensures the comparability between a number of sieve tests not only in a manufacturer's plant but between tests of a supplier and a customer.

The Ro-Tap is equipped to handle from 1 to 13 sieves at a time and is equipped with a timer that automatically terminates the test after any predetermined time.

Another mechanical shaker is the End-Shak, made by the Newark Wire Cloth Co. Sieves used are Newark test sieves, made to conform with the U.S. standard series.

A number of less expensive sieve shakers are on the market, such as the Dynamic, by Soiltest Inc., Chicago; the Cenco-Meinzer, by Central Scientific Co., Chicago; the Tyler portable, by W. S. Tyler, Inc., Mentor, Ohio; and also a number of electromagnetic vibratory shakers. The latter should be used only when strict comparability with other tests is not required, since it is difficult to be sure that identical intensity of vibration was present in the tests being compared.

Equipment Screening machines may be divided into five main classes: grizzlies, revolving screens, shaking screens, vibrating screens, and oscillating screens. Grizzlies are used primarily for scalping at 0.05 m (2 in) and coarser, while revolving screens and shaking screens are generally used for separations above 0.013 m (½ in). Vibrating screens cover this coarse range and also down into the fine meshes. Oscillating screens are confined in general to the finer meshes below 4 mesh.

Grizzly Screens These consist of a set of parallel bars held apart by spacers at some predetermined opening. Bars are frequently made of manganese steel to reduce wear. A grizzly is widely used before a primary crusher in rock- or ore-crushing plants to remove the fines before the ore or rock enters the crusher. It can be a stationary set of bars or a vibrating screen.

Stationary grizzlies. These are the simplest of all separating devices and the least expensive to install and maintain. They are normally lim-

TABLE 19-6 U.S. Sieve Series and Tyler Equivalents
(ASTM—E-11-61)

Sieve designation		Sieve opening		Nominal wire diam.		
Standard	Alternate	mm	in (approx. equivalents)	mm	in (approx. equivalents)	Tyler equivalent designation
107.6 mm	4.24 in	107.6	4.24	6.40	0.2520	
101.6 mm	4 in†	101.6	4.00	6.30	.2480	
90.5 mm	3½ in	90.5	3.50	6.08	.2394	
76.1 mm	3 in	76.1	3.00	5.80	.2283	
64.0 mm	2½ in	64.0	2.50	5.50	.2165	
53.8 mm	2.12 in	53.8	2.12	5.15	.2028	
50.8 mm	2 in†	50.8	2.00	5.05	.1988	
45.3 mm	1¾ in	45.3	1.75	4.85	.1909	
38.1 mm	1½ in	38.1	1.50	4.59	.1807	
32.0 mm	1¼ in	32.0	1.25	4.23	.1665	
26.9 mm	1.06 in	26.9	1.06	3.90	.1535	1.050 in
25.4 mm	1 in†	25.4	1.00	3.80	.1496	
22.6 mm*	⅞ in	22.6	0.875	3.50	.1378	0.883 in
19.0 mm	¾ in	19.0	.750	3.30	.1299	.742 in
16.0 mm*	⅝ in	16.0	.625	3.00	.1181	.624 in
13.5 mm	0.530 in	13.5	.530	2.75	.1083	.525 in
12.7 mm	½ in†	12.7	.500	2.67	.1051	
11.2 mm*	7/16 in	11.2	.438	2.45	.0965	.441 in
9.51 mm	⅜ in	9.51	.375	2.27	.0894	.371 in
8.00 mm*	5/16 in	8.00	.312	2.07	.0815	2½ mesh
6.73 mm	0.265 in	6.73	.265	1.87	.0736	3 mesh
6.35 mm	¼ in†	6.35	.250	1.82	.0717	
5.66 mm*	No. 3½	5.66	.223	1.68	.0661	3½ mesh
4.76 mm	No. 4	4.76	.187	1.54	.0606	4 mesh
4.00 mm*	No. 5	4.00	.157	1.37	.0539	5 mesh
3.36 mm	No. 6	3.36	.132	1.23	.0484	6 mesh
2.83 mm*	No. 7	2.83	.111	1.10	.0430	7 mesh
2.38 mm	No. 8	2.38	.0937	1.00	.0394	8 mesh
2.00 mm*	No. 10	2.00	.0787	0.900	.0354	9 mesh
1.68 mm	No. 12	1.68	.0661	.810	.0319	10 mesh
1.41 mm*	No. 14	1.41	.0555	.725	.0285	12 mesh
1.19 mm	No. 16	1.19	.0469	.650	.0256	14 mesh
1.00 mm*	No. 18	1.00	.0394	.580	.0228	16 mesh
841 micron	No. 20	0.841	.0331	.510	.0201	20 mesh
707 micron*	No. 25	.707	.0278	.450	.0177	24 mesh
595 micron	No. 30	.595	.0234	.390	.0154	28 mesh
500 micron*	No. 35	.500	.0197	.340	.0134	32 mesh
420 micron	No. 40	.420	.0165	.290	.0114	35 mesh
354 micron*	No. 45	.354	.0139	.247	.0097	42 mesh
297 micron	No. 50	.297	.0117	.215	.0085	48 mesh
250 micron*	No. 60	.250	.0098	.180	.0071	60 mesh
210 micron	No. 70	.210	.0083	.152	.0060	65 mesh
177 micron*	No. 80	.177	.0070	.131	.0052	80 mesh
149 micron	No. 100	.149	.0059	.110	.0043	100 mesh
125 micron*	No. 120	.125	.0049	.091	.0036	115 mesh
105 micron	No. 140	.105	.0041	.076	.0030	150 mesh
88 micron*	No. 170	.088	.0035	.064	.0025	170 mesh
74 micron	No. 200	.074	.0029	.053	.0021	200 mesh
63 micron*	No. 230	.063	.0025	.044	.0017	250 mesh
53 micron	No. 270	.053	.0021	.037	.0015	270 mesh
44 micron*	No. 325	.044	.0017	.030	.0012	325 mesh
37 micron	No. 400	.037	.0015	.025	.0010	400 mesh

*These sieves correspond to those proposed as an international (I.S.O.) standard. It is recommended that wherever possible these sieves be included in all sieve analysis data or reports intended for international publication.

†These sieves are not in the fourth-root-of-2 series, but they have been included because they are in common usage.

ited to the scalping or rough screening of dry material at 0.05 m (2 in) and coarser and are not satisfactory for moist and sticky material. The slope, or angle with the horizontal, will vary between 20 and 50°. Stationary grizzlies require no power and little maintenance. It is, of course, difficult to change the opening between the bars, and the separation may not be sufficiently complete.

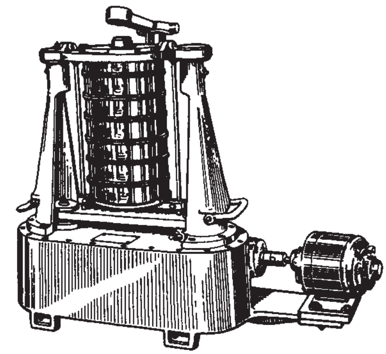


FIG. 19-15 Ro-Tap testing sieve shaker. (*W. S. Tyler, Inc.*)

Flat grizzlies. These, in which the parallel bars are in a horizontal plane, are used on tops of ore and coal bins and under unloading trestles. This type of grizzly is used to retain occasional pieces too large for the following plant equipment. These lumps must then be broken up or removed manually.

Vibrating grizzlies. These are simply bar grizzlies mounted on eccentrics so that the entire assembly is given a back-and-forth movement or a positive circle throw. These are made by companies such as Allis-Chalmers, Hewitt Robins, Nordberg, Link-Belt, Simplicity, and Tyler.

Revolving Screens Revolving screens, or trommel screens, once widely used, are being largely replaced by vibrating screens. They consist of a cylindrical frame surrounded by wire cloth or perforated plate, open at both ends, and inclined at a slight angle. The material to be screened is delivered at the upper end, and the oversize is discharged at the lower end. The desired product falls through the wire-cloth openings. The screens revolve at relatively low speeds of 15 to 20 r/min. Their capacity is not great, and efficiency is relatively low.

Mechanical Shaking Screens These screens consist of a rectangular frame which holds wire cloth or perforated plate and is slightly inclined and suspended by loose rods or cables or supported from a base frame by flexible flat springs. The frame is driven with a reciprocating motion. The material to be screened is fed at the upper end and is advanced by the forward stroke of the screen while the finer particles pass through the openings. In many screening operations such devices have given way to vibrating screens.

Shaking screens, such as the mechanical-conveyor type made by Syntron Co., may be used for both screening and conveying.

The advantages of this type are low headroom and low power requirement. The disadvantages are the high cost of maintenance of the screen and the supporting structure owing to vibration and low capacity compared with inclined high-speed vibrating screens.

Vibrating Screens These screens are used as standard practice when large capacity and high efficiency are desired. The capacity, especially in the finer sizes, is so much greater than that of any of the other screens that they have practically replaced all other types when efficiency of the screen is an important factor. Advantages include accuracy of sizing, increased capacity per unit area, low maintenance cost per ton of material handled, and a saving in installation space and weight.

There are a great number of vibrating screens on the market, but basically they can be divided into two main classes: (1) mechanically vibrated screens and (2) electrically vibrated screens.

Mechanically Vibrated Screens The most versatile vibration for medium to coarse sizing is generally conceded to be the vertical circle produced by an eccentric or unbalanced shaft, but other types of vibration may be more suitable for certain screening operations, particularly in the finer sizes. One well-known *four-bearing mechanically vibrated screen,* installed in an inclined position, is the Ty-Rock (Fig. 19-16). This is a balanced circle-throw machine mounted on a base frame, having a full-floating body mounted on shear rubber mounting units which absorb the shocks of heavy material and allow the shaft to revolve around its own natural center of rotation.

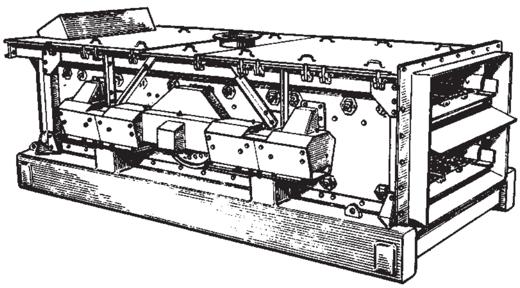


FIG. 19-16 Ty-Rock screen with air-seal enclosure. (*W. S. Tyler, Inc.*)

Two-bearing screens, of which there are many types, have the same screen body as the four-bearing type but without the two outer bearings and the base frame. The gyrating motion is caused by eccentric weights on the shaft, and the screen itself is supported by overhead cables or springs on the floor.

Screening machines actuated by rotating unbalanced weights have a symmetrical shaft through the screen body with an unbalanced flywheel on each end. Counterweights on each flywheel, which may be moved in relation to the shaft, permit adjustment of the amplitude of vibration. On some makes of machines the complete shaft assembly is contained in a unit bolted to the top of the screen body.

The horizontal-type screen is actuated by an enclosed mechanism consisting of off-center weights geared together on short horizontal shafts. The mechanism is usually mounted between the side plates and above the screen body (Fig. 19-17).

Electrically Vibrated Screens These screens are particularly useful in the chemical industry. They handle very successfully many light, fine, dry materials and metal powders from approximately 4 mesh to as fine as 325 mesh. Most of these screens have an intense, high-speed (25 to 120 vibrations/s) low-amplitude vibration supplied by means of an electromagnet.

Typical of these is the Hum-mer screen used throughout the chemical industry. Figure 19-18 shows one used throughout the fertilizer industry for handling mixed chemical fertilizers.

Oscillating Screens These screens are characterized by low-speed oscillations [5 to 7 oscillations per second (300 to 400 r/min)] in a plane essentially parallel to the screen cloth.

Screens in this group are usually used from 0.013 m (½ in) to 60 mesh. Some light free-flowing materials, however, can be separated at 200 to 300 mesh. Silk cloths are often used.

Reciprocating Screens These screens have many applications in chemical work. An eccentric under the screen supplies oscillation, ranging from gyratory [about 0.05-m (2-in) diameter] at the feed end to reciprocating motion at the discharge end. Frequency is 8 to 10 oscillations per second (500 to 600 r/min), and since the screen is inclined about 5°, a secondary high-amplitude normal vibration of about 0.0025 m ($\frac{1}{10}$ in) is also set up. Further vibration is caused by balls bouncing against the lower surface of the screen cloth.

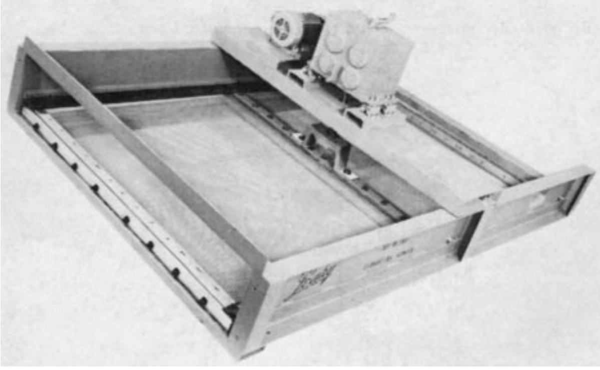


FIG. 19-17 Mechanically vibrated horizontal screen. (*Courtesy of Deister Concentrator Company, Inc.*)

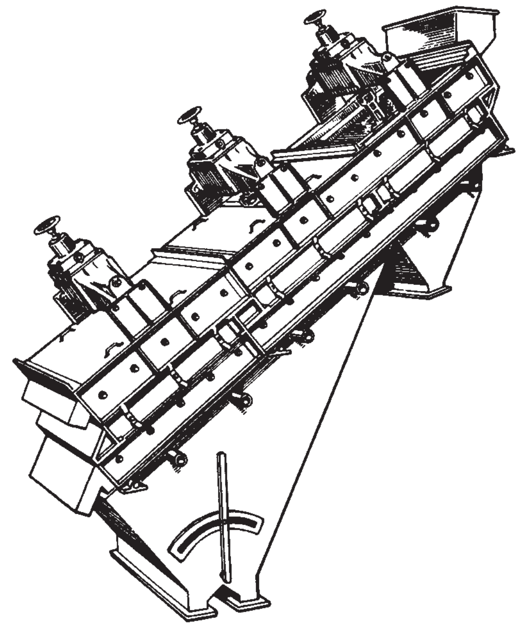


FIG. 19-18 Type 38 Hum-mer screen. (*W. S. Tyler, Inc.*)

These screens are used extensively in the United States and are standard equipment in many chemical and processing plants for handling fine separations even down to 300 mesh. They are used to handle a variety of chemicals, usually dry, light, or bulky materials, light metal powders, powdered foods, and granular materials. They are not designed for handling heavy tonnages of materials like rock or gravel. Machines of this type are exemplified by Fig. 19-19.

Gyratory Screens These are boxlike machines, either round or square, 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 gyratory screens have an auxiliary vibration caused by balls bouncing against the lower surface of the screen cloth. A typical machine is shown in Fig. 19-20. Machines of this type are operated con-

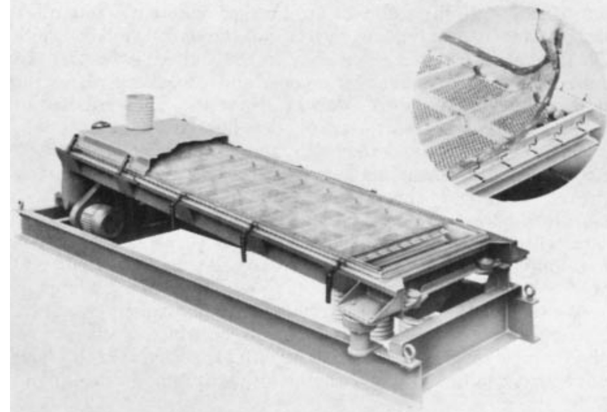


FIG. 19-19 Reciprocating screen. (*Courtesy of Rotex Corp.*)

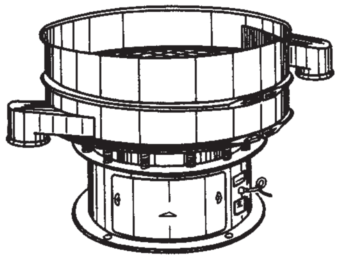


FIG. 19-20 Vibro-energy screen. (*Southwestern Engineering Company*.)

tinuously and can be located in line in pneumatic conveying systems as scalping screens. The size ranges from 0.6 to 1.5 m (24 to 60 in).

Gyratory Riddles These screens are driven in an oscillating path by a motor attached to the support shaft of the screen. The gyratory riddle is the least expensive screen on the market and is intended normally for batch screening.

Screen Surfaces The selection of the proper screening surface is very important, and the opening, wire diameter, and open area should all be carefully considered. The four general types of screening surfaces are woven-wire cloth, silk bolting cloth, punched plate, and bar or rod screens.

Woven-Wire Cloth This type has by far the greatest selection as to screen opening, wire diameter, and percentage of open area. Thousands of specifications are available from over 0.10 m (4 in) clear opening to 500 mesh. Woven-wire screens are obtainable in a variety of metals and alloys. Steel and high-carbon steel are generally favored for the coarser openings because of their abrasion-resistant qualities, and other materials, such as phosphor bronze, Monel, and stainless steel, are used for their corrosion-resisting or noncontamination qualities.

Square-mesh cloth is the conventional type of screen cloth, but there are many types of cloth with an oblong weave. This latter construction provides greater open area and capacity and in addition makes it possible to use stronger wire for the same size of screen opening and for the same percentage of open area.

In choosing a wire-cloth specification there must be a compromise between sharpness of separation, capacity, freedom from blinding, and life of the wire cloth. The square-mesh cloth will give the closest control of the maximum size particle in the undersize material; but the effective size of the openings will be reduced, because of the foreshortening when used at an angle of inclination, with consequent reduction in capacity. It should be realized that it is often necessary to use a cloth specification with an aperture larger than the smallest-size material acceptable in the oversize in order to ensure thorough removal of the undersize. A screen with a rectangular opening will increase the capacity with but little loss of sharpness when handling rounded or cubical grains. Slabby or flat material may also be handled on rectangular-opening cloth if the final-product specification will allow in the undersize a certain percentage of flat pieces having one dimension greater than the specified square-opening sieve. In other words pieces that might fall through a rectangular cloth and be allowed in the product might not go through the limiting square-mesh sieve on which the specification is based. If the through product is to be further ground or processed, a small amount of this material will not be objectionable.

Screen-cloth specifications having a relatively large length-to-width ratio are desirable when moisture or sticky material tends to cause blinding with square or short rectangular openings.

The finer the diameter of the wire from which a given specification is woven, the greater will be its screening capacity, although its screening life will be shorter. Since production capacity is generally more important than screen-surface cost, care should be taken to avoid using too heavy a specification which might restrict the capacity of the screening unit on which it is used and thus create a bottleneck in the flow.

Catalogs of wire-cloth manufacturers should be consulted for further study of the different types of wire-cloth specifications.

Silk Bolting Cloth This material originated in Switzerland and is generally woven from twisted multistrand-natural silk. The system of numbers and grades for both bolting cloth and gritz gauze has been handed down from the original Swiss weavers. In recent years, nylon and similar synthetic materials woven largely from monofilaments have been introduced. The nylon grades are generally designated by their micrometer opening and are available in light, standard, and heavy weights.

Comparative Openings of Screening Cloths In screening any material, the size of the particles going through the screen is determined by the actual opening and not by the number of meshes per linear unit. As a rule, the lighter grades of wire-screen cloth, having greater percentages of open area, screen more freely and accurately and should be used whenever they will give satisfactory length of service. Tables of comparative openings are available for selecting a screen specification with a specific opening or for picking a specification having a heavier or lighter wire but the same opening.

Punched Plates These are available in a variety of perforations including round, square, hexagonal, and elongated openings. Punched metal will generally wear longer than wire cloth and has more rigidity, which is an advantage in certain applications. However, it usually does not give the capacity per unit area that wire cloth does and is generally heavier. Its use is normally limited to the coarser separations.

Bar Screens These screens are generally used in handling large and heavy pieces of material. They are formed from rails, rods, or bars, suitably shaped; made from rolled steel or castings; fixed in parallel position and held by crossbars and spacers. Bars, which taper in thickness from top to bottom and may also taper in width from one end to the other, are recommended because they tend to avoid blinding.

Rod decks, composed of spring-steel rods approximately 0.6 m (2 ft) long, sprung into position between molded-rubber blocks, and held in position by means of rubber spacers, are also available.

Probability Screening Principle Probability screening uses the fact that particles moving almost at right angles to a screening surface are not likely to pass through when the particle size is greater than about half of the distance between the screen elements. Screens utilizing the probability principle are manufacturing by Dutch State Mines (DSM), Bartles (CTS), and Morgensen. The last-named incorporates multiple decks. Higher throughput, longer screen life, and lower capital costs are claimed for these screening systems. The performance of several types of probability screens was reviewed by Moir (op cit.).

Factors in Selecting Screening Equipment In attempting to pick a screening machine for a specific screening problem it should be emphasized that generalized formulas and charts used to predict screen capacity can give only an approximation because of the many variables which may affect performance. Screen consultants will readily admit that they must depend largely on laboratory tests and field experience. However, two governing factors bear mention: generally width of screen relates to capacity and length of screen relates to efficiency. Width is necessary to reduce the bed thickness to a practical maximum and length to allow the undersize to be removed without an inordinate amount of fines in the oversize. In attempting to choose a screening machine for a particular screening application the customer and manufacturer should consider the following: (1) Full description of material involved, including the name and type of material, bulk density, and physical characteristics such as hardness, particle shape, flow characteristics (free-flowing, sluggish, or sticky), percent of moisture and temperature. (2) Normal and maximum total rate of feed to screen. (3) Complete sieve analysis of screen feed, including maximum lump size, and sieve analysis of desired product. (4) Separation or separations required and the purpose of screening. Can slotted or rectangular openings be used in place of square openings? (5) Is screening to be accomplished dry or wet, and what amount of water is available? (6) Other important factors include method of delivering feed to the screen, open or closed circuit, open or enclosed screens, previous screening experience with the material, flow sheet or description of related equipment, operating hours per day, power available, and space limitations.

Variables in Screening Operations It will readily be seen that many variables in a screening operation can easily be changed in the field, and practical operators will always be trying to improve their operations or adapt them to new products or processes. Capacity and efficiency in screening operations are closely related. Capacity may be large if low efficiency is not objectionable. Usually, as the tonnage to a screen is increased, efficiency is decreased.

Method of Feed The screening machine must be fed properly in order to obtain maximum capacity and efficiency. The feed should be spread evenly over the full width of the screen cloth and approach the screen surface in a direction parallel to the longitudinal axis of the screen and at as low a practical velocity as is possible.

Screening Surfaces It is generally agreed that the most efficient screening results when a series of single-deck screens is used. This is true because lower decks of multiple-deck screens are not fed so that their entire area is used and because each separation requires a different combination of angle, speed, and amplitude of vibration for maximum performance.

Angle of Slope The optimum slope of inclined vibrating screens is that which will handle the greatest volume of oversize and still remove the available undersize required by the standards of the particular operation. To separate a material into coarse and fine fractions, the bed thickness must be limited so that vibration can stratify the load and allow fines to work their way to the screen surface and pass through the opening. Increased slope naturally increases the rate of travel, and at a given rate it reduces the bed thickness.

In the oscillating screen the angle of inclination must be coordinated with the speed and stroke for best results.

Direction of Rotation In circle-throw screens somewhat greater efficiency can be obtained by counterflow rotation, that is, having the material move down the screen against the rotation. Screens rotating with the flow of material will handle greater tonnage and operate at a lower angle.

Vibration Amplitude and Frequency Speed and amplitude of vibration should be designed to convey the material properly and to prevent blinding of the cloth. They are somewhat dependent upon the size and weight of the material being handled and are related to the angle of installation and the type of screen surface. The object, of course, is to see that the feed is properly stratified for the most efficient separation.

Noise and Safety Noise is generated in screening due to the impact of the feed material on the screen surface. The drive mechanism also generates noise. Rubber and rubber bearings reduce substantially feed-impact noise with the added longer life of the decks. Noise from the drive mechanism is reduced by enclosing the mechanism in a box or by adding rubber linings to the side plates to dampen the noise.

Depending on the feed materials, the dust generated during operation may be hazardous because of possible emissions and toxicity. These hazards must be carefully evaluated before proper design of the facility and selecting the apparatus.

Performance Formulas

Screen Efficiency There is confusion concerning the meaning of screen efficiency, as a uniform method for figuring efficiency has never been established. A sound method of evaluating screen performance is given by W. S. Tyler, Inc., Mentor, Ohio, in its *Sieve Handbook,* no. 53. In this formula, when material put through the screen is the desired product, "efficiency" is the ratio of the amount of undersize obtained to the amount of undersize in the feed.

$$E = (R \times d)/b \tag{19-5}$$

where E = efficiency, R = percent of fines through the screen, d = percent finer than the designated size in screen fines, and b = percent finer than the designated size in screen feed.

When the object is to recover an oversize product from the screen, efficiency may be expressed as a ratio of the amount of oversize obtained to the amount of true oversize:

$$E = (O \times c)/a \tag{19-6}$$

where O = percent of oversize over the screen, c = percent coarser than the designated size in screen oversize, and a = percent coarser than the designated size in screen feed.

Other formulas for the derivation of screen efficiency are used. Taggart (*Handbook of Mineral Dressing*) gives the formula

$$E = 100 \times \frac{100(e - v)}{e(100 - v)}$$

where E is the efficiency, e is the percentage of undersize in the feed, and v is the percentage of undersize in the screen oversize.

Graphical methods of evaluating efficiency, using sieve analyses, are also employed and are recommended when serious research on screening is done.

Estimating Screen Capacity Various methods of predicting screening capacity have been proposed, and each has its limitations. The throughflow method of Matthews uses the following equation:

$$A = 0.4C_t/C_u F_{oa} F_s \tag{19-7}$$

where A = screen area
C_t = throughflow rate
C_u = unit capacity
F_{oa} = open-area factor
F_s = slotted-area factor

The unit capacity C_u can be determined from Fig. 19-21. Figure 19-22 can be used to determine the open-area factor F_{oa}, and the slotted-opening factor F_s for various screen types is given in Table 19-7.

WET CLASSIFICATION

GENERAL REFERENCES: Dyakowski, T., and Williams, R. A., "Modelling Turbulent Flow within a Small Diameter Hydrocyclone," *Chemical Engineering Science,* vol. 48, p. 1143, 1993. Heiskanen, K., "Particle Classification," Scarlett, B., (ed.), *Powder Technology Series,* Chapman & Hall, 1993. Fitch, B., "Gravity Sedimentation Operations," McKetta J. J., (ed.), *Unit Operations Handbook,* vol. 2, Mechanical Separation and Material Handling, p. 51, Marcel Dekker, Inc., New York, 1993. Neal Abernathy, M. W., "Gravity Settlers, Design" McKetta J. J., (ed.), *Unit Operations Handbook,* vol. 2, Mechanical Separation and Material Handling, p. 127, Marcel Dekker, Inc., New York, 1993. Zanker, A., "Gravity Settlers, Sizing of Decanters," McKetta J. J., (ed.), *Unit Operations Handbook,* vol. 2, Mechanical Separation and Material Handling, p. 136, Marcel Dekker, Inc., New York, 1993. Rajamani, R. K., and Milin L., "Fluid-Flow Model of the Hydrocyclone for Concentrated Slurry Classification" Svarovsky, L., and Thew M. T., (eds.), *Hydrocyclones Analysis and Applications,* p. 95, Kluwer Academic Publishers, Dordrecht, The Netherlands, 1992. Dahlstrom D. A., "Fundamental of Solid-Liquid Separation," Mular A. L., and Anderson, M. A., (eds.), *Design and Installation of Concentration and Dewatering Circuits,* p. 103, SME, New York, 1986. Kelly, E. G., and Spottiswood, D. J., *Introduction to Mineral Processing,* John Wiley & Sons, New York, 1982. Tarr, D. T., Jr., "Hydrocyclones," Weiss, N. L. (ed.), *SME Mineral Processing Handbook,* vol. 1., p. 3D-10, SME, New York, 1985. Devulapalli, B., and Rajamani, R. K., "A comprehensive CFD model for particle size classification in industrial hydrocyclones," Claxton, D., Svarovsky, L., and Thew, M. (eds.), Hydrocyclone '96, p. 83–104, *Mechanical Engineering Publications Limited,* London and Bury St Edmunds, UK, 1996.

Introduction Wet classification is defined here as that art of separating the solid particles in a mixture of solids and liquid into fractions according to particle size or density by methods other than screening. In general, the products resulting are (1) a partially drained fraction containing the coarse material (called the underflow) and (2) a fine fraction along with the remaining portion of the liquid medium (called the overflow).

The classifying operation is carried out in a pool of fluid pulp confined in a tank arranged to allow the coarse solids to settle out, whereupon they are removed by gravity, mechanical means, or induced pressure. Solids which do not settle report as overflow. Mesh of sepa-

TABLE 19-7 Slotted-Opening Factors

Screen type	Length-to-width ratio
Square and slightly rectangular openings	Less than 2
Rectangular openings	Equal to or greater than 2 but less than 4
Slotted openings	Equal to or greater than 4 but less than 25
Parallel-rod decks	Equal to or greater than 25

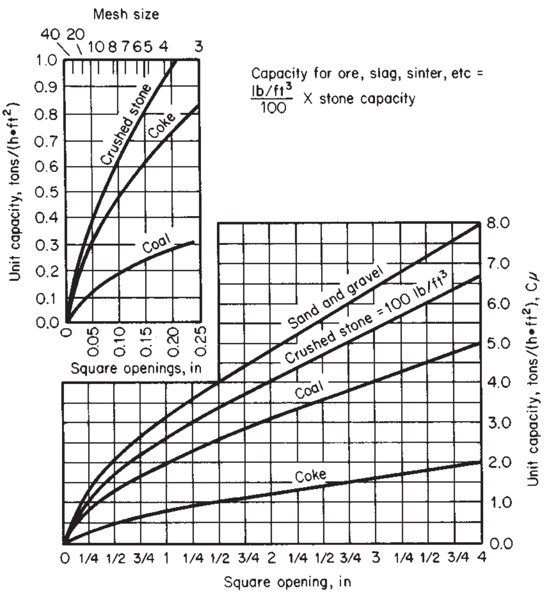


FIG. 19-21 Unit capacity (C_u) for square-opening screens. To convert inches to meters, multiply by 0.0254; to convert tons per hour-square foot to kilograms per second-square meter, multiply by 2.7182.

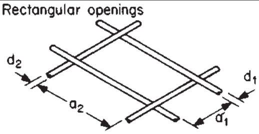
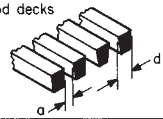
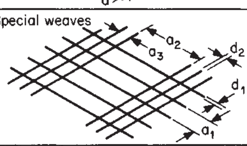
Aperture	Formula	
Rectangular openings 	$F_{oa} = \frac{a_1 a_2}{(a_1 + d_1)(a_2 + d_2)} \times 100$ F_{oa} is open area, %; d is diameter of wire, or horizontal width of bar (for plate); a is clear opening dimension	(21-4)
Square openings Specified by opening size	$F_{oa} = 100 \left(\frac{a}{a+d}\right)^2 \quad a_1 = a_2 = a \quad d_1 = d_2 = d$	(21-5)
Square openings Specified in mesh, m	$F_{oa} = 100 a^2 m^2 \qquad m = \frac{1}{a+d}$	(21-6)
Parallel-rod decks 	$F_{oa} = \frac{100a}{(a+d)}$	(21-7)
Special weaves 	Assuming $a_3 = a_1$; $F_{oa} = 100 \left[\frac{a_1(a_2 + 2a_1)}{(a_2 + 2a_1 + 3d_2)(a_1 + d_1)}\right]$	(21-8)

FIG. 19-22 Open-area factor (F_{oa}) for flow-through screen-capacity calculation.

ration as used in this text is the screen size retaining 1½ percent of the overflow solids.

All wet classifiers depend on the difference in settling rate between coarse and fine or heavy- and light-specific gravity particles to be separated. Rates can be controlled to some extent by mild agitation, providing for hindered settling, and centrifugal force versus gravity in centrifuging types of units.

Several fundamental laws on classification are:

1. Coarse particles have a relatively faster settling velocity than fine particles of the same specific gravity.

2. Heavy-gravity particles have a relatively faster settling velocity than light-gravity particles of the same size. High solids concentration increases the viscosity and density of the fluid medium.

3. Settling rates of solid particles become progressively slower as the viscosity or density of the fluid medium increases.

a. There is a point (called critical dilution) where the lowering of density or viscosity by addition of more liquid creates a velocity effect which overcomes normal classification settling velocity, thereby coarsening the separation.

b. Conversely, at this point less liquid will cause a viscosity and buoyancy effect which will also coarsen the separation.

Typical problems to be solved by wet-classification means fall into several broad categories such as (1) to effect a simple sand-slime separation resulting in two products; (2) to effect a concentration of smaller heavy-gravity particles in a product containing larger light-gravity particles; (3) to obtain a washing effect by successive dewatering, repulping in weaker solution, and further dewatering; (4) to sort solids having a full range of screen sizes into a number of partials each having a short range of screen sizes; and (5) to achieve closed-circuit control of grinding mills.

Classification is by definition used preponderantly in the treatment of raw materials. However, these raw materials find their way into chemical processing per se and thus become of interest to the chemical engineer, particularly when the products to be treated react better when of a defined cleanliness, size, gravity, or moisture content.

Classifier types fall into two basic categories: (1) gravitational and (2) centrifugal classifiers. Gravitational classifiers can be subdivided into (1) sedimentation and (2) hydraulic classifiers. Furthermore each type falls into mechanical and nonmechanical types.

There are numerous machines and machine types to obtain a number of different particle-size classes from solids having a full range of sizes, and there is much overlapping in the possibilities. Usually, one type will provide optimum economy for the specific problem involved.

The quick reference Table 19-8 will help by way of rapid elimination of poor possibilities. Following that the brief comments and illustrations will help pinpoint most probable selections. Further study of the more elaborate data in the references and contact with the usual suppliers are recommended, as there are many possible modifications of equipment which can improve operating results from any type of machine finally selected.

Nonmechanical Classifiers

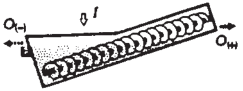
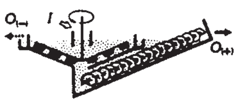
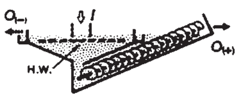
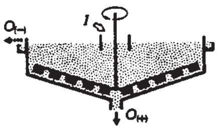
Cone Type Cone classifiers are one of the oldest types but are still used for relatively crude work because of low cost of installation. They are limited in diameter because of high headroom requirements caused by the ±60° sloping sides. Units are simple and are often fabricated locally with millwright ingenuity fashioning the apex opening arrangement for adjustment or control of the spigot coarse product. Operating attention is often necessary to a greater degree than for the more positive mechanical types. Cost figures are not available.

Hydrocyclone The wet cyclone classifier has rapidly achieved prominence since the 1950s and continues to gain popularity throughout chemical and ore-dressing industries. Standout virtues are its low capital cost and ability to make extremely fine separations by proper adjustment of design/operating condition. See Fig. 19-23.

In simplest terms the unit has a top cylindrical section and a lower conical section terminating in an apex opening, often adjustable. The unit operates under pressure induced by a static hydraulic head or by means of a pump forcing new feed into the cylindrical portion tangentially, thus producing centrifuging action and vortexing. The cover has a downward-extending pipe to cut the vortex and remove the overflow product called vortex finder. Coarse solids travel down the sides of the steeply sided cone section and are removed in a partially dewatered form at the apex.

Hydrocyclones are available in numerous sizes and types ranging from pencil-sized 10-mm diameters of plastic to the 1.2-m (48-in) diameter of rubber-protected mild or stainless steel. Porcelain units 25 to 100 mm (1 to 4 in) in diameter are becoming popular, and in the 150-mm (6-in) size the starch industry has standardized on special molded nylon types. Small units for fine-size separations are usually manifolded in multiple units in parallel with up to 480 ten-mm

TABLE 19-8 The Major Types of Classifiers

Classifier	(Type*)	Description	Size (m) Width Diameter Max. length	Limiting size (max. feed size)	Feed rate (t/hr)	Vol. % solids Feed overflow underflow	Power (kW)	Suitability and applications
Sloping tank classifier (spiral, rake, drag) 	(M-S)	Classification occurs near deep end of sloping, elongated pool. Spiral, rake or drag mechanism lifts sands from pool.	0.3 to 7.0 2.4 (spiral) 14	1 mm to 45 μm (25 mm)	5 to 850	Not critical 2 to 20 45 to 65	0.4 to 110	Used for closed circuit grinding, washing and dewatering, desliming; particularly where clean dry underflow is important. (Drag classifier sands not so clean.) In closed circuit grinding discharge mechanism (spirals especially) may give enough lift to eliminate pump.
Log washer 	(M-S)	Essentially a spiral classifier with paddles replacing the spiral.	0.8 to 2.6 0.6 to 1.1 4.6 to 11	(100 mm)	40 to 450		7.5 to 60	Used for rough separations such as removing trash, clay from sand. Also to remove or break down agglomerates.
Bowl classifier 	(M-S)	Extension of sloping tank classifiers, with settling occurring in large circular pool, which has rotating mechanism to scrape sands inwards (outwards in Bowl Desiltor) to discharge rake or spiral.	0.5 to 6.0 1.2 to 15 12	150 μm to 45 μm (12 mm)	5 to 225	Not critical 0.4 to 8 50 to 60 (15 to 25 in Bowl Desiltor)	Bowl: 0.75 to 7.5 Rake: 0.75 to 20	Used for closed circuit grinding (particularly regrind circuits) where clean underflow is necessary. Larger pool allows finer separations. Bowl Desiltor has larger pools (and capacities). Relatively expensive.
Hydraulic bowl classifier 	(M-F)	Basically a hydraulic bowl classifier. Vibrating plate replaces rotating mechanism in pool. Hydraulic water passes through perforations in plate and fluidizes sands.	–1.2 to 3.7 1.2 to 4.3 12	1 mm to 100 μm (12 mm)	5 to 225	Not critical 2 to 15 50 to 65	Vib: 2.2 to 7.5 Rake: 3.7 to 15	Gives very clean sands and has relatively low hydraulic water requirements (0.5 t/t underflow). One of the most efficient single-stage classifiers available for closed circuit grinding and washing. Relatively expensive.
Cylindrical tank classifier 	(M-S)	Effectively an overloaded thickener. Rotating rake feeds sands to central underflow.	— 3 to 45 —	150 μm to 45 μm (6 mm)	5 to 625	Not critical 0.4 to 8 15 to 25	0.75 to 11	Simple, but gives relatively inefficient separation. Used for primary dewatering where the separations involve large feed volumes, and underflow drainage is not critical.

*M: Mechanical transport of sands to discharge
N: Nonmechanical (gravity or pressure) discharge of underflow
S: Sedimentation classifier
F: Fluidized bed classifier
From Kelley, E. G. and D. J. Spottiswood, *Introduction to Mineral Processing*, John Wiley & Sons, New York, 1982, pp. 200–201, with permission.

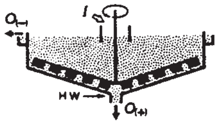
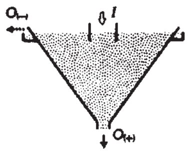
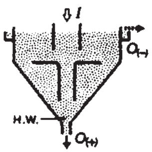
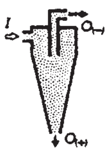
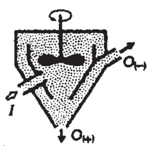
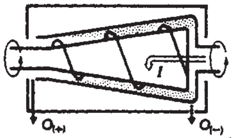
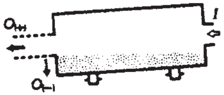
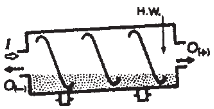
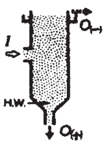
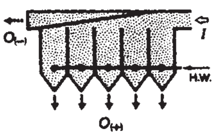
Classifier	(Type*)	Description	Size (m) Width Diameter Max. length	Limiting size (max. feed size)	Feed rate (t/hr)	Vol. % solids Feed overflow underflow	Power (kW)	Suitability and applications
Hydraulic cylindrical tank classifier 	(M-F)	Hydraulic form of overloaded thickener. Siphon-Sizer (N-F) uses siphon to discharge underflow instead of rotating rake.	— 1.0 to 40 —	1.4 mm to 45 µm (25 mm)	1 to 150	Not critical 0.4 to 15 20 to 35	0.75 to 11	Two-product device giving very clean underflow. Requires relatively little hydraulic water (2 t/t solids feed). Used for washing, desliming, and closed circuit grinding.
Cone classifier 	(N-S)	Similar to cylindrical tank classifier, except tank is conical to eliminate need for rake.	— 0.6 to 3.7 —	600 µm to 45 µm (6 mm)	2 to 100	Not critical 5 to 30 35 to 60	None	Low cost (simple enough to be made locally), and simplicity can justify relatively inefficient separation. Used for desliming and primary dewatering. Solids buildup can be a problem.
Hydraulic cone classifier 	(M-F)	Open cylindrical upper section with conical lower section containing slowly rotating mechanism.	— 0.6 to 1.6 —	400 µm to 100 µm (6 mm)	10 to 120	Not critical 2 to 15 30 to 50	3 to 7.5	Used primarily in closed circuit grinding to reclassify hydrocyclone underflow.
Hydrocyclone 	(N-S)	(Pumped) pressure feed generates centrifugal action to give high separating forces, and discharge.	— 0.01 to 1.2 —	300 µm to 5 µm (1400 µm to 45 µm)	to 20 m^3/min	4 to 35 2 to 15 30 to 50	35 to 400 kN/m^2 pressure head	Small cheap device, widely used for closed circuit grinding. Gives relatively efficient separations of fine particles in dilute suspensions.
Air separator 	(N-S)	Similar shape to hydrocyclone, but higher included angle. Internal impellor induces recycle within classifier.	— 0.5 to 7.5 —	2 mm to 38 µm	to 2100		4 to 500	Used where solids must be kept dry, such as cement grinding. Air classifiers may be integrated into grinding mill structure.

TABLE 19-8 The Major Types of Classifiers (*Concluded*)

Classifier	(Type*)	Description	Size (m) Width Diameter Max. length	Limiting size (max. feed size)	Feed rate (t/hr)	Vol. % solids Feed overflow underflow	Power (kW)	Suitability and applications
Solid bowl centrifuge 	(M-S)	Power generates high settling forces. Slurry centrifuged against rotating bowl, and removed by slower rotating helical screw conveyor within bowl.	— 0.3 to 1.4 1.8	74 μm to 1 μm (6 mm)	0.04 to 2.5 m^3/ min	2 to 25 0.4 to 20 5 to 50	11 to 110	Relatively expensive, but high capacity for a given floor space; used for finer separations.
Scrubber 	(M-S)	Essentially a rotating drum mounted on slight incline.	— 1.5 to 3.5 3 to 10	(450 mm)	to 700		1 to 55	Similar applications to log washer, but lighter action. Tumbling (85% critical speed) provides attrition to remove clay from sand. Also removes trash.
Countercurrent classifier 	(M-F)	One form based on scrubber, another on spiral classifier. They have wash water added to flow essentially horizontally in opposite direction to underflow which is conveyed and *resuspended* by some form of spiral.	— 0.5 to 3.3 (spiral type) 12 (spiral type)	2 mm to 40 μm	3 to 600	Not critical 2 to 15 50 to 65	0.2 to 19	Very clean coarse product, but relatively low capacity for a given size.
Elutriator 	(N-F)	Basically a tube with hydraulic water fed near bottom to produce hindered settling. Underflow withdrawn through valve at base. Column may be filled with network to even out flow.	— 1.2 to 4.3 —	2.4 mm to 100 μm (7.5 mm)	4 to 120	15 to 35 0.4 to 5 20 to 35	0.75 for valves	Simple and relatively efficient separation. Normally a two-product device but may be operated in series to give a range of size fractions.
Pocket classifier 	(N-F)	A series of classification pockets, with decreasing quantities of hydraulic water in each, producing a range of product sizes.	0.5 to 6.0 — 12	2.4 mm to 100 μm (10 mm)	4 to 120	15 to 35 0.4 to 5 20 to 35		Efficient separations, but requires 3 t hydraulic water/t solids feed. Used to produce exceptionally clean underflow fractioned into narrow size ranges.

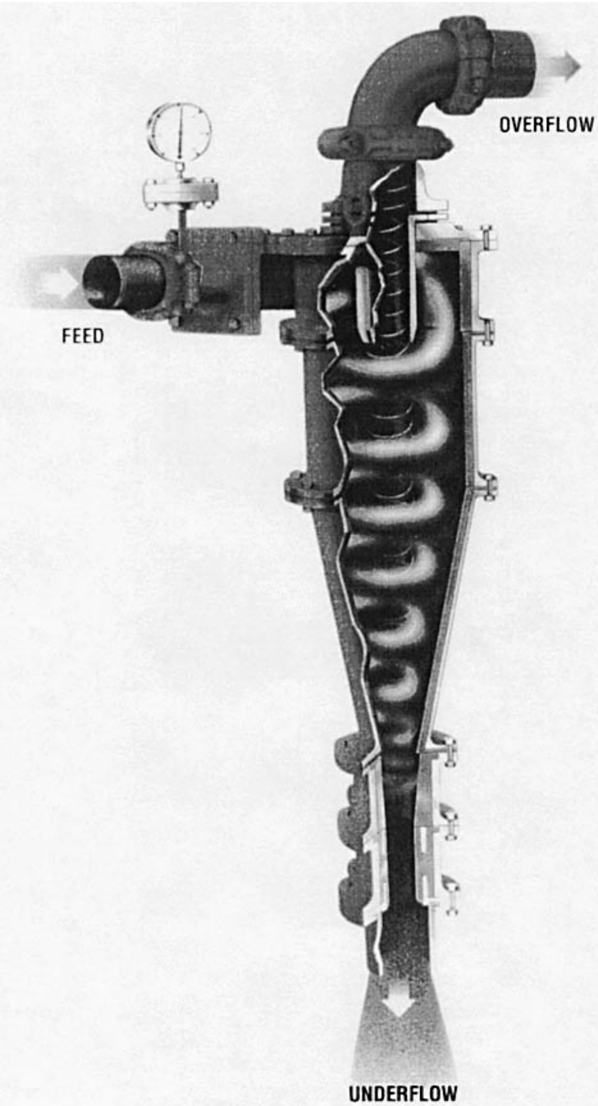


FIG. 19-23 Hydrocyclone. (*Courtesy Krebs Engineers.*)

cyclones in a single case. Larger sizes may be used singly or manifolded by outside piping.

The hydrocyclone has mostly replaced other classifiers in closed-circuit grinding.

Typical uses more in line with chemical applications are degritting milk of lime and of red mud in alumina production, removal of carbonaceous material in upgrading gypsum produced in making phosphoric acid, open-circuit washing of fine uranium pulps, classification of crystal magma such as lactose and sodium bisulfite, and classifying pigment and plastic beads into size ranges.

Mechanical Classifiers

Drag Classifiers Single endless-belt or chain suspensions with cross flights running in an inclined trough have long been used for draining and classifying. Many styles, sizes, and shapes have resulted from locally built units, and operating results on a scientific basis are meager. In general, they have served their purpose consistent with the type of engineering and cost included.

The Hardinge Overdrain* classifier is of the belt type, but it embodies the innovation of allowing entrapped water and slimes to escape through holes in the belt just uphill of the cross flights in an upward direction and thence flow down on top of the belt into the pool without again intermingling with the coarse product being advanced by the cross flights. Coarse product with lower moisture and fines content result from this action. Modern design and materials of construction permit sizes up to 3 m (10 ft) wide and 12.5 m (41 ft) long on steeper than average slopes and for very high tonnages.

Rake and Spiral Classifiers Rake-type classifiers such as the Dorr† classifier and spiral types such as the Akins‡ have been the workhorses for general-classification problems for half a century, and their names describe the mechanisms installed in sloping-bottom tanks. See Fig. 19-24. Mechanically the devices are powerfully built, and functionally they are versatile and flexible. They were the first classifiers used successfully for closed-circuit grinding. Separations as fine as 325 mesh can be accomplished at reduced tonnage rates.

Control of water into the classifiers is important since separation into fine and coarse products is made largely by the buoyancy, viscosity, and degree of agitation in the pool.

Both types of devices will produce rake products of consistent moisture content even with considerable variation in feed tonnage or volume.

Bowl Classifier The bowl classifier was developed to provide more separation area necessary for fine separations consistent with high tonnage. In essence a shallow bowl with revolving plows is superimposed over a rake or screw dewatering section. Feed enters at the center of the bowl, and fine solids overflow at the periphery. Coarse solids collected on the bowl bottom are raked to the center for discharge into the dewatering compartment below where wash water may be added for counterflow.

Hydrocyclones are rapidly taking over the functions formerly handled by bowl classifiers because of lower capital costs and floor-area requirements.

Bowl Desilter The bowl desilter provides for separation areas well beyond areas possible in bowl classifiers, in which larger sizes are limited by mechanical design. Its use is in operations involving large flow volumes and fine separations. Rake tonnages can be great or small with a dewatering compartment to suit the conditions.

In the bowl desilter the rotating blades in the bowl plow outward and discharge settled coarse material at the periphery, where it drops into the drainage compartment. This configuration does away with the long cantilevered rake construction necessary in bowl classifiers.

Widest application has been for the recovery of and drainage of very fine material overflowing coarser washing units in glass sand, concrete sand, coal, and limestone processing plants.

Hydroseparator The hydroseparator is merely a thickener-type machine receiving more flow than can be clarified in the area provided. Thus the overflow contains fine solids, and the greater the feed rate per unit of area the coarser the solids in the overflow.

Classification efficiency of the hydroseparator compares with that of the cone classifier and is appreciably lower than that obtained from mechanical or hydraulic units. The chief virtue of the hydroseparator is its ability to receive and slough off great quantities of water at low per-unit-volume cost.

Typical applications include primary dewatering of phosphate rock matrix and silica sand products following wet screening. In ore dressing it is used mainly to protect large-diameter thickeners by scalping out +65-mesh material.

Solid-Bowl Centrifuge The Bird solid-bowl centrifuge uses power instead of gravity and can develop centrifugal forces up to 1800 times the force of gravity. It is therefore a unique type in classification practice.

The unit consists essentially of two rotating elements, the outer being a solid-shell conical-shaped bowl and the inner comprising a helical-screw conveyor revolving at a speed slightly lower than that of the bowl. Raw feed slurry is delivered through a stationary feed pipe

* Trademark of Koppers Co., Inc.
† Trademark of Dorr-Oliver Inc.
‡ Trademark of Mine & Smelter.

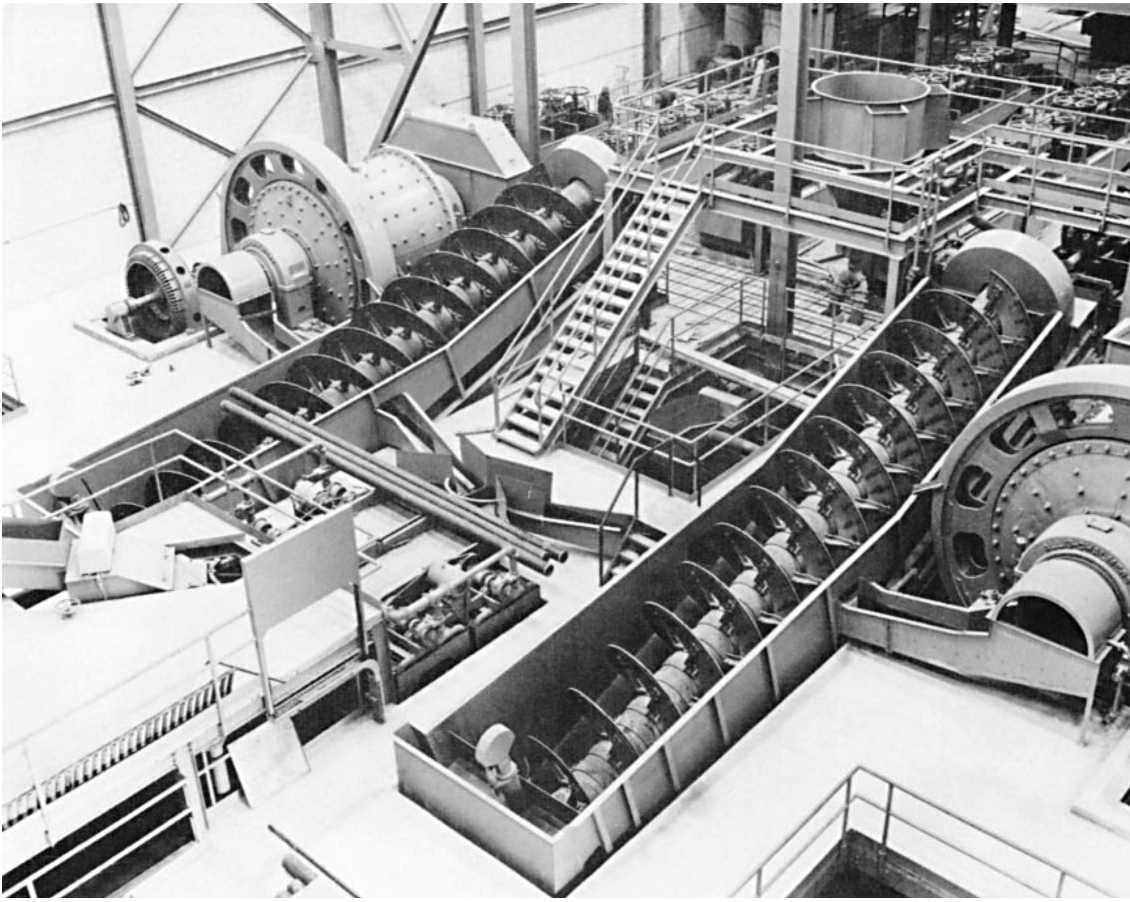


FIG. 19-24 Spiral type classifier. Wemco S-H 78-in classifiers in closed-circuit grinding operation at St. Joseph Lead Co., Indian Creek plant. (*Courtesy Wemco Div., Envirotech Corp.*)

to the conveyor, where, urged by centrifugal force, it is transferred to the revolving bowl. A circumferential classifying area is formed and contained at the larger diameter of the cone shell. The ports for oversize material are located closer to the axis of rotation than the ports for the overflow to effect a beach line and drainage.

Centrifugal force deposits the oversize particles against the bowl wall, from which they are conveyed by the helix. The overflow fractions flow around the helix to the liquid-discharge ports. Size of separation is controlled by feed rate and degree of centrifugal force.

Several prime features of this totally enclosed unit are its high capacity per unit of floor area, small volume of material in process, high degree of separation, and shear action for dispersion of solids. Typical applications are desliming to upgrade cement rock, sizing of abrasives, fractionating for reagent control, and classification of pigments.

Countercurrent Classifier The countercurrent classifier is an inclined, slowly rotating cylindrical drum with continuous spiral flights attached to the interior of the shell forming helical troughs. Direction of rotation is such that material in the troughs is impelled toward the higher end. The lower end of the shell is closed except for a central overflow opening. Attached to the upper end is a coarse solid dewatering elevator which rotates with the shell. Wash water introduced at the upper end drains from the lifting flights above the normal water level and progresses countercurrently to the sand toward the overflow.

Usual application is for sand-slime separations, washing and for closed construction restricting escape of heat and chemical fumes, easy start-up after shutdown, and general simplicity. Weights range from 500 to 55,000 kg (1100 to 120,000 lb).

Hydraulic Classifiers

Jet Sizer* and SuperSorter† The Jet Sizer and SuperSorter are multicompartment and, therefore, multiproduct classifiers operating on the basis of hindered settling. The classification pockets are arranged in series for throughflow with parallel pockets to take care of high tonnage size fractions. Each compartment is served with low-pressure hydraulic water.

Hydraulic classification ensures the highest separating efficiency obtainable by wet-classification means. The amount of hydraulic water is controlled so that in each succeeding compartment the coarsest particles are maintained in hindered-settling condition and the finer fractions pass along for similar treatment. Two compartments will normally capture 90 percent of a two-screen-size fraction. Spigot discharge is controlled by air-actuated valves in the Jet Sizer and motor-driven pincer-type valves in the SuperSorter. Solid fractions can be taken from single or combinations of compartments as desired.

Typical applications include careful sizing of silica-glass sand, washing phosphate rock, sizing of abrasives, smokeless powder, sodium aluminate, etc.

D-O SiphonSizer* The D-O SiphonSizer (Fig. 19-25) is a high-efficiency hydraulic classifier developed originally for the washing and sizing of phosphate rock. In ore-dressing work it is normally a two-product unit; but by use of an upper column sealed at the top and

* Trademark of Dorr-Oliver Inc.
† Trademark of Deister Concentrator Company, Inc.

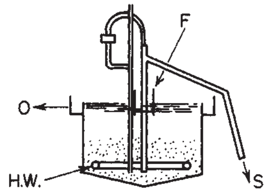


FIG. 19-25 D-O SiphonSizer.

open at the bottom, three products are possible: coarse, intermediate, and fine fractions.

Feed to be sized is put into hindered-settling condition by hydraulic water in quantity only sufficient to teeter the smallest particle wanted in the coarse product. The finer fractions report to the overflow or pass into the upper column for removal in a three-product unit.

Coarse solids are discharged by siphons extending to the bottom of the hindered-settling zone. Siphon control is obtained by a novel hydrostatically actuated valve which makes or breaks the siphon to flow only when the teeter zone is in correct condition. Discharge by an intermediate fraction from the upper column is by means of additional siphons. Hydraulic-water consumption is considerably lower than required for multipocket sizers.

SiphonSizers vary so widely in configuration that general cost data are not meaningful.

JIGGING

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Introduction A jig is a mechanical device used for separating materials of different specific gravities by the pulsation of a stream of liquid flowing through a bed of materials. The liquid pulsates, or "jigs" up and down, causing the heavy material to work down to the bottom of the bed and the lighter material to rise to the top. Each product is then drawn off separately.

Jigging is one of the oldest processes used for concentrating heavy mineral particles from the light. Jigging is best suited for coarse material that is unlocked in the size range 20 mesh and coarser and when there is a considerable difference between the effective specific gravity (sp gr mineral minus sp gr water) of the valuable and the waste material. Jigs are simple in operation. Water consumption is high, and the tailings losses on metallic ores are usually high. Also, because of the scarcity of still-available ore deposits having coarse mineralization, the jigs are used to a limited extent, mostly to treat iron ores, a few lead-zinc ores, and some heavy nonmetallic ores like barite and diamonds. Jigging is widely employed for the concentration of coal. Over 50 million tons of coal is concentrated by jigs annually in the United States. High-speed types of jigs are used for the recovery of fine-grained heavy minerals from placer deposits, gold, tin, and tungsten, and for recovering a portion of coarse metallic values liberated in ball-mill grinding circuits. Jigging has been superseded in many milling operations by the adoption of the dense-media process or by fine grinding followed by flotation.

Principles of Operation The principle of jig operation can easily be understood by taking a 10-mesh laboratory sieve, placing a 1 cm thick bed of a mixture of heavy and light particles, immersing the sieve in a bucket of water, and oscillating it up and down under water. The pulsations will dilate the bed of material and make the particles settle as the larger and denser particles forming the lower layers with the finer and lighter particles on the top.

The motion of the mixture of particles during jigging is modulated by the amplitude and frequency of jigging strokes and these strokes result in displacement of particle bed in a harmonic wave (Fig. 19-26*a*). During the pulsation stroke the original bed (Fig. 19-26*b*) dilates resulting in the bed as shown in Fig. 19-26*c*. During the suction stroke the bed of particles undergoes differential initial acceleration followed by hindered settling and consolidation trickling (Wills, op. cit.). It is found that the initial acceleration of the particles is independent of size and dependent only on the densities of the solids and fluid, thus causing the heavy particles to settle faster than the lighter as illustrated in Fig. 19-26*d*. The hindered settling on the other hand is controlled by both size and density of particles with smaller particles settling less and heavier settling more (Fig. 19-26*e*). Finally, during the consolidation trickling the bed begins to compact, the larger particles interlock and allow the smaller grains to move downwards (Fig. 19-26*f*).

Types of Jigs A jig is essentially an open tank filled with water and provided with a horizontal screen on the top and a *hutch* compartment fitted with a spigot (Fig. 19-27*a*). A layer of coarse, heavy particles, known as *ragging,* is placed on the top of the screen onto which the feed slurry is introduced. The feed moves over the ragging and the separation takes place as the bed is pulsated by a different mechanical device. The heavy particles are collected into the hutch compartment and removed through the spigot while the lighter particles are made to overflow from the top of the tank.

Several types of jigs are currently available with the main differences being in the pulsating mechanism and the stroke modification. Figures 19-27*b* through Fig. 19-27*e* illustrate four different designs that are commonly used. One of the earliest designs of jigs is the *Harz* and it uses reciprocating plunger with differential piston action (Fig. 19-27*b*). The Harz jig is commonly used in the treatment of gold, tungsten, and chromite ores. Remer jig (Fig. 19-27*c*) is an improvement over Harz by providing a driving mechanism that has two motions, a normal jig pulse of 80 to 120 strokes per minute on which imposed a fast pulse in the range of 200 to 300 per minute. This kind of jig is commonly used in concentrating such materials as iron and barite ores and in removing impurities such as wood, shale, and lignite from sand and gravel. In contrast, *Baum* and *Batac* jigs make use of air pulsations and are widely used in the coal-preparation industry to reduce the ash content of the run-of-mine coals. The standard Baum jig (Fig. 19-27*d*) operates by forcing air under pressure at about 17.2 kPa (2.5 lbf/in^2) into a large air chamber on one side of the jig vessel to pulsate the jig water which in turn pulsate the bed of particles fed onto the screen. Several design variations exist in the removal of lighter coal and heavier ash fractions (Green, op. cit). The Batac jig (Fig. 19-27*e*) is a modification of the Baum jig in that it employs multiple air chambers under the screen with electronic controls for air input and exhaust. This design is found to provide a uniform flow across the whole bed and a wide control of the speed and length of the jigging strokes. Batac jig is reported to treat both coarse- and fine-size coals satisfactorily (Chen, 1980) and has become an industry standard for coal cleaning (Zimmerman).

Jig Feed In coal washing jigging is practiced on unsized material as coarse as 175 mm (7 in). In metal-milling practice jigging is now seldom employed on material coarser than 20 mm (¾ in). Float-and-sink methods have largely superseded jigs as a way of concentrating metallic ores in the minus 75 to plus 10-mm (3 to plus ½-in) range. Shaking tables usually are considered more efficient than jigs for treating ores finer than 2 mm (10 mesh). Jigs are used in some plants to obtain flowsheet simplicity since they can handle a wide range of sizes. Jigs, except when extremely heavy minerals such as gold, galena, cassiterite, or tungsten minerals are treated, recover only a small percentage of the sizes finer than 65 mesh (¼ mm).

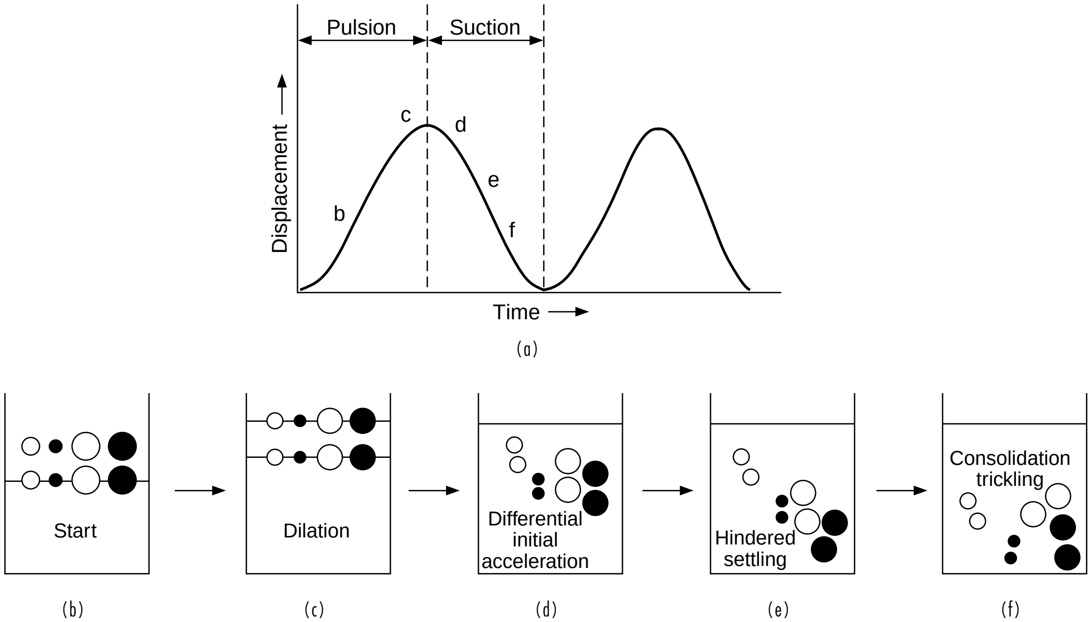

FIG. 19-26 Movement of particles in a jig. (*a*) Displacement of the bed as a function of time. (*b*) Starting position of particles. (*c*) After dilation. (*d*) After differential initial acceleration. (*e*) After hindered settling. (*f*) After consolidation trickling.

Capacity The Jeffrey-Baum will treat minus 100-mm (4-in) coal at the rate of 8 kg/(s·m²) [3 tons/(h·ft²)] of active screen area. For fine sizes capacity decreases. A standard 1.52- by 4.87-m (5- by 16-ft) Wemco-Remer jig will treat minus 9.5-mm (⅜-in) iron ore at the rate of 7.5 to 11.3 kg/s (30 to 45 tons/h). A Cooley jig, a variation of the Harz jig consisting of six compartments 1.07 by 1.22 m (42 by 48 in), will handle 6 to 7.5 kg/s (25 to 30 tons/h) of minus 19-mm (½-in) Mid-Continent zinc ore. The largest commercially available jig is the IHC Cleveland 25, a circular jig of 7.5-m (24.6-ft) diameter with a nominal capacity range of 30 to 60 kg/s (130 to 260 tons/h) of coal.

Power Requirements The power required in jigging depends on the screen area, the size of material treated, the percentage of opening in the jig screen, the depth of the bed, the length of stroke, and the choke frequency. The power required for plunger-type jigs treating 12.7-mm (½-in) material is about 7 W/m² (0.1 hp/ft²) jig screen surface.

Water Consumption Jigs require much water. In most installations, the Harz-type jig uses 0.006 to 0.01 m³ water/kg (1500 to 2500 gal/ton) material treated. Water requirements for treating minus 10-mm (⅜-in) iron ore in a Wemco-Remer rougher-cleaner jig circuit are approximately 0.005 m³ water/kg (1200 gal/ton) of material processed.

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Wet Tabling Tabling is a concentration process whereby a separation between two or more minerals is effected by flowing a pulp across a riffled plane surface inclined slightly from the horizontal, differentially shaken in the direction of the long axis, and washed with an even flow of water at right angles to the direction of motion. A separation between two or more minerals depends mainly on the difference in specific gravity between the minerals and to a lesser degree on the shape and size of the particles. The process is best suited for the concentration of ore and coal where there is a considerable difference between the effective specific gravity (sp gr mineral minus sp gr water) of the valuable and the waste material. Tables treat metallic ores effectively in the size range from 6 to 150 mesh but can be used to treat lighter materials such as coal of a considerably larger size.

Tabling is best suited for the treatment of material containing only one valuable mineral that is free at a granular size and when a considerable difference exists between the effective specific gravities of the mineral constituents. Flotation has been found to be best in treating complex ores containing several valuable minerals, those requiring fine grinding for liberation, and those having small gravity differentials.

The heaviest particles in a table feed are the least affected by the current of water washing down over the tables, and they collect in the riffles along which they move to the end of the table. The lighter materials ride above the heavy minerals and tend to be washed over the riffles to the low side of the table. Suitable launders are placed at the end of the low side of the table to catch the various products as they are discharged. These launders are provided with movable dividing devices to separate the concentrates from the middlings and the middlings from the tailings. It seldom is possible in tabling to make a sharp separation of the feed into a high-grade concentrate and a low-grade tailing with one pass. Some material of intermediate grade is almost invariably present as a band between these products, and it is customary to return such middlings either with or without additional grinding to the head of the circuit for retreatment. The amount of middling recirculated may amount to 25 percent of weight of the feed to the table.

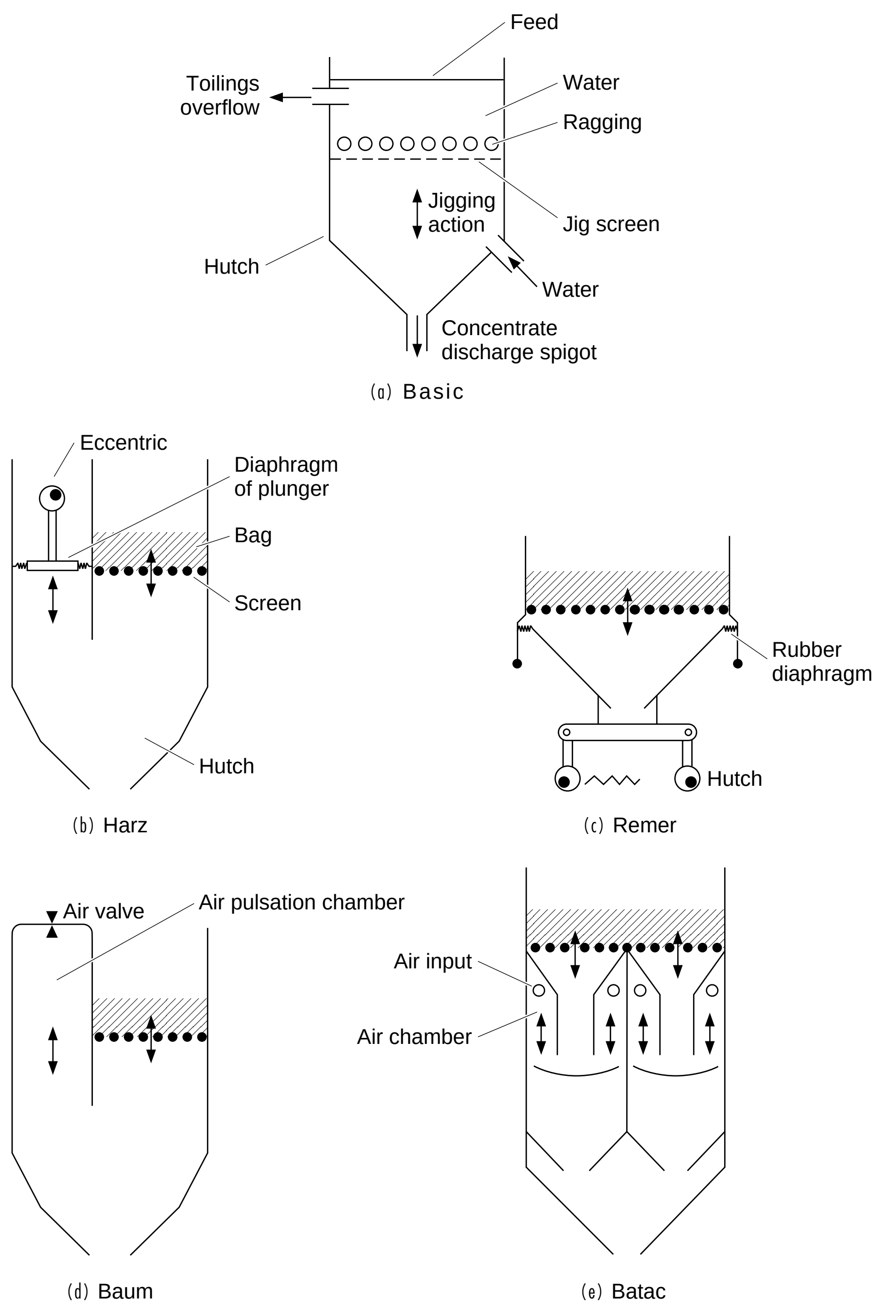


FIG. 19-27 Schematic diagrams of the jigs. (*a*) Basic, (*b*) Denver/Harz, (*c*) Remer, (*d*) Baum, and (*e*) Batac.

Tables usually are surfaced either with heavy battleship linoleum or with rubber. The riffles may be a clear grade of sugar pine or may be rubber strips. Such riffles usually taper from the feed end of the table to the discharge end. Almost all mill operators employ different styles of riffling table, which they believe best for their particular separations. The usual method of riffling is shown in Fig. 19-28.

If the object of tabling is to produce as clean a concentrate as possible, a diagonal area in the upper discharge side corner is left unriffled. This area is known as the cleaning deck. If the table is to be used in making only a rough concentrate and a finished tailing, the riffling is extended by many operators. Tables are provided with adjustable tilting devices so that the transverse slope may be varied. The head motion 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 that causes the material to migrate toward the discharge end. The length of stroke may be adjusted. This will vary from 0.03 m (1¼ in) for coarse material to 0.01 m (½ in) for fines. Modern tables operate from 4 Hz (270 strokes/min) for coarse to 6 Hz (350 strokes/min) for fines.

Present table practice is to use multiple decks. Multiple-deck tables consisting of from two to three decks effect space saving proportionate to the number of decks employed. They also have the advantage in

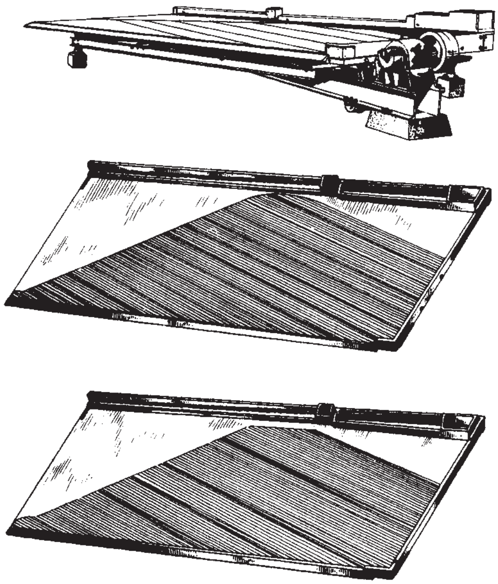


FIG. 19-28 Deister-Overstrom diagonal deck table. Center, diagonal deck with pool riffle system for sand; bottom, diagonal deck with pool riffle system for fine sand and slime.

that no heavy floor supports need be supplied since such tables are supported by suspended mountings. Multiple-deck installations reduce capital expenditures since a single motor and less piping and fewer launders are required than for a comparable number of single-deck installations. A two-deck configuration is shown in Fig. 19-29.

General information for standard-size tables operating on various-sized feeds is shown in Table 19-9. The No. 6 table of the Deister Concentrator Company, Fort Wayne, Indiana, has a diagonal deck approximately 1.83 m (6 ft) wide and 4.27 m (14 ft) long. The No. 7 table used primarily for coal work is approximately 2.44 m (8 ft) wide and 4.88 m (16 ft) long. The figures given apply to single-deck installations. In modern practice, each table, whether it be a single-deck or a multiple-deck installation, is driven by a single motor which is connected to the actuating mechanism by a V-belt drive. The installed

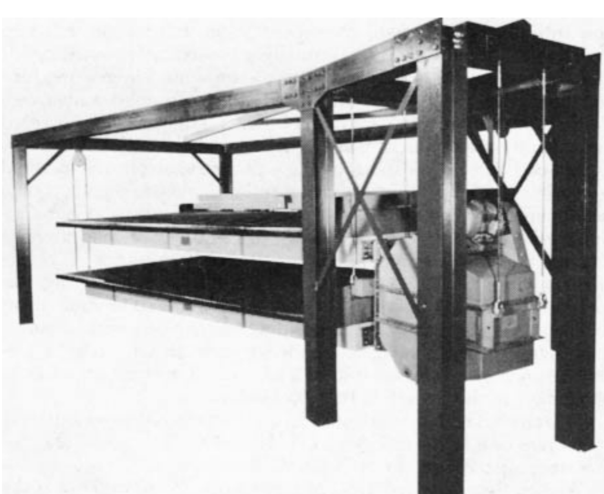


FIG. 19-29 Two-deck concentrating table. (*Courtesy Deister Concentrator Company, Inc.*)

horsepower for the large No. 7 deck is 1120 W (1.5 hp). A comparable figure for the smaller No. 6 deck is 746 W (1 hp) per deck. The actual power consumed in operation is somewhat less.

An essential factor for good table operation is that the rate of feed must be uniform, both as to tonnage and as to physical properties. No one factor will cause more trouble to the table operator than to have a surging feed. The feed to tables may be unsized, or it may be either screened or hydraulically classified. For treating fine coals a common procedure is to use hydrocyclones both to deslime the material and to give a cyclone underflow of about 40 percent solids, which constitutes the table feed.

Tabling is a relatively cheap operation. If the feed is uniform, one operator can take care of many tables. In a modern coal plant with multiple-deck tables, a single operator can handle the tabling of as much as 300 kg/s (1200 tons/h). In an ore-tabling plant such as a lead or zinc operation, a table operator can watch as many as 50 tables with a total capacity in the order of 50 kg/s (200 tons/h). Labor is the principal item of cost. Power requirements and maintenance are both low. The installed cost of a table including supports and launders is from \$8000 to \$15,000 per deck. In the past, one of the disadvantages of table installation was the relatively large floor space required for the tonnage treated. This disadvantage has now largely been overcome by the use of multiple-deck tables. Their main advantage is that, in the size range for which they are suited, tabling is a cheap and effective method of concentrating simple ores and coal.

Dry Tabling Tabling may be done dry as well as wet, and for such use tables of special design are used. The Sutton, Steele and Steele table is an example of this type of equipment. It 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 when it is desirable to treat material dry, either because of water shortage or because it is undesirable to wet the materials. An advantage of this table is the ability to handle material coarser than that treated on most wet tables. Ores as coarse as 0.006 m (¼ in) and coal as coarse as 0.076 m (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 improved dry methods for sizing fine material by the use of various cyclonelike devices has tended to increase the use of this apparatus on finer sizes.

Dry tables are used commercially in the separation of many types of minerals. Their greatest use is in the treatment of coal, but ilmenite, various tungsten ores, and even copper ores are so treated. Another important use is the cleaning of industrial materials such as seeds, cork, bagasse, fiber, nuts, wood chips, and coffee. 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 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, the oil-filmed surface, and the 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, mica, and other ores.

The process is limited to granular material in the size range from 10 to 100 mesh. In this respect it differs from flotation, which functions best on material 48 mesh and finer. For best results the material should be well deslimed and should be conditioned with the agglomerating reagents at a high percentage of solids, 65 percent or greater. A collector is used that will selectively film the mineral to be agglomerated. In phosphate and limestone practice, this collector is usually a

TABLE 19-9 Generalized Operating Data for Superduty Diagonal-Deck Concentrating Table

Table No.	Feed	Feed size	Feed capacity, tons/hr	Speed, rpm	Stroke, in	Water with feed, gal/min	Dressing water, gal/min	Size of deck
6	Ore	¼ in.–35 mesh	2.0–10.0	275	1.25	30–150	10–100	6′5″ × 14′1″
6	Ore	35–150 mesh	1.0–2.5	285	0.75	16–40	5–20	6′5″ × 14′1″
6	Ore	Minus 150 mesh	0.25–1.0	300	.50	3–12	3–10	6′5″ × 14′1″
7	Coal	1½ in	15.0–25.0	270	1.25	125–210	55–90	8′¼″ × 16′9¼″
7	Coal	¾ in	10.0–15.0	280	1.00	60–85	20–35	8′¼″ × 16′9¼″
7	Coal	½ in	7.5–12.0	285	1.00	42–65	18–31	8′¼″ × 16′9¼″
7	Coal	⅛ in	5.0–7.5	290	0.75	28–42	12–18	8′¼″ × 16′9¼″
7	Coal	1⁄16 in	3.0–5.0	290	.75	15–28	9–12	8′¼″ × 16′9¼″

NOTE: To convert inches to meters, multiply by 0.0254; to convert tons per hour to kilograms per second, multiply by 0.252; to convert revolutions per minute to hertz, multiply by 0.0167; to convert gallons per minute to cubic meters per second, multiply 6.309×10^{-5}; and to convert feet to meters, multiply by 0.3048.

cheap fatty acid such as tall oil. In potash separation long-chain amines are used to film sylvite (KCl).

A bulk oil is always used in addition to the collector to give body to the film and to assist in forming agglomerates. In Florida practice, it is customary to use 0.14 to 0.23-kg/ton (0.3- to 0.5-lb/ton) tall oil and 1.8 to 23 kg/ton (4 to 5 lb/ton) of a 22°Bé fuel oil. Operating data for the agglomerate tabling of phosphate and potash ore are shown in Table 19-10.

Agglomerate tabling works best on simple ores consisting of two free minerals. It has several advantages over the usual tabling method in that it can be used to separate two minerals the difference in specific gravity of which is so small that an effective separation cannot be made by gravity separation alone. Tables treating an agglomerated feed have a considerably larger capacity than tables using untreated feeds, since the capacity of a table treating an agglomerated feed is limited only by the carrying capacity of the riffles. Disadvantages of the method that must be considered are the cost of the reagents used and the fact that if the mineral fraction filmed is the one to be sold, the oily film may be objectionable and must be removed.

SPIRAL CONCENTRATION

General References: Adair, "New Method for Recovery of Flake Mica," *Min. Eng.,* **3,** 252 (1951). Brown, "Humphreys Spiral Concentration on Mesabi Range Ores," *Trans. Am. Inst. Min. Metall. Pet. Eng., Min. Branch,* **184,** 187 (1949). Gleeson, "Why the Humphreys Spiral Works," *Eng. Min. J.,* **146**(3), 85 (1945). Burt, *Gravity Concentration Technology,* Elsevier (1985). Lenhart, "Spiral Concentrators for Gravity Separation of Minerals," *Rock Prod.,* **54**(12), 92, 131 (1951). Miller, "Design and operating experience with the Goldsworthy Mining Limited Batac jig and spiral concentrator iron ore beneficiation plant," *Minerals Engineering,* **4,** 411 (1991). Roberts, "How New Highland Plant Recovers Titaniferous Minerals," *Min. World,* **17**(11), 52, 72 (1955). Roe, *Iron Ore Beneficiation,* Minerals Publishing Company, Lake Bluff, Ill., 1957. Sivamohan and Forssberg, "Principles of spiral concentration," *Int. J. Min. Proc.,* **15,** 173 (1985). Thompson, "The Humphreys Spiral: Some Present and Potential Applications," *Eng. Min. J.,* **151**(8), 87 (1950). Thompson, "The Humphreys Spiral Concentrator: Its Place in Ore Dressing," *Min. Eng.,* **10**(1), 84 (1958).

Principle of Operation Spiral concentration of ores and industrial materials is based primarily on the specific-gravity differentials of the materials to be separated. The shape factor of the feed material is also important, and utilization of reagentized feed can change the apparent specific gravity of component minerals by forced attachment of air bubbles to mineral flocs. The best known spiral-type concentrator is the Humphreys spiral concentrator, which first proved its commercial feasibility in 1943. In that year an Oregon plant successfully demonstrated ability to recover chromium minerals from low-grade beach sand deposits.

The Humphreys spiral concentrator is a spirally shaped channel or launder with a modified semicircular cross section, as illustrated in Fig. 19-30. The standard spiral consists of five complete turns, but three-turn units are used in some instances when an unusually rapid and clean separation takes place, as in second-stage or cleaner spirals. There is a drop of 0.34 m (13.5 in)/turn as the flowing pulp progresses from the top to the bottom of the spiral. One spiral concentrator occupies about 0.37 m^2 (4 ft^2) of floor space and about 2.1 m (7 ft) of headroom measured from feed to discharge box. The optimum particle-size range of feed particles for spirals is about 10 to 200 mesh (2 to 0.074 mm).

As the feed slurry flows down the spiral channel, the particles with the highest specific gravity sink to the bottom and move inward toward the inside of the channel. The lighter-weight particles move to the outside and are carried away by the faster, more dilute pulp stream. At 120° intervals circular concentrate "ports," or openings, appear in the bottom of the channel near the inside edge, as illustrated in Fig. 19-31. There are 15 ports in a five-turn spiral, but usually more than half of them are blocked off with smooth stainless-steel disks in order to allow proper configuration of the concentrate stream and good washing of the concentrate. Wash water is available along the entire inside edge of the spiral, where it flows at the rate of 0.2 to 0.6 L/s (3 to 10 gal/min) in a separate wash-water channel. Thus the spiral provides repeated washing stages as the pulp flows down the channel. Generally the richest concentrate is withdrawn from the concentrate ports near the top end of the spiral. Concentrate ports are fitted with very simple stainless-steel "belt-disk splitters," which can split out the desired portion of the concentrate stream. As the gradually impoverished pulp flows down the spiral, wash water is proportioned from the wash-water channel by a series of notches and directed so as to wash repeatedly across the concentrate band and sweep out unwanted gangue particles. The lowest-specific-gravity solids wash outward, and the finest particles actually climb the sloping wall of pulp on the outside of the channel. The concentrate withdrawn from ports near the bottom end of the spiral is usually low-grade and, if liberated, may be recirculated to obtain additional recovery of values and a higher grade of concentrate.

Although the spiral concentrator is mechanically a very simple piece of equipment, the separating action taking place is complex. It involves centrifugal force, friction against the spiral surface, gravity, and the drag of the water.

TABLE 19-10 Operating Data for Agglomerate Tabling of Phosphate and Potash Ore

Type of table	Feed size	Feed capacity, tons/h	Table speed, r/min	Table stroke, in	Water with feed, gal/min	Dressing water, gal/min	Size of deck
No. 6 superduty diagonal deck	10–48 mesh	2.5–3.5	295	1.0	20–40	8–15	6′5″ × 14′1″

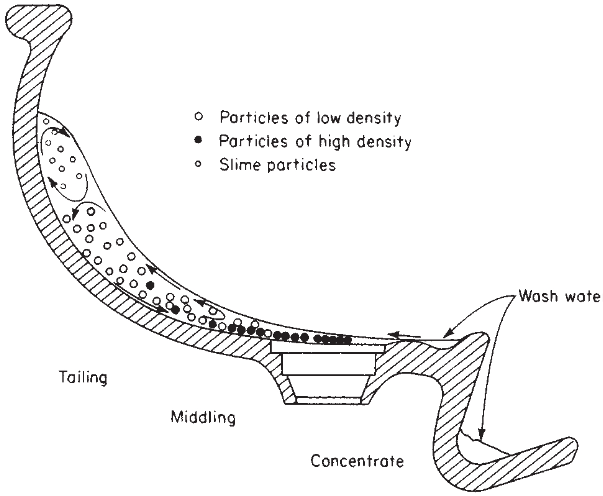


FIG. 19-30 Heavy-mineral separation in the Humphreys spiral concentrator.

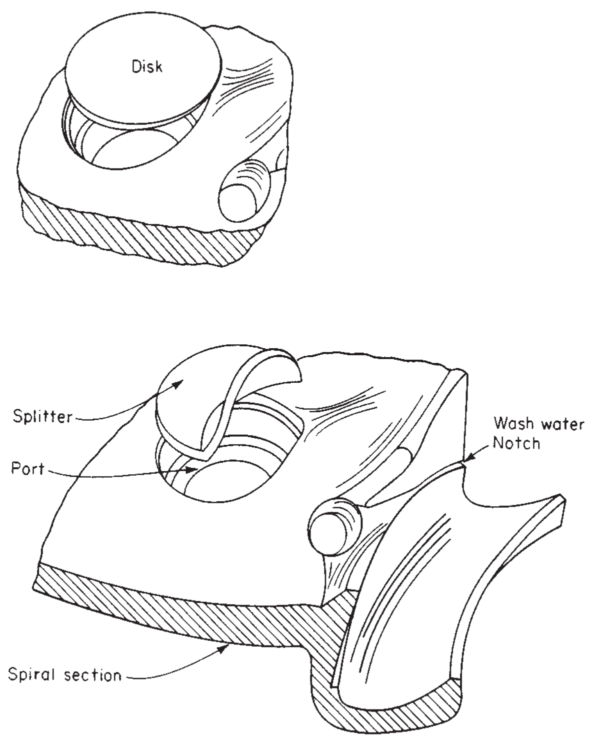


FIG. 19-31 Disks and splitters as used in the Humphreys spiral concentrator.

Basic Requirements for Spiral Concentration Minerals or materials of different specific gravity can usually be concentrated on spirals if the heavy particles do not exceed 10 mesh (2 mm) or are not finer than 200 mesh (0.074 mm). The size of the low-specific-gravity component is not critical when the values to be recovered are in the heavier-particle fraction. In this case the size of the light particles may range from 4 mesh (4.76 mm) to zero. The quantity of locked grain components (referred to as middlings in the mineral industry) that are present in a given pulp can be critical because this material is frequently recirculated and may eventually accumulate to a degree that will inhibit the separation operations. One solution to such a problem is continual removal of all or part of the middling stream to a grinding mill, followed by separate recovery of values in another spiral circuit or other concentrating machines.

Examples of good feed materials for spiral concentration are (1) beach sands that are processed for recovery of chromite, ilmenite, rutile, zircon, tin, and iron-ore minerals; (2) hard-rock iron ores in which good liberation of iron values occurs in the 10- to 200-mesh size range; (3) some mica and phosphate ores; (4) tailings from concentrating plants that contain heavy mineral components not recovered by flotation and other concentrating methods; and (5) some fractions of coal [minus 6-mm (¼-in) sizes] that can be upgraded by spiral concentration. The spiral used for coal cleaning has six complete turns with a more gradual slope [a 0.25-m (10-in) pitch]. The six turns require about the same headroom as the conventional five-turn spiral.

The spiral concentrator has shown unusual capability in the gravity processing of tailing streams from conventional magnetic and froth-flotation types of ore-processing plants. There are a number of minerals plants where some of the iron values are first recovered by spirals and the tailings are then sent to magnetic separators. There are also iron plants in which the reverse order of processing is used. In spirals the nonmagnetic iron minerals can be efficiently recovered as a high-grade product. An outstanding example of tailings processing in spirals is illustrated by a Colorado molybdenum-ore treatment plant. Spirals recover salable tungsten, pyrite, and tin concentrates from thousands of tons of flotation-plant waste every 24 h. There is no other known ore-processing method that can economically recover tin and tungsten values from this source. The crude ore contains only 0.03 percent tungstic oxide and a trace of tin. The tin occurs as the mineral cassiterite and the tungsten as the mineral hubnerite.

Operating Characteristics Spiral capacity can range from 0.12 to over 0.5 kg/s (0.5 to over 2 tons/h) of new feed. The grade of concentrate produced can be adversely affected by either too low or too high a feed rate. A good average feed rate for most spiral installations is 1.5 short tons of new feed per hour. The pulp density of spiral feed may range from 10 to 50 percent solids. If the values are contained in coarse heavy minerals, a high pulp density is preferred, whereas if the values are in fine-sized heavy minerals, it is better to use low pulp densities. Generally 20 to 30 percent solids by weight will constitute a suitable pulp feed.

Water Requirement The water requirement per spiral can range from 1.0 to 2.5 L/s (15 to 40 gal/min); this includes 0.2 to 0.6 L/s (3 to 10 gal/min) of water used in the wash-water channel. An attractive feature of the spiral is that reclaimed water can generally be used in all except the very final upgrading step.

Maintenance The only moving parts in spiral concentrators are those in the pumps that supply the feed and recirculate intermediate products. However, there are sometimes minor maintenance problems associated with the spiral trough itself. Some ores contain sharp particles of very abrasive minerals. The presence of these minerals in some ore causes rapid formation of deep grooves in the surface of cast-iron spirals. Wear grooves can be patched with a variety of plastic and metallic cements. Most spirals presently in service are made of cast iron with molded and vulcanized liners. These liners have successfully solved most wear problems.

Other than the wear problems, actual in-plant maintenance usually involves removal of wood, pieces of blasting wire, and other trash from the ports. When a reagentized feed is used, layers of oily reagents can build up on the spiral surface and sometimes require scrubbing for removal. With feeds containing oily reagents that attack rubber, abrasion-resistant alloy spiral sections are used.

Spirals have been manufactured from concrete, plastics, solid rubber, iron, and special iron alloys.

Operating Costs The operating cost of a spiral concentrator plant will be among the lowest costs of any ore-processing plant handling similar feed material. The only moving equipment parts involved are the pumps included in the flow sheet for the purpose of elevating feed and water to the spirals. One large pump can feed 100 or more spirals in a large plant. At a Canadian iron-ore plant twelve 0.14 m^3/s (2200-gal/min) pumps provide feed for 1152 rougher (or first-stage) spirals. In many plants the second- and third-stage spirals are gravity-fed. Thus maintenance is largely limited to pump repair. When unusually abrasive ores are processed, maintenance of worn sections of the spirals can be extensive unless rubber coating or other abrasion-resistant materials are used on the wearing surfaces.

Labor requirements for a spiral plant are low and are governed primarily by the type of material being fed to the spirals. For example, a phosphate ore containing roots, leaves, and other trash will contain sufficient fibrous material to block concentrate ports, and generally prevent good spiral operation. When such an ore is processed, considerably more labor is required for cleaning and adjustments. Generally metallic ores are quite free of fibrous material that will hang up in the spirals, and one person can operate 100 or more spirals.

Power requirements for spiral plants are low, consisting primarily of pumping energy and possibly a thickener or other pulp-handling equipment associated with the flow sheet.

A typical summary of an approximate range of spiral concentration plant direct costs is given in Table 19-11.

TABLE 19-11 Approximate Range of Direct Costs for Spiral Concentration

Cost element	Cents per short ton of spiral feed
Labor	3.0–5.0
Power	1.6–3.0
Maintenance	2.0–3.0
Depreciation	2.8–4.0
Total	9.4–15.0

DENSE-MEDIA SEPARATION

GENERAL REFERENCES: Aplan and Spedden, "Viscosity Control in Heavy Media Suspension," *Proc. 7th Int. Miner. Process. Congr.,* New York, Sept. 20, 1964, Gordon and Breach, New York, 1965, p. 103. Browning, *Heavy Liquids and Procedures for Laboratory Separation of Minerals,* U.S. Bur. Mines Inf. Circ. 8007. Burton, "The economic impact of modern dense medium systems," *Minerals Engineering,* **4,** 225 (1991). Deurbrouck and Hudy, *Performance Characteristics of Coal-Washing Equipment: Dense-Medium Cyclones,* U.S. Bur. Mines Rep. Invest. 7673. Doyle, "The Sink-Float Process in Lead-Zinc Concentration," *AIME Symp. Lead Zinc,* St. Louis, 1970. "Mineral Engineering Techniques," *Chem. Eng. Prog. Symp. Ser.,* **50**(13), (1954). Mular and Bhappu (eds.), *Mineral Processing Plant Design,* 2d ed., Society of Mining Engineers, AIME, New York, 1980. Rodis and Cremer, "Why an Atomized Ferrosilicon?" *Min. World,* **22**(3), 36 (March 1960). Ruff, "New developments in dynamic dense-medium systems," *Mine & Quarry,* **13,** 24 (1984). Tippin and Browning, *Heavy Liquid Cyclone Concentration of Minerals,* U.S. Bur. Mines Rep. Invest. 6969 and 7134. Volin and Valentyik, "Control of Heavy Media Plants," *Pit Quarry,* **62,** 111 (December 1969). Walker and Allen, "Beneficiation of Industrial Minerals by Heavy Media Separation," *Trans. Am. Inst. Min. Metall. Pet. Eng., Min. Branch,* **184,** 17 (1949). Williams and Kelsall, "Degradation of ferrosilicon media in dense medium separation circuits," *Minerals Engineering,* **5,** 1 (1992).

Dense-media separation, also known as heavy-media or sink-float processing, is an adaptation of the common laboratory procedure for separating solids of differing specific gravities by immersing them in a heavy liquid of specific gravity intermediary between those of the solids, thereby causing the lighter particles to float while the heavier sink. However, in dense-media separation, the parting liquid is produced by dispersing relatively fine-grained solids of a high specific gravity in water and maintaining this pulp in suspension by light agitation. The method is very effective and can be used to separate solids with differences in specific gravity of as little as 0.005. It is often the only process needed for the removal of deleterious wastes from coal. The method is used extensively for beneficiating ore minerals, and it is finding increasing use for the processing of shredded automobile scrap and for the recovery of values from solid municipal waste.

Dense-media separation may be used to produce either a finished concentrate or an upgraded feed for subsequent processing. In the latter case, it provides a low-cost means to reject a significant amount of essentially barren waste at a coarse size.

Sink-float plants are usually custom-designed for each individual application. However, for coal beneficiation modular units are available. For most large mineral-processing applications the plants will be permanently located for easy access of feed and disposal of waste, but for smaller coal and aggregate operations the plants are often constructed with the anticipation of relocation when the deposits have been depleted.

The response of any given feed to sink-float processing can be accurately established in the laboratory by testing with various heavy liquids. The liquids generally used for this purpose are listed in Table 19-12. These halogenated hydrocarbons are mutually miscible, which enables the preparation of almost any pulp density attainable in a commercial plant. Heavy-liquid test work provides the basis for specifying the optimum screen size for the preparation of the feed.

Continuous pilot-plant test runs are generally recommended to verify the laboratory results and to establish criteria for plant design. Facilities for these runs are available at a number of minerals-processing research centers.

Feed Preparation and Feed Size The ability to achieve a separation of different solid particles on the basis of density, as in all physical separation, depends on the degree to which the particles are liberated (detached) from each other. Liberation can be achieved by breaking the material in a manner that causes it to fracture and free the individual grains of the constituents to be recovered. The degree of separation that can be realized by the dense-media process will depend on the degree of liberation of the individual grains.

There will be an optimum size reduction of the feed material for the dense-media process. This size range can depend on the overall objectives of its use. For example, if the process is to be used in conjunction with a subsequent separation method such as flotation, the intent may be more the rejection of barren waste material at a relatively coarse size and at high recovery of the values, although the resulting grade from the dense-media operation may still be low. On the other hand, if the concentrate grade must be high, a finer degree of liberation will be needed at some loss in recovery. The initial test work can establish the so-called grade versus recovery limitations of the dense-media operation for the specific material of interest. This testing should recognize that the dense-media process is not effective for treating material which contains any substantial amount of particles smaller in size than about 0.5 mm (20 to 28 Tyler mesh).

The largest size that can be treated depends mostly on the dimensions of the separating vessel; coal up to 0.3 m (12 in) has been successfully processed in a drum separator of the type illustrated in Fig. 19-32. Complete removal of fines is usually necessary to ensure proper viscosity of the media. Fines increase viscosity and slow the separation process.

The feed-preparation screen between crusher and separatory vessel may be of either the revolving or the vibrating type. Wash water is applied only to the feed end of the screen so that the process feed will enter the separator moist but without any free water, which would lower pulp density. In a few instances it has been found advantageous to provide for surge storage between screen and separator to drain off further excess water.

A typical flow sheet is shown on Fig. 19-33.

Preparation of the Media Various solid materials have been used to prepare the media. In the initial development of the process, a suspension of sand and also mixtures of barite and clay were used for separating coal from slate. Galena (lead sulfide mineral) was also used

TABLE 19-12 Liquids Used to Test Feeds

Name	Specific gravity, 25°C
Methylene iodide	3.33
Tetrabromoethane	2.96
Bromoform	2.89
Tribromoethane	2.61
Methylene bromide	2.48
Ethylene dibromide	2.17
Methylene chlorobromide	1.92
Pentachloroethane	1.67
Carbon tetrachloride	1.59
Trichloroethylene	1.46
Ethylene dichloride	1.26

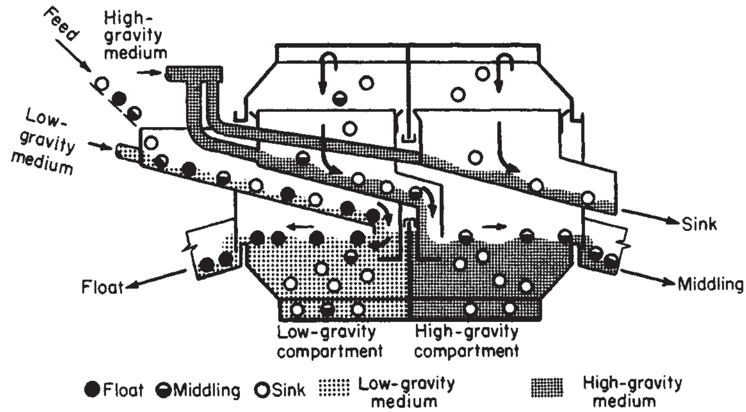


FIG. 19-32 Revolving-drum-type dense-media separatory vessel. (*Courtesy of Western Machinery Co.*)

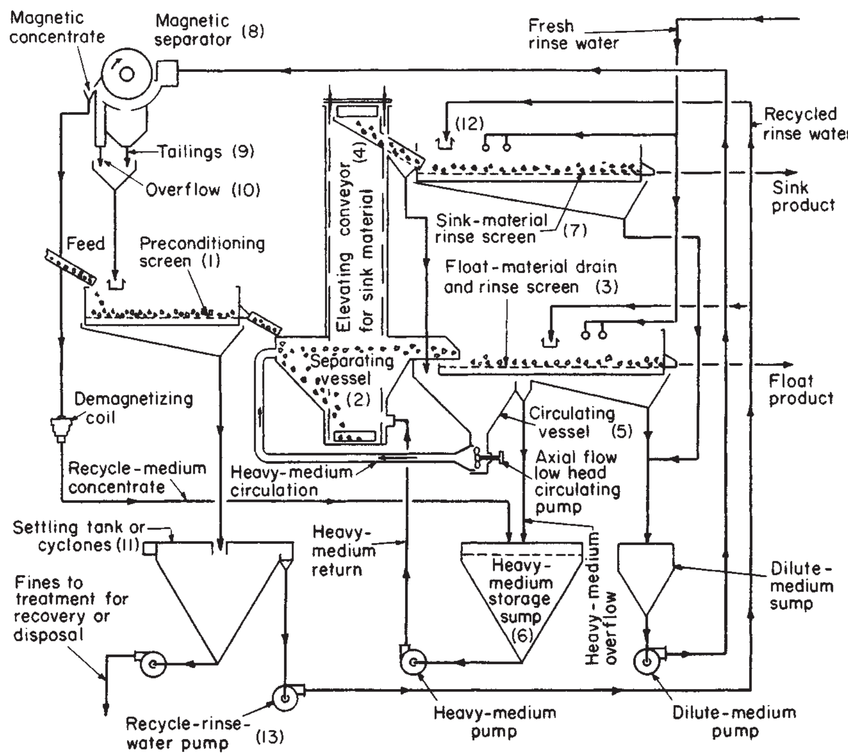


FIG. 19-33 Typical dense-media flow sheet for a coal-cleaning plant. (*Courtesy of Process Machinery Division, Arthur G. McKee Co.*)

to achieve a higher pulp density. In present processing, iron-based particles such as magnetite and ferrosilicon are preferred because they offer suitable density, high resistance to attrition, and ease of recovery by magnetic methods. With magnetite, a pulp density 2.5 times that of water can be obtained. Ferrosilicon can provide a density factor of 3.3, which is effective for separating most gangue constituents from metallic ores. Both materials might be used to obtain intermediate media densities.

Media-Particle Size The size of the media particle is important. A relatively coarse medium (minus 100 mesh) is commonly used in larger-volume static-type separators such as cones. However, in dynamic separators, a much finer size is desirable. Ground magnetite or atomized ferrosilicon is advantageous in this application. The latter is produced by pouring molten ferrosilicon into an atomizing chamber, where a jet of steam forms the alloy into spherical particles. These particles are more resistant to wear than are the particles in a ground product and cause less abrasion to the equipment. Atomized ferrosilicon permits pulp densities above about 3.4-density factor.

Custom-ground natural magnetite is available in many size ranges. Table 19-13 gives a typical specification sheet. Cost is based on truckload quantities in 45.4-kg (100-lb) paper bags, FOB Frazer, Pennsylvania.

TABLE 19-13 Typical Size Distribution of Ground Natural Magnetite*

	Percent retained by weight for mesh size				
Product grade	100	200	325	Less than 325	1978 cost
A	0.6	12.0	17.8	69.6	$72
B	0.1	1.0	7.6	91.4	$74
C	5.0	22.0	23.0	50.0	$72
D	6.0	29.5	22.9	36.3	$71
E	0.1	0.4	1.9	97.7	$85
G	0.2	6.2	15.5	78.1	$74

*Foote Mineral Company.

Pulverized ferrosilicon containing approximately 15 percent silicon is available from the Foote Mineral Company and from Carborundum Co. in the sizes and at the prices shown in Table 19-14. Cost is based on truckload quantities in 227-kg (500-lb) steel drums, FOB Keokuk, Iowa, and Niagara Falls, Ontario.

Atomized ferrosilicon is at present available only from West Germany through American Hoechst Corp. in the sizes shown on Table 19-15. Costs vary with the exchange of U.S. dollars to deutsche marks but will be around $770 per metric ton, FOB Germany (1978 estimate).

Chemical Additives The use of chemical additives in sink-float processing is not common except for the use of lime to prevent oxidation and decomposition of the medium. A small amount of clay is sometimes added to improve the kinetic stability of the suspension.

Considerable laboratory work has indicated that the use of a dispersant such as sodium hexametaphosphate may assist in the stabilization of the medium; more recent data report the beneficial effect of the addition of polymers that reduce media viscosity while simultaneously producing a very low settling rate of the ferrous compound. This should be of great value for difficult separations, but at present no data are available from commercial operations.

Separating Vessels Many different types of separating vessels have been proposed and used for sink-float separation. For applications at coarse sizes involving a high ratio of float to sink or a high gravity differential, as in the case of coal, trough-shaped vessels (as shown in Fig. 19-34) or rotating-wheel separators are commonly used. However, for the beneficiation of most ores, other types of separators have found general acceptance. The optimum design for any given ore will depend on such variables as the rate of feed, the size of feed particles, the ratio of float to sink, and the gravity differential between the solids to be separated. Separators are classified as either static or dynamic, depending on whether or not centrifugal force is applied.

TABLE 19-14 Typical Size Distribution of Pulverized Ferrosilicon

Particle size; mesh size less than	1978 cost range
48	$245–279
65	$248–282
100	$252–291
200	$289–400

Drum Separators Very coarse solids, up to 0.3 m (12 in), are often processed in a drum separator of the type shown in Fig. 19-32. This is similar to a ball-mill shell with lifters permanently attached to the wall. Medium and feed enter at one end, and the float product flows out through the discharge trunnion, while the sink is lifted by the rotation of the drum to a stationary launder, through which it is flushed out. Modifications of this type include division of the shell into two compartments, which permits simultaneous operation at two different pulp densities resulting in various grades of products. The two-compartment revolving drum is illustrated in Fig. 19-32.

Drum separators have capacities up to 250 kg/s (900 t/h), cone separators to about 125 kg/s (500 t/h), and dynamic separators a maximum of 28 kg/s (100 t/h); however, these can readily be manifolded for any required tonnage. Economics will dictate the minimum tonnage for which a plant would be justified; several plants of 2.8-kg/s (10-t/h) capacity have been built.

Cone Separators Feed materials in the intermediate sizes, 0.1 to 0.01 m (4 to 0.5 in), may be processed in cone separators as shown in Fig. 19-35. These have a large surface area and increased volume pulp, which permit longer retention time than that of most other types of separators; this is a great advantage when separating solids of small gravity differential. The feed is introduced into the cone at a point below the pool surface and as far from the overflow baffle as possible. Slow-moving scrapers prevent a buildup of medium on the cone wall, provide the necessary agitation to prevent settling of the medium, and push the float particles toward the overflow weir. The sink product is removed from the cone bottom by rock pump, internal air lift, or external air lift. Mechanical elevators of the screw or bucket type have also been used but require more maintenance.

Cyclone Separators Finer feed solids, from 0.04 to 0.0005 m (1.5 in to 28 mesh), may be treated in dynamic separators of the Dutch State Mines cyclone type (Fig. 19-36). In cyclone separators, the medium and the feed enter the separator together tangentially at the feed inlet (1); the short cylindrical section (2) carries the central vortex finder (3), which prevents short circuiting within the cyclone. Separation is made in the cone-shaped part of the cyclone (4) by the action of centrifugal and centripetal forces. The heavier portion of the feed leaves the cyclone at the apex opening (5), and the lighter portion leaves at the overflow top orifice (6).

The sharpness of separation of the mineral from the gangue is dependent on (1) the stability of the suspension, which is influenced by the size of the medium; (2) the specific gravity of the medium; (3) the cleanliness of the medium; (4) the cone angle; (5) the size and ratios of the internal openings in the cyclone (inlet, apex, and vortex); and (6) the pressure at which the pulp is introduced into the cyclone. A 20° cone angle is the most common. Cyclone diameter will be determined by the separation to be made as well as by the capacity required. The 0.5- and 0.6-m (20- and 24-in) cyclones are most common in coal plants, whereas multiple cones of 0.25- or 0.3-m (10- or 12-in) diameter are used in higher-gravity separations.

Dense-media cyclones are generally operated in the $(0.7–1.0) \times 10^6$·Pa (10–15-lbf/in^2) range. It is not advisable to go below $(0.4–0.56) \times 10^6$ Pa because the recovery of low-specific-gravity material and the rejection of impurity are improved at higher pressures,

TABLE 19-15 Size Distribution of Atomized Ferrosilicons

Particle size, greater than Tyler mesh	Manufacturers' grade distribution					
	Extra coarse	Coarse	Fine	Cyclone 60	Cyclone 40	Cyclone 20
48	15–0	5–0				
65	25–7	8–2	4–0			
100	42–20	22–7	8–2	3–0		
150	55–35	35–17	22–5	8–0	3–0	
200	70–50	50–35	30–15	20–5	8–3	2–0
Particle size, less than Tyler mesh						
200	30–50	50–65	70–85	80–95	92–97	98–100
325				40–60	70–85	90–100
625				10–20	40–55	70–80

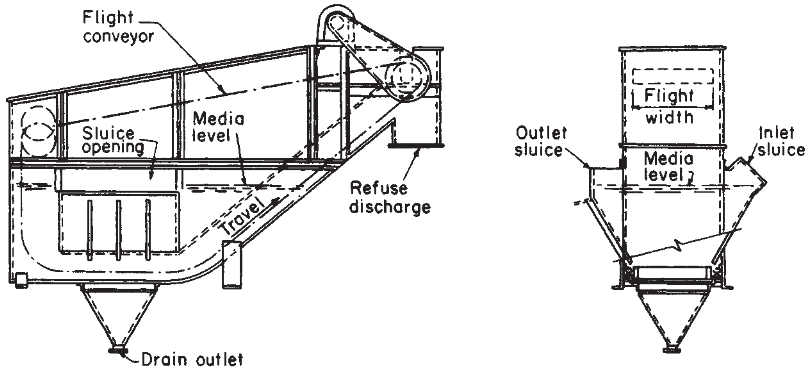


FIG. 19-34 Drag-tank-type dense-media separatory vessel. (*Courtesy of Link-Belt Co.*)

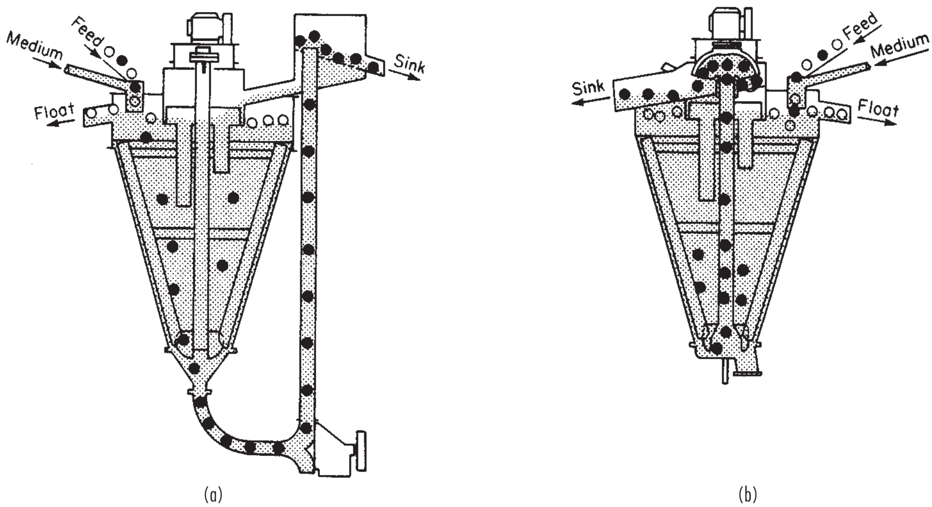


FIG. 19-35 Dense-media cone-vessel arrangements. (*a*) Single-gravity two-product system with pump sink removal. (*b*) Single-gravity two-product system with compressed-air sink removal. (*Courtesy of Process Machinery Division, Arthur G. McKee Co.*)

especially for the finer sizes. Pressures as high as 2.5×10^6 Pa (36 lbf/in^2) have been used, and they increase capacity but accelerate wear. Residence time of the ore particles is very short in the cyclone, and a large volume of medium is circulated for each ton of feed treated in the cone. Loss of media is higher in cyclone plants because of the finer media required and the additional volume encountered in these plants. Media loss may be 2 to 5 kg/ton (5 to 10 lb/ton) of ore treated in cyclone plants, as compared with 0.2 to 0.8 kg/ton (0.5 to 1.5 lb/ton) in coarse, static heavy-medium circuits. Cyclone-plant labor requirements are low and efficiency is high. A 0.6-m (24-in) heavy-medium cyclone can handle 75 tons of coal per hour.

Dyna Whirlpool A unique vessel design for capacities up to 100 t/h has been developed by the American Zinc Co. The separation occurs in a cylindrical-shaped separatory vessel maintained in an inclined position from horizontal. This system, known as the Dyna Whirlpool (DWP) process, provides for separate entry of the medium and the feed solids, as illustrated in Fig. 19-37. A distinct feature of this separator is that the feed enters the separator via gravity flow. Feed size may range from 0.05 to 0.0002 m (2 in to 65 mesh). Magnetite or ferrosilicon is generally used.

Process Control As is the case in all concentration processes, optimum results will be obtained under steady operating conditions. Because of the simplicity of the dense-media process, these can readily be maintained.

Uniformity of the rate of feed will be ensured by a constant-weight feeder; density control may be automatically obtained through a measuring probe on the media-return line that adjusts delivery of the necessary volume of media from the densifier or media thickener; the viscosity can be controlled automatically by continuously testing a predetermined volume of return media and adjusting the divider under the drainage screen for media cleaning as needed; pH control can be automated by conventional methods.

Notwithstanding the possibility of such automation, many successful operations depend almost entirely on manual sampling. Density determinations of the pulp on the media-return line and on each of the drainage screens are made at scheduled intervals, and the operator adjusts the media flow as needed.

Costs Because sink-float processing is applied to relatively coarse particles and is a single-pass operation, capital and operating costs are usually considerably lower than would be required for a flotation or a gravity mill of the same capacity. A large flow of water is required for feed preparation and for media recovery, but almost total recovery for recirculation is possible. A minimum of two job-trained operators per shift is generally required by law, but these would be able to attend several separators at almost any feed rate.

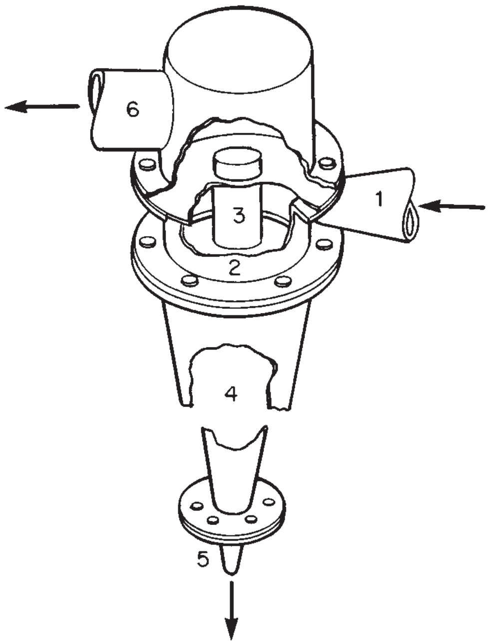


FIG. 19-36 Dutch State Mines cyclone separator.

Estimates for a 30-kg/s (100-ton/h) plant using a dynamic separator is approximately \$350,000 (1978), exclusive of power, water, compressed air, crushing, foundations, and housing; installed power for such a plant will be about 298 kW (400 hp).

Direct operating costs usually vary between 30 cents and \$1 per ton of feed, depending on hourly tonnage and on media recovery. Media losses are usually higher in a dynamic than in a static separator, mainly because a finer particle size is required. Media loss may be from 2 to 20 kg/t (5 to 10 lb/ton) of mixture processed in a cyclone plant compared with a static plant, which generally operates with media losses of an order of magnitude lower.

MAGNETIC SEPARATION

General References: Kolm, Oberteuffer, and Kelland, "High-Gradient Magnetic Separation," *Sci. Am.,* **233**(5), 46–54 (November 1975). Lawyer and Hopstock, "Wet Magnetic Separation of Weakly Magnetic Materials," *Min. Sci. Eng.,* **6**(3), 154–172 (July 1974). Marston, "The Use of Electromagnetic Fields for the Separation of Minerals," World Electrical Congress, Moscow, 1977. Taggart, *Handbook of Mineral Dressing,* 2d ed., Wiley, New York, 1945.

The principles of magnetic separation have been applied commercially for nearly 100 years. Applications range from the removal of coarse tramp iron to more sophisticated separations, such as the elimination of weakly magnetic iron-stained particulates from paper-coating clays. The application of magnetic-separation methods to weakly magnetic particles has been made possible by recent advances in separator design. Magnetic separators now have a great many industrial applications and range in size from small laboratory-scale devices to those capable of processing hundreds of tons hourly.

Selecting the best separator for a specific application requires an understanding of basic principles of magnetism plus an evaluation of separator capability on the basis of design and application variables such as type of material to be processed, wet or dry processing, particle-size range, magnetic characteristics of the feed, desired throughput rate, etc.

Principles of Magnetic Separation Any particle introduced into a magnetic field will become magnetized to some extent and act as a magnetic dipole. Depending on the magnetic characteristics of the material, it can be classified as ferromagnetic, paramagnetic (magnetically attracted), or diamagnetic (repelled by a magnetic field). Ferromagnetic substances (e.g., iron, nickel, and cobalt) may be permanently magnetized and have strong magnetic moments per unit volume. Paramagnetic substances are further classified as strongly or weakly magnetic according to the strength of the magnetic moment produced per unit volume in the external magnetic field.

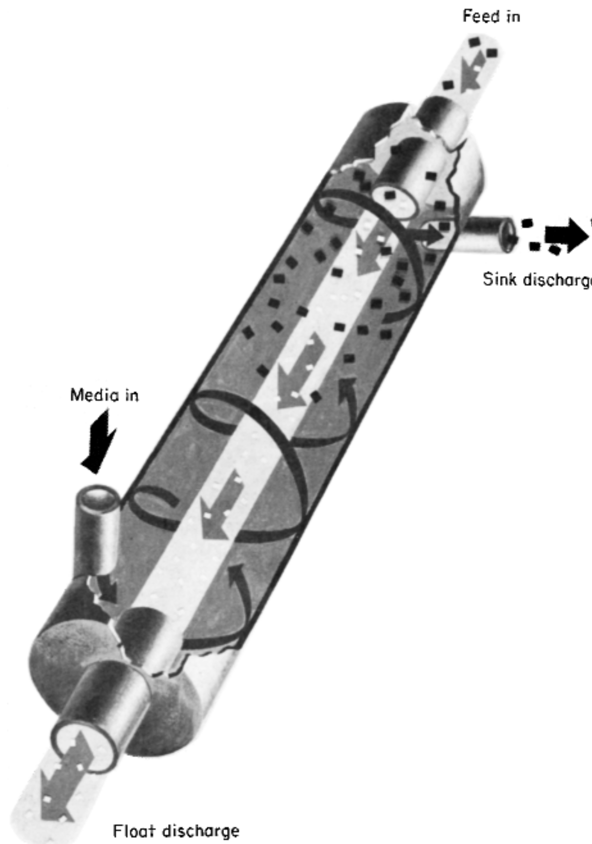


FIG. 19-37 Dyna Whirlpool separator. (*Courtesy of American Zinc Co.*)

A magnetic field and magnetic-field gradients are produced in a variety of ways and vary in both field geometry and strength. The magnetic field of a magnet is the space through which its influence extends. It is mapped by the lines of magnetic force. A magnetic field is considered uniform or homogeneous when these lines are parallel and equally spaced. It can be noted in Fig. 19-38*a* and *b* that neither the bar (permanent) magnet nor the coils plus iron magnet typical of the C-frame magnet type can produce a uniform magnetic field.

The intensity of the magnetic field H is measured in amperes per meter. For a single-layer solenoid, at any point along its axis the magnetic field intensity is

$$H = \tfrac{1}{2}NI(\cos\theta_2 - \cos\theta_1) \tag{19-8}$$

where H is measured in amperes per meter, A/m; N is the number of turns per unit length or the number of turns per meter; I is the current per turn in amperes, A; θ_1 and θ_2 are the angles included between the axis and the lines drawn from the measured point to its near and far edges.

The magnetic flux density is

$$B = \mu_0(H + M) \tag{19-9}$$

Flux density is calculated as the permeability of free space times the sum of the magnetic-field intensity and the induced magnetization

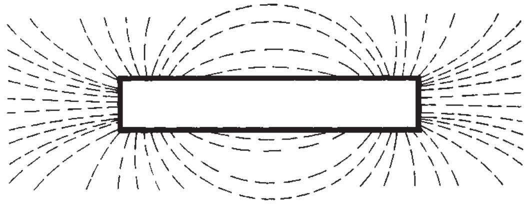


FIG. 19-38a Lines of force surrounding a bar-type magnet.

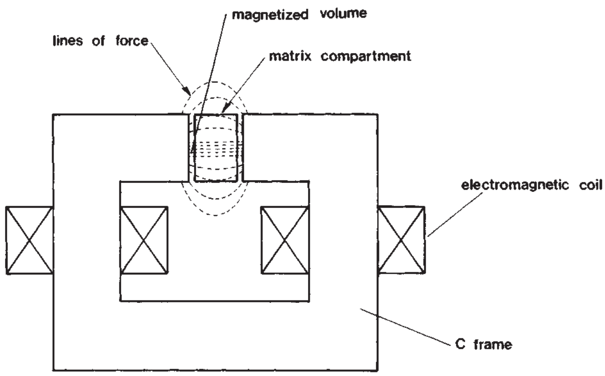


FIG. 19-38b Lines of force produced by a C-frame magnet (coils and iron-magnet surface).

observed whenever a magnetic material is placed in a magnetic field; it is measured in teslas, T.

The strength of the induced magnetization is equivalent to M dipoles per cubic meter where μ_0 is the permeability of free space, equal to $4\pi \times 10^{-7}$, N/A^2; and M is the magnetization, A/m.

Another method for calculating B is

$$B = \mu H \tag{19-10}$$

where μ is the permeability of the material.

The magnetic susceptibility of a material (χ, volume susceptibility) is dimensionless and is defined as the ratio of induced magnetization to magnetic field intensity. It is expressed as

$$\chi = M/H \tag{19-11}$$

Thus,

$$B = \mu_0 H(1 + \chi) \tag{19-12}$$

Specific magnetic susceptibility (ψ) is

$$\psi = \chi/\rho \tag{19-13}$$

where ρ is material density.

Paramagnetic substances have positive susceptibilities, and induced magnetization augments the magnetic-flux density within the substance. Diamagnetic materials have negative susceptibilities, and an induced field in this case cancels part of the magnetic-field intensity.

Permeability (μ), which is often used, albeit imprecisely, in referring to ferromagnetic substances, is the ratio of the magnetic-flux density to the magnetic-field density.

$$\mu = B/H \tag{19-14}$$

Relative permeability is μ/μ_0.

In cgs units,

$$B = H + 4\pi I \tag{19-15}$$

$$K = I/H \tag{19-16}$$

$$K = \chi/4\pi \tag{19-17}$$

Table 19-16 shows the magnetic susceptibility of minerals and elements. Magnetization of various materials is directly dependent on two factors: (1) the degree of magnetic susceptibility and (2) the applied magnetic-field intensity. It can be seen in Fig. 19-39 that ferromagnetic materials quickly become magnetically saturated and that an increase in magnetic-field intensity will have no effect after a certain point. For paramagnetic materials (e.g., hematite), which are more difficult to magnetize, the magnetic-flux density is directly proportional to the magnetic-field intensity, and some of these substances, practically speaking, cannot be saturated.

In addition, the magnetic characteristics of a material can change as a function of stress (e.g., unannealed series 316 stainless steel can be magnetic after machining), temperature, pressure, and physical and chemical treatment. Therefore, when two paramagnetic materials with similar magnetic susceptibilities are to be separated, the possibility that pretreatment will facilitate subsequent separation should be studied.

A magnetic field exerts a force on each of the two poles of a dipole (particle), forcing it to align itself with the lines of magnetic force.

TABLE 19-16 Magnetic Susceptibility of Elements and Minerals

Substance	Susceptibility, 10^{-6} cgs	Substance	Susceptibility, 10^{-6} cgs
Aluminum	+10.5	Ferberite	+39.3
Al_2O_3	−37.0	Galena	−0.4
Apatite	+1.0 to +18.0	Garnierite	+30.7
Aragonite	−0.4	Gold	−28.0
Asbolan	+150.0	Ilmenite	+15.45 to 70.0
Azurite	+12.2 to +19.0	Lead	−23
Anatase	+0.96 to +5.60	Malachite	+10.5 to +14.5
Beryl	+0.4	Millerite	+0.21 to +3.85
Braunite	+35.0 to +150.0	Molybdenite	+4.93 to +7.07
Biotite	+40.0	Molybdenum	+89.0
Barite	+10.0	Platinum	+201.9
Barite (pure)	−71.3	Rutile	+0.85 to +4.78
Brannerite	+3.5	Scheelite	+0.13 to +0.27
Chromium	+180.0	Siderite	+65.19 to +103.81
Chromite	+125.6 to +450.0	Titanium	+150.0
Cobalt	Ferromagnetic	Tungsten	+59.0
Cobaltine	+2.0	Uranium	+395.0
Cobaltite	+0.34 to +0.64	Vanadium	+255.0
Columbite	+32.55 to +37.20	Vanadinite	−0.2 to +0.27
Copper	−0.1	Wolframite	+42.2
Chalcopyrite	+1.0 to +5.0		

NOTE: Extensive listing of magnetic susceptibilities of elements and organic and inorganic compounds can be found in G. Foex, *Tables de constantes et donnes numériques,* Massou et Cie., Paris, 1957.

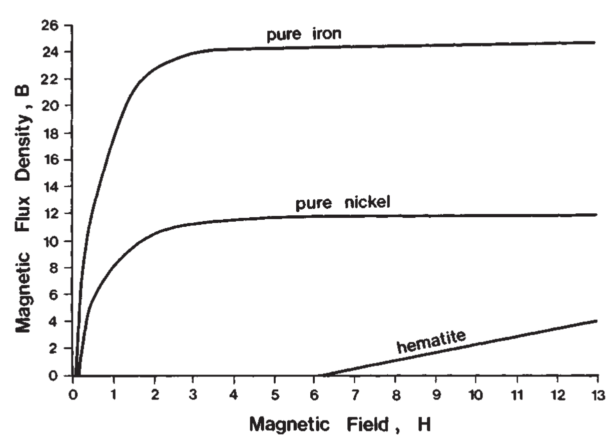


FIG. 19-39 Magnetization curves for ferromagnetic and paramagnetic materials.

These are exerted in opposite directions, and if the magnetic field is uniform, they will be equal. Therefore, the net force on the dipole will be zero. However, if the field varies in space (has a gradient), the force on the dipole will be greater in the direction of the higher field and will be proportional to the magnetic-dipole moment and the magnitude of the magnetic-field gradient.

A model of the forces operating in such a case is shown in Fig. 19-40, where

$$F_m = m\, dB/dz \tag{19-18}$$

$$F_d = 3\pi\eta b v \tag{19-19}$$

where
- F_m = magnetic tractive force
- F_d = hydrodynamic drag force
- F_g = gravitational force
- m = magnetization characterization of the particle ($m = \chi HV$)
- H = magnetic-field intensity
- dB/dz = magnetic-field gradient
- η = fluid viscosity
- b = particle diameter
- v = fluid velocity
- V = particle volume
- χ = magnetic permeability of particle

In order to retain the magnetic fraction of the material in the collection volume of the separator it is necessary that

$$F_m \geq F_d + F_g \tag{19-20}$$

The preceding analysis shows that density is important and cannot be influenced and that particle size is of extreme importance, as the magnetic force F_m is directly dependent on the b^3 of the particle while the drag force F_d is dependent on b. This means that separations between particles with close magnetic susceptibilities can be successfully performed only if the particle-size distribution is within a relatively narrow range. The influence of gravitational force is dependent on the relative direction of the slurry flow. [Buoyant forces are neglected in the relationship expressed in Eq. (19-20).]

It should be noted that effective magnetic separation requires that particles of different species be liberated from each other. It is also important that the finest possible matrices (filamentary type) be used because these produce the highest magnetic-field gradients. The use of high-gradient-producing matrices can substantially reduce the magnetic-field intensity (magnet strength) required to gain the same separation results, thus lowering both capital investment and process costs. The most economical results are achieved when the diameters of the matrix filaments are matched to the size of the particulates being processed.

Factors which adversely influence the separation of very fine particle systems are brownian motion and London forces. However, it is possible to counter these forces by the use of dispersants, temperature control, and so on.

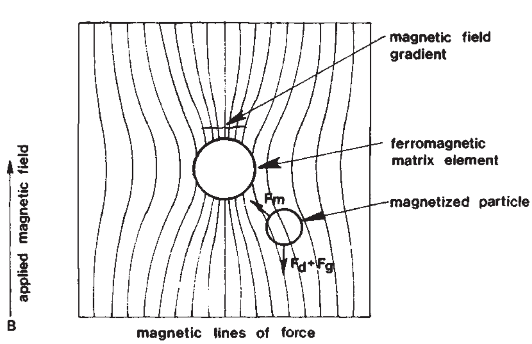


FIG. 19-40 Model of particle-capture forces.

Equipment Separator designs differ for the various types of materials to be separated. In general, magnetic separation devices can be grouped as follows:

Grate-Type Magnets This type of device consists of a series of tubes (often of stainless steel) which are packed with ceramic magnets and installed in a trap perpendicular to the fluid-flow direction. Grate magnets are used for the wet or dry removal of tramp coarse or fine iron. The various available equipment designs include self-cleaning grates, wing-and-drawer-type magnetic grates, permanent magnetic grates, vibratory grates, and rota-grates. Each of the designs is manufactured in a range of sizes, with single or multiple rows (banks) of magnetic tubes. Applications include ferrous traps for slurries such as detergents (e.g., in chemical plants), sugar and candy (e.g., in food plants), ink recycling (e.g., in printing operations), or pulp in paper mills.

Grates may be installed in all circuits of dry, pulverized material where contamination or accidents may occur from tramp or fine iron.

Plate Magnets and Magnetic Humps These devices are used to remove tramp iron from materials being conveyed pneumatically or falling in gravity flow. Tramp iron is removed by being trapped against a magnetized plate. This type of magnet must be cleaned periodically. A chute angle of 45° is recommended. The plate magnet should be close to the feed point to eliminate the influence of velocity. Complete lines of plate magnets and magnetic humps are offered by many manufacturers.

Plate magnets, which are used in chutes, can be either permanent magnets or electromagnets. For the permanent type, magnet width extends to 1.23 m; there is a maximum width of 2.85 m for electromagnets. Capacities are approximately 250 m^3/h for each meter of width, at a 45° angle, with the magnet located in the bottom of the chute approximately 0.6 m from the top and material introduced at a slow velocity. Capacity varies with the size of the tramp-iron particles to be removed and with the angle of the chute (see Table 19-17).

Lifting Magnets These devices operate in either a continuous or a cyclic manner. Continuous devices usually have a belt which moves over the lifting magnetic poles to carry the magnetized particles into a region of low or zero magnetic field, where they are released. Depending on the design of the poles, these units can be either high- or low-intensity devices. Figure 19-41 shows the in-line and cross-belt methods for installing a lifting magnet above a conveyor belt.

Cross-belt magnetic separators are based on the same principle as lifting magnets. Although these units have relatively low capacities, the same unit can produce selective separations with different products by using different pole gaps and field strengths. (See Fig. 19-42.)

The magnet designs shown in Figs. 19-41 and 19-42 are used for tramp-iron removal. Suspended magnets are positioned from 5 to 10 cm above the highest point of the material on the conveyor and may be designed to be self-cleaning. Sizes for devices of this type range up to 2.8/1.6 m. The installation shown in Fig. 19-41*a* is often preferred because it requires a less powerful magnet and can clean material from a higher-speed conveyor belt (over 1.75 m/s). For self-cleaning units, the belt is run at up to 2.5 m/s.

Drum and Pulley Magnets Since Thomas Edison invented and developed the magnetic pulley for the concentration of nickel ore, drums and pulleys have become the most common types of magnetic separators. These devices can be built with either a permanent magnet or an electromagnet, and the drum separator can operate with

TABLE 19-17 Tramp-Iron Removal with Plate Magnet, 0.6 m from Top*

Particle size	Relative capacity, percent		
	Chute angle, 35°	Chute angle, 45°	Chute angle, 60°
Over 30 g (1 oz)	125	100	75
Over 8 mesh (2.38 mm) to 30 g	100	75	45
Under 8 mesh (2.38 mm)	33	25	10

*Courtesy of Eriez Magnetics.

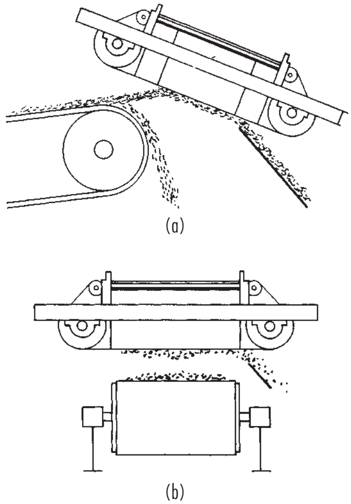


FIG. 19-41 Types of lifting magnets. (*a*) In-line lifting magnet. (*b*) Cross-belt lifting magnet. (*Courtesy of Eriez Magnetics*.)

either dry or wet feeds. Figure 19-43 is a schematic for mounting a magnetic pulley.

Dry magnetic drums can be designed to perform as lifting magnets or pulleys. Magnetic drum devices have stationary magnets; pulley drums rotate. Other schematics of possible arrangements are presented in Fig. 19-44.

In the drum-separator category, several specialized devices are worthy of mention.

Alternating-polarity drum separator. This device is used for the treatment of coarse material (minus 40 mm, plus 0.15 mm) containing strongly magnetic particles when a high-grade concentrate is required. The capacity of this device varies with feed-particle size, up to 100 t/(h·m).

Unigap drum separator. This device is used for materials finer than 6 mm at feed rates of up to 10 t/(h·m).

High-speed, low-intensity drum magnetic separator. This device is designed to handle very fine material (minus 0.15 mm and finer) to produce a high-grade magnetic concentrate.

Depending on the required results—high recovery of magnetics or high-grade concentrates (clean magnetics)—wet drum separators are designed to work in concurrent, countercurrent, or counterrotating fashion by using one or more drums in any possible combination. Figure 19-45 presents schematics of these wet drum magnetic separators.

Magnetic pulleys. These vary in size from 0.203 to 1.219 m in diameter and from 2.03 to 1.526 m in width. The acceptable depth of the material on the conveyor belt depends on the diameter of the pulley and the linear velocity of the belt (see Table 19-18). Table 19-19 indicates the maximum capacity for such units. Depending on the application, the correction factors given in Table 19-20 should be applied. For sizing and maximum efficiency, multiply the actual volume of material to be handled by the correction factor shown and select the magnetic pulley having a capacity equal to or greater than the resultant volume.

Wet Drum Magnetic Separators These devices are used for the concentration of strongly magnetic coarse particles. The size of the separator is influenced by several variables: slurry volume, percent solids in the slurry, percent magnetics in the slurry, required recovery of magnetic particles, and required concentration of magnetic product. This type of separator is built by several manufacturers; drum sizes range from 0.023 to 1.2 m in diameter, with widths up to 3.0 m. The concurrent type can process slurries with 20 percent solids by weight for single-drum separators and with 35 to 45 percent for units with two drums. Recommended maximum particle size is 6 mm (¼ in), but with special tanks these devices can handle even coarser material. Countercurrent-type separators can handle particles finer than 0.8 mm (20 mesh) and obtain optimum results with slurries containing about 30 percent solids. This design has the advantage of being able to handle wide fluctuations in throughput. The counterrotating separator is recommended for applications in which recovery is more important than grade. This unit can handle particles up to 3 or 4 mm (⅛ in) in size, but with less satisfactory results for particles finer than 0.5 mm and slurries containing 30 to 40 percent solids by weight. Figure 19-46 shows a gauss (tesla) chart for a 1.2-m-diameter wet drum separator. The influence of drum diameter in separation is shown in Table 19-21 and in Fig. 19-47. These data result from the processing of a partially martised magnetite ground to 75 percent minus 0.044 mm (325 mesh). The influence of drum diameter and grind in separator capability is summarized in Table 19-22. Figure 19-48 shows the influence of drum diameter on investment. The **installed cost** of a wet single-drum magnetic separator can vary between $25,000 and $75,000 per meter of magnet width, depending on the design, dimensions, and manufacturer. Multiple-drum costs increase in about direct proportion to the number of drums required. **Maintenance costs** per year vary between 3 and 5 percent of the initial investment.

Induced-Roll Separators These devices, which have been in commercial use since 1890, handle only dry, granulated material. They are similar to drum separators, with the difference that the cylinder rotates in the gap of an electromagnet. Magnetic-field gradients are obtained by creating sharply edged ridges on the surface of the cylinder or by constructing a cylinder of alternate magnetic and non-

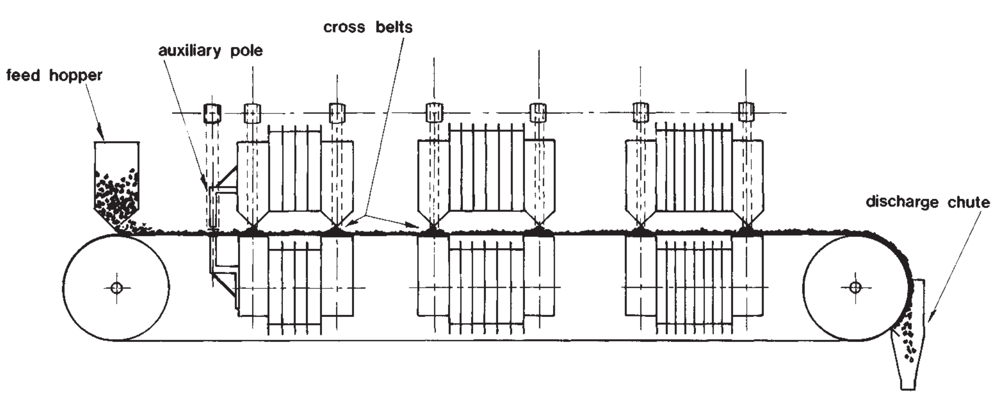


FIG. 19-42 Six-pole, seven-cross-belt magnetic separator. (*Courtesy of Readings, Inc.*)

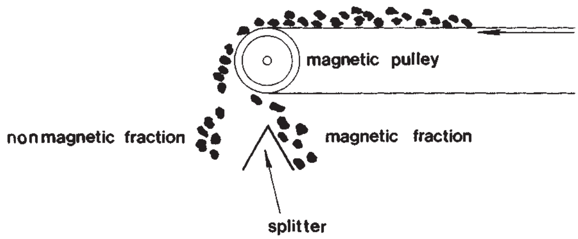


FIG. 19-43 Magnetic pulley.

magnetic disks. A schematic of an induced-roll separator is shown in Fig. 19-49. The best particle-size distribution for separation is minus 2 mm, plus 0.074 mm (minus 10 mesh, plus 200 mesh). Industrial devices are built with multiple rolls, which operate either in a series or in parallel, and can be used as concentrators or as purifiers (see Fig. 19-50). Standard widths for the rolls are 0.25, 0.5, and 0.75 m. Capacities vary between 1.5 and 18 t/(h·m). Induced-roll separators are used only to process weakly magnetic materials. **Capital costs** for this type of device are relatively low compared with those of other high-intensity magnetic separators, but **total process costs** are high owing to moisture-free feed requirements. A wet-process induced-roll separator was developed in the U.S.S.R. during the early 1960s and is reported to have a capacity of up to 100 t/h. In 1964 an Australian manufacturer introduced a wet-type induced-roll separator designed with a laminated, grooved rotor that rotates around a vertical axis (pole). These devices are built with up to 10 poles and are used principally to concentrate ilmenite sands. Capacity is approximately 0.8 t/h per magnetic pole.

Separations similar to those obtained with dry induced-roll devices can be obtained with cross-belt separators (Fig. 19-42). These units are built with up to eight poles, each of which can operate at different magnetic-field intensities to allow simultaneous production of different concentrates. However, capacity is low, and installed costs per ton capacity are high compared with induced-roll units.

Induced-Pole Separators In devices of this type, magnetic-field gradients are produced by the application of background magnetic field to a ferromagnetic matrix, thereby inducing magnetic poles around matrix edges. The correlation of edge and field directions

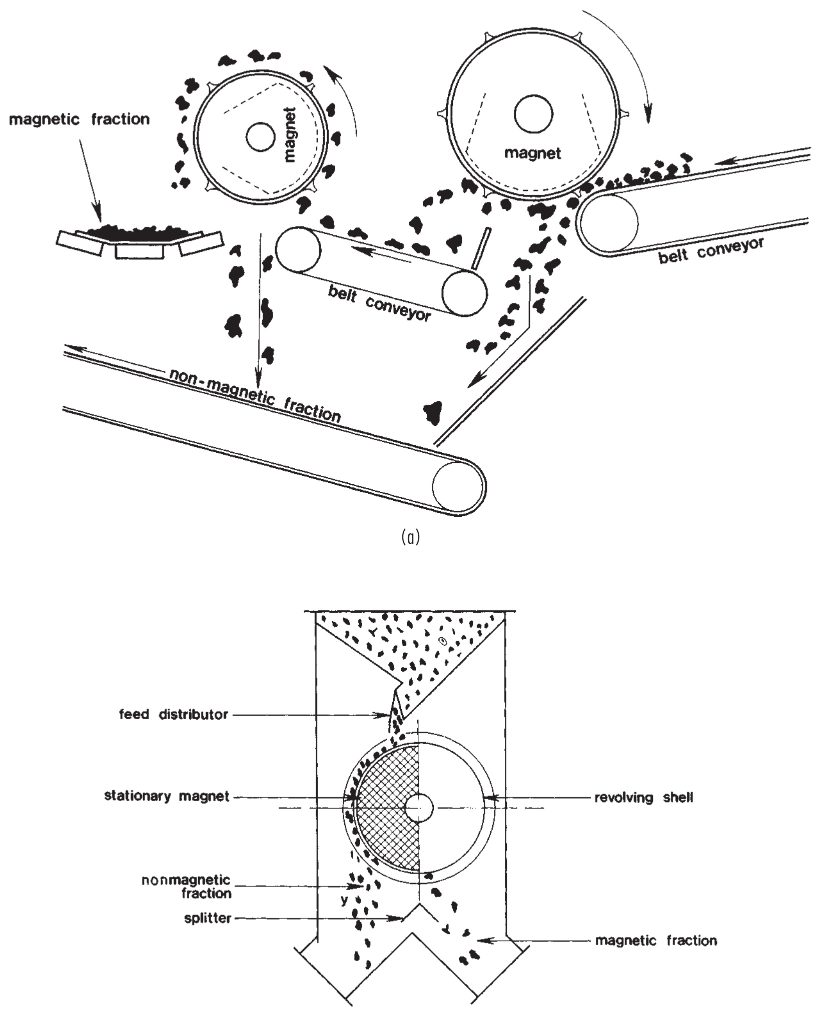


FIG. 19-44 Arrangement of magnetic drum separators. (*a*) Magnetic drum operating as a lifting magnet. (*b*) Magnetic drum operating as a pulley. (*Adapted from design courtesy of Eriez Magnetics.*)

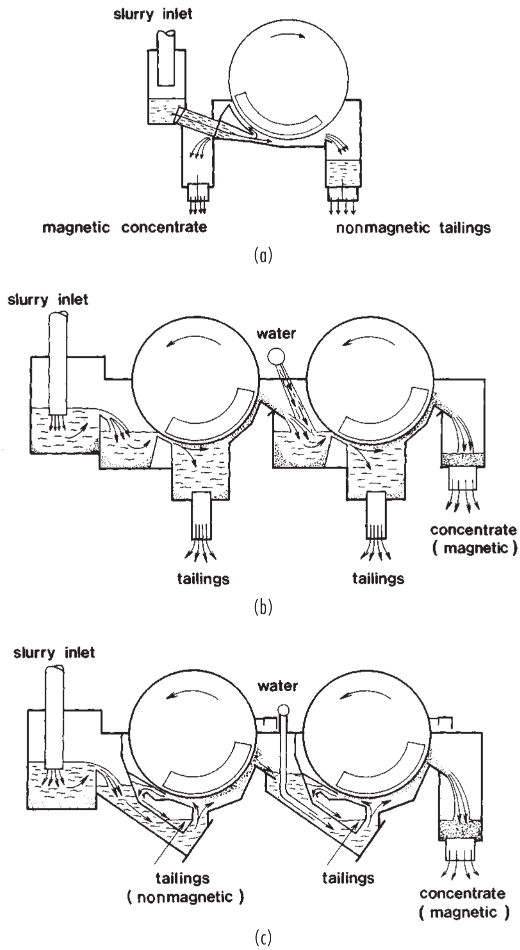


FIG. 19-45 Wet-drum-magnetic-separator arrangements. (*a*) Counterrotation-type wet magnetic drum separator. (*Courtesy of Sala International, Inc.*) (*b*) Concurrent-type wet magnetic double-drum separator. (*c*) Countercurrent-type wet magnetic double-drum separator.

determines whether the separator is a parallel field-to-flow unit or a perpendicular field-to-flow unit. In this category there are only two practical types of separator constructions, C-frame and solenoid. Uniformity of background magnetic field depends on design.

1. *C-frame magnets.* As shown in Fig. 19-38*b*, these magnets employ a ferromagnetic matrix placed between the poles of an electromagnet. With this design, however, the background magnetic field is not uniform. Also, high-magnetic-fringe fields are usually noticed in the flush region of these separators, and these can cause possible matrix clogging when even relatively small amounts of ferromagnetic particles are present in the slurry. The ferromagnetic material used to transfer the magnetic-flux lines from pole to pole occupies between 40 and 80 percent of the magnetized volume.

2. *Solenoid magnets.* These devices can be designed for wet or dry feeds. Depending on design, they can have a relatively uniform background magnetic field. It can be noted from Fig. 19-51 that the use of the return frame is important for generating a uniform mag-

TABLE 19-18 Maximum Depth of Material for Separator by Magnetic Pulley Based on Pulley Diameter and Linear-Velocity Belt

Diameter of pulley, mm	Belt linear velocity, m/s	Depth of mterial, mm
203	0.584	38
305	0.890	70
380	1.017	89
508	1.271	121
610	1.448	140
762	1.678	165
914	1.855	191
1067	2.033	210
1219	2.211	235

TABLE 19-19 Maximum Capacity for Magnetic Pulley Separator*

Pulley diameter, mm	Belt width, mm	Belt velocity, m/s	Capacity, m^3/h
	203		12.2
203	406	0.585	24.9
	610		62.3
	914		133.0
	305		38.0
	406		50.0
381	610	1.017	113.0
	914		255.0
	1219		515.0
	305		47.0
	406		59.0
457	610	1.143	130.0
	914		300.0
	1219		623.0
	406		88.0
	610		170.0
610	914	1.448	374.0
	1219		755.0
	1524		1133.0
	457		153.0
	610		218.0
914	914	1.855	481.0
	1219		935.0
	1529		1500.0

*Courtesy of Eriez Magnetics.

TABLE 19-20 Correction Factors for Magnetic-Pulley Capacities*

Type of application	Type of tramp iron to be removed	Correction factor
Crusher and primary-mill protection	Large and medium, over 30 g (1 oz)	1.0
Secondary-mill, pulverizer, and general separation	Large, over 30 g	1.0
	Medium, 30 to 240 g	1.3
	Small, 8 mesh (2.38 mm) to 30 g	2.0
Product purification	Fine ferrous contamination; finer than 8 mesh (2.38 mm)	4.0

*Courtesy of Eriez Magnetics.

TABLE 19-21 Influence of Magnetic-Drum Diameter on Separation*

Separator diameter, m	No. of stages	Composition of concentrates		Fe recovery, %	Feed rate, t/(h·m)
		SiO_2, %	Fe, %		
0.600	6	1.1–1.3	70.0	92	10–12
0.916	6	0.9–1.0	70.0	98	28–33
1.200	4	0.9–1.0	70.0	98	62–85

*Courtesy of Sala International Inc.

TABLE 19-22 Influence of Drum Diameter and Grind on Separator Capability*

Feed			Recommended capacities, t/(h·m)		
			Diameter of drum, m		
Description	Percent of feed minus 74 μm	Separator arrangement	0.60	0.90	1.20
Coarse	15–25	Concurrent	15–25	70–90	120–160
Medium	50	Concurrent or full countercurrent	10–15	35–50	60–90
Fine	75–95	Semicountercurrent	6–10	30–50	60–90

*Courtesy of Sala International Inc.

netic field. Filamentary-type matrices, which occupy less than 10 percent of the magnetized volume yet still provide very high field gradients, can be used with these types of magnets.

The most familiar of the C-frame, matrix-type industrial magnetic separators are the Carpco, Eriez, Readings, and Jones devices. The Carpco separator employs steel balls as a matrix, Eriez uses a combination of expanded metal matrices, and the Readings and Jones separators have grooved-plate matrices. Capacities for this type of unit are reported to up to 180 t/h (in the case of Brazilian-hematite processing).

Solenoid magnetic separators are designed for batch-type, cyclic, and continuous operation. Devices which can use matrices of expanded metal, grooved plates, steel balls, or filamentary metals have been designed. Continuous separators with capacities to 600 t/h for iron ores (similar to the Brazilian hematite) are commercially available (Sala International Inc.). Selection of the method of operation is application-dependent, being based on variables such as temperature, pressure, volume of magnetics in the feed, etc.

A familiar type of cyclically operated solenoid electromagnet is the Franz separator, a well-known continuous type of solenoid separator manufactured by Krupp-Sol. An enclosed flux return-frame solenoid design for cyclic and continuous use is built by Sala International Inc.

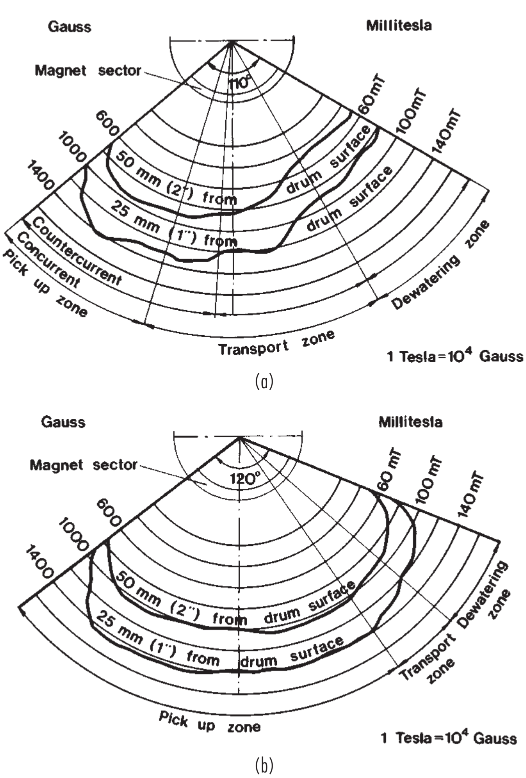


FIG. 19-46 Magnetic-field distribution charts. (*a*) Concurrent and countercurrent wet drum magnetic separator, 1.2-m diameter. (*b*) Counterrotation wet drum magnetic separator, 1.2-m diameter. (*Courtesy of Sala International, Inc.*)

A schematic of a continuous Sala high-gradient magnetic separator is shown in Fig. 19-52.

Depending on the type of matrix used, induced-pole magnetic separators can be classified as either high-intensity magnetic separators, which utilize grooved plates or steel balls as the matrix material, or as high-gradient magnetic separators, which use filamentary matrices such as steel wool or expanded metal. Filamentary matrices have proved to be more advantageous.

The maximum magnetic field produced by a C-type device is 2 T. For solenoids, conventional designs produce magnetic-field intensi-

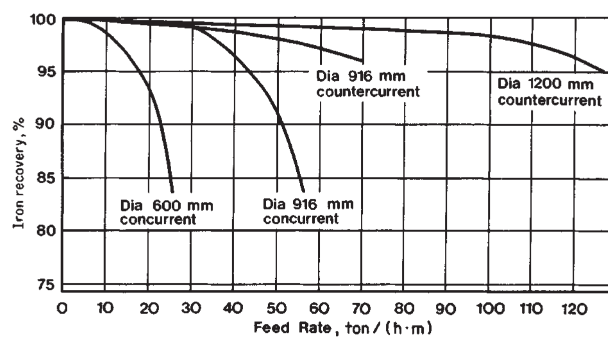


FIG. 19-47 Influence of drum diameter on separation. (*Courtesy of Sala International, Inc.*)

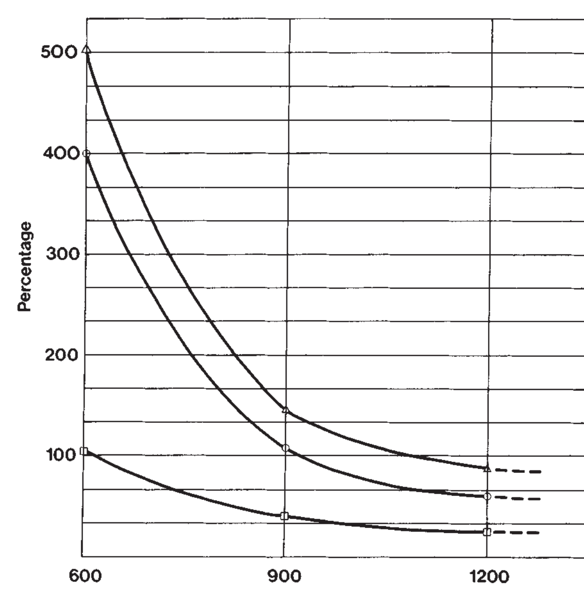


FIG. 19-48 Influence of drum diameter on separation cost. △, Two stages coarse separation; ○, three stages fine separation; □, sum of coarse and fine separations. (*Courtesy of Sala International, Inc.*)

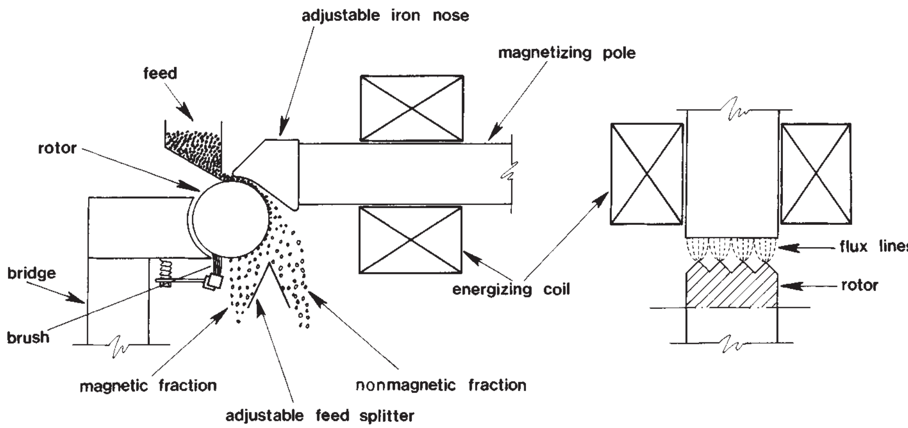


FIG. 19-49 Schematic diagram of an induced-roll separator.

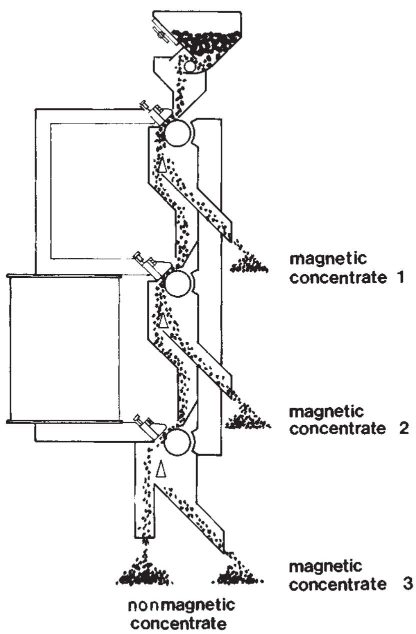


FIG. 19-50 Multiple induced-roll magnetic separator.

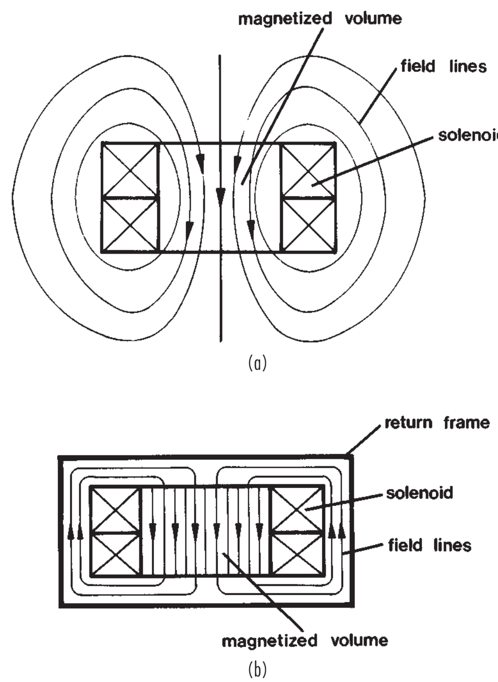


FIG. 19-51 Solenoid-type magnets. (*a*) Nonuniform field. (*b*) Uniform field by use of return frame.

ties up to 2 T, while superconducting units can be constructed with ratings up to 8 T.

In all induced-pole devices, the more magnetic particles are retained on the matrix while the less magnetic fraction is carried away in the slurry.

Dynamic (or Deflecting) Devices The oldest type of dynamic separation device is the Franz Isodynamic separator. This laboratory device has a dipole configuration with the poles shaped so that the value of $H\,dB/dz$ is constant throughout the working separation volume. Material to be separated can be fed through a vibratory chute or dropped between the poles, producing a separation based only on relative magnetic susceptibility.

There are a variety of new developments in magnetic separation which are of possible interest, but their commercial applicability is still not yet assured. These include quadripole separators and a spiral-flow device.

Table 19-23 lists potential applications for all types of magnetic separators.

SUPERCONDUCTING MAGNETIC SEPARATION

Superconducting Magnets In a superconducting magnet the magnetic field is generated in exactly the same way as on a normal electrical solenoid, coil, or winding. The only real difference is that the conductor is made from superconducting alloy which has to be maintained at a suitably low temperature to maintain the superconducting state. There are a number of possible material compositions which can be used for the superconducting winding but for industrial applications where economics and reliability play a key role an alloy of niobium and titanium is presently the preferred choice.

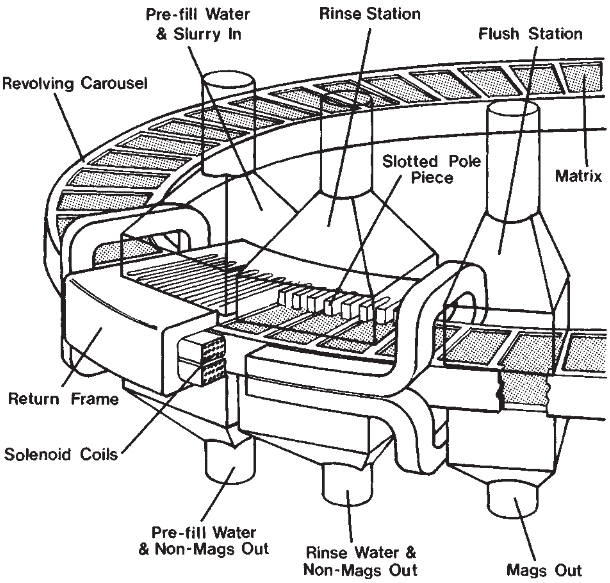


FIG. 19-52 Schematic of continuous high-gradient magnetic separator. (*Courtesy of Sala International, Inc.*)

Superconducting windings of N6Ti are now so commonplace that they can be considered as conventional superconductors. What is more relevant to the construction of a superconducting magnetic separator is the choice of refrigeration or cryogenic system used to cool the magnet winding, and it is the cryogenic system which to a large extent dictates the economics and practicality of these machines. There are presently three cryogenic routes which have been successfully applied.

1. *Closed-cycle liquefier systems.* In this design the superconductor resides in a bath of liquid helium and boil-off gas is recirculated through a helium liquefier. The installation of such a system is quite complex but these installations have proved good reliability providing there are no long-term interruptions to the supply of electrical power and cooling water.

2. *Low-loss system.* In a low-loss system the winding also resides in a reservoir of liquid helium but a very efficient insulation system enables the magnet to operate for long periods, typically 1 year or more, between liquid helium refills. An important feature of these systems is that they are relatively immune to short-term electrical power failures, which has enabled complete reliability even in extremely difficult environments.

3. *Indirect cooling.* The advent of heat engines based on the Gifford McMahon cycle to generate temperatures of 4 kelvin or less has made it possible to cool superconducting windings without the need for liquid helium. This technique offers great potential for small-scale systems where the economics of helium supply or the cost of a liquefier cannot be justified. The only drawback is that a constant supply of electrical power is essential for reliable operation.

The key benefits offered by superconducting magnets are (1) very low power consumption resulting from zero resistance of the magnet winding and (2) much higher magnetic fields which can be generated.

Superconducting magnets are presently being used in two distinct types of devices: *high-gradient magnetic separators* (HGMS) and *open-gradient magnetic separators* (OGMS). We shall consider these in turn.

Superconducting HGMS The HGMS principle relies on the capture of magnetic particles on a magnetized ferromagnetic matrix, as described in previous sections. In this type of device where an increase in magnetic-field induction enables capture of weaker magnetics and power consumption of large-scale systems is an important economic factor it is not surprising that superconducting magnets have made a significant impact. Indeed, for large-scale HGMS, superconducting magnets are by far the preferred choice.

A key feature of the HGMS process is that periodically the matrix must be demagnetized to flush out the captured magnetics. For superconducting magnets this demagnetization can be achieved by either de-energizing the magnet (switched-mode HGMS) or by moving the matrix canister (referred to as reciprocating canister HGMS) out of the magnetic field. The reciprocating matrix canister method is unique to superconducting HGMS and is shown schematically in Fig. 19-53. The differences resulting from the superconducting and resistive electromagnet HGMS systems are quite evident from the engineering data shown in Table 19-24.

TABLE 19-23 Potential Applications of Magnetic Separators

Device type	Type of construction	Maximum background magnetic field, Oe	Type of matrix which can be used	Maximum field gradient obtainable, G/cm	Required magnetic susceptibility for particulates	Particle size to be treated, mm	Materials which can be treated; fields of use
Grate	Permanent magnet	500	Rods	500	Ferro	<12	Tramp and fine iron
Pulley	Permanent magnet and electromagnet	100–200	—	100–1000	Ferro, strongly	<50	Ferro and strongly magnetic
Belt	Electromagnet	100–1000	—	100–1000	Strongly	0.15–30	Strongly magnetic
Drum	Permanent magnet and electromagnet	500–1000	—	500–1000	Strongly	0.02–20	Magnetite processing
Franz Isodynamic	Electromagnet	10,000	—	2000	Strongly, weakly	>0.01	Only for laboratory
Solenoid; Franz ferrofilter	Electromagnet	20,000	Steel ribbons, balls	200,000	Strongly, weakly	>0.01	Tramp and fine iron, ceramic slurries, industrial minerals, chemical industry
Induced rolls	Electromagnet	20,000	—	200,000	Strongly	0.03–3	Dry, dedusted, weakly magnetic particles
C-frame type; Jones	Electromagnet	20,000	Grooved plates	200,000	Strongly, weakly	0.01–2	Iron ores, industrial minerals
Carpco	Electromagnet	20,000	Steel balls	45,000	Weakly	0.01–1	Iron ores, industrial minerals
Marston Sala high-gradient magnetic separator	Electromagnet, superconducting	20,000 50,000	Steel wool, expanded metal, steel balls	25×10^6	Strongly to very weakly	0.0001–2	Iron ores, industrial minerals, coal, liquefied coal, wastewaters, purifiers, catalyst recovery, chemical industry

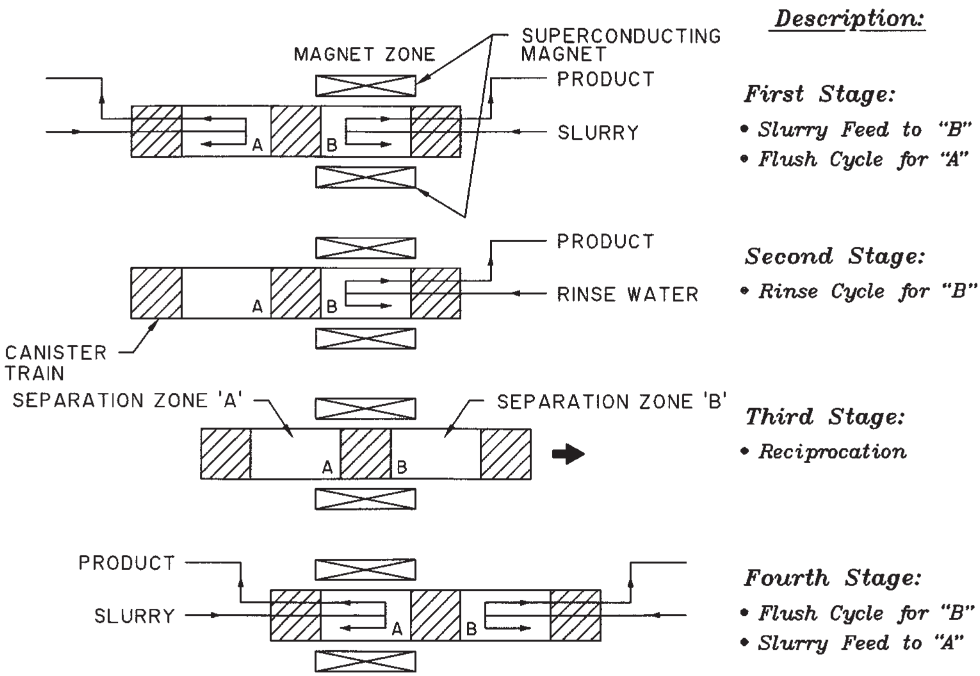


FIG. 19-53 Basic process cycle for a reciprocating canister superconducting magnetic separator. (*Courtesy Carpco, Inc.*)

One might expect that the perceived complexities of superconducting magnets would restrict their use to highly developed and industrialized areas. It is therefore noteworthy that the simplicity and reliability of the combination of low-loss cryogen technology coupled with the reciprocating canister principle has enabled a number of these HGMS units to operate with total reliability in areas as remote as the Amazon rain forest areas of Munguba and Rio Capim. Figure 19-54 shows the installation of a typical large-scale reciprocating canister HGMS.

A new development which shows promise of imminent industrial application is superconducting HGMS designed for treating dry feeds. One such unit employing a vibrating matrix in a reciprocating canister design has been evaluated with promising results. Once again the advantages of low power consumption and significantly improved levels of beneficiation are the key factors in driving this technology into industry.

Superconducting OGMS In open-gradient magnetic separators (OGMS) the magnet structure is arranged to provide a region in open space with a highly divergent field. Thus, the magnet geometry provides both the magnetic field and field gradient. Any paramagnetic material passing through this region will experience a force directly proportional to the field intensity and magnitude of field gradient. There are many conventional devices ranging from lift magnets to rare earth-drum and roll-type separators that operate on this principle. Superconducting OGMS has the benefit of offering not just higher magnetic force profiles, but a significantly greater depth of reach (i.e., larger separation volume) than permanent magnet and electromagnet devices. All superconducting OGMS units at the present operate with dry feeds usually with particle size >75 μm.

The earliest industrial application of superconducting OGMS was based on a drum separator design referred to as the "desces" separator. The drum was 1 meter in diameter and generated a peak field of 3 tesla and field gradient of 40 tesla/meter. A helium re-liquefier was required to provide adequate cryogenic capacity which significantly affected the capital cost, nevertheless, the unit has been operated successfully for many years in the beneficiation of magnesite processing normally 150 mm material at feed rates of up to 100 TPH.

A somewhat simpler and more compact OGMS device referred to as the "Cryostream" has recently been introduced. This unit operates on the inclined-fall process in which the magnet is held at an incline and material is simply allowed to fall through the magnetic region. The treatment size for the Cryofilter is 25 × 0.5 mm. The winding structure and overall geometry of this system make it an ideal candidate for indirect cooling by providing both overall simplicity and economic benefits.

TABLE 19-24 Comparative Data for 50 TPH HGMS Plant

	Superconducting Cryofilter 5T/460	Switched-mode superconducting HGMS	Conventional electromagnet
Magnetic induction	5 tesla	2 tesla	2 tesla
Separator weight	45 tons	250 tons	300–400 tons
Operator power consumption	10 kW	40–80 kW	280–400 kW
Overhead service crane for matrix canister exchange	2 tonne rail mounted	15 tonne gantry	15 tonne gantry
External cryogenic requirements	None required	1. Helium liquefier 2. Helium compressor 3. Liquid helium storage tank 4. Liquid nitrogen storage tank 5. Helium gas ballast tank	Not applicable

FIG. 19-54 Superconducting magnetic separator operating in a kaolin treatment plant. (*Courtesy Carpco, Inc.*)

The Cryostream operates with a peak field of 4 tesla and a magnetic force of 250 T^2/m allows separation of minerals with magnetic susceptibilities in the order of 10^6 emu/g. In essence the Cryostream is an industrial-scale version of the well known laboratory Frantz Isodynamic Separator.

ELECTROSTATIC SEPARATION

GENERAL REFERENCES: *SME Mineral Processing Handbook,* Society of Mining Engineers of the AIMMPE, NY, 1985 Edition. Moore, *Electrostatics and Its Applications,* Wiley-Interscience, New York, 1973. Knoll, Taylor, *Advances in Electrostatic Separation,* Minerals & Metallurgical Processing, 1985. E. Tondu et al., *Commercial Separation of Unburned Carbon from Fly Ash,* Mining Engineering, June 1996, p. 47–50.

General Principles Electrostatic separation (of particles), also commonly known as high-tension separation, is a method of separation based on the differential attraction or repulsion of charged particles under the influence of an electrical field. Applying an electrostatic charge to the particles is a necessary step before particle separation can be accomplished. Various techniques can be used for charging. These include contact electrification, conductive induction, and ion bombardment.

Regardless of the method of charging, the amount of charge that can be accumulated on a particle is limited by the maximum achievable charge density and the surface area of the particle. Electrostatic separation of mixed particles is possible when the electrostatic force acting on some particles is great enough to overcome gravity or inertial forces. Because the surface area of a solid varies as the square of a linear dimension whereas the mass varies as the cube of that dimension, gravity and inertial forces acting on solid particles increase faster with particle size than do electrostatic forces for charged particles in electric fields. Thus, there are upper size limits beyond which electrostatic separation of particles of a given shape is not feasible. For granular materials, this upper size limit is about 4 mm; for thin pieces of large cross-sectional area and for long pieces of small cross-sectional area, the limit can be greater than 25 mm.

The motion of fine particles immersed in a moving fluid is more greatly affected by fluid drag forces than that for similar large particles. For very small particles in a fluid, particle motion approximates the motion of the enveloping fluid. Industrial electrostatic separation of solid particles, which is universally conducted in air (or other easily ionizable gas), is difficult at particle sizes less than about 0.074 mm (200 mesh).

Charging Mechanisms

***Contact Electrification* (Fig. 19-55*a*)** When dissimilar materials touch each other, there is an opportunity for the transfer of electric charges. The extent of charge transfer can be such that a significant surface charge of opposite sign is developed when the materials are

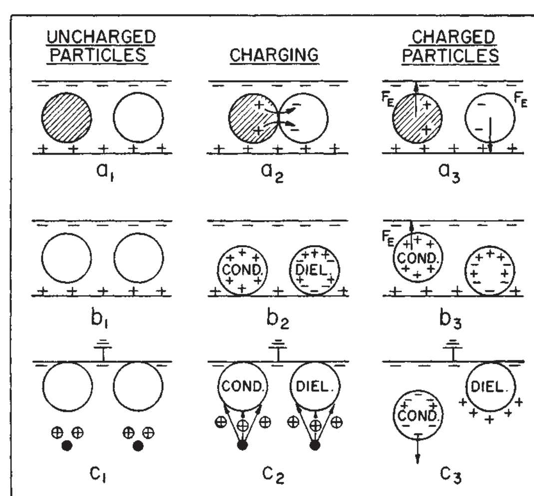


FIG. 19-55 Schematic representation of charging mechanisms. (*A*) Contact electrification. (*B*) Conductive induction. (*C*) Ion bombardment. Cond. = conductor particle; diel. = dielectric particle; ● = high-voltage dc electrode; ⊕ = ions from corona discharge at high-voltage electrode.

later separated. High temperatures and low humidity favor the development of high surface charges through the mechanism of contact electrification. Rubbing the materials together to increase the area of effective contact can also lead to high surface charges.

Particles carrying charges of opposite polarity due to contact electrification will be attracted to opposite electrodes when passing through an electric field and thus can be separated from each other.

***Conductive Induction* (Fig. 19-55*b*)** The term *conductive induction* describes the process by which an initially uncharged particle that comes into contact with a charged surface assumes the polarity and, eventually, the potential of the surface. A particle that is an electrical conductor will assume the polarity and potential of the charged surface very rapidly. However, a nonconducting particle will become polarized so that the side of the particle away from the charged surface develops the same polarity as the surface. Particles of intermediate conductivity may be initially polarized but approach the potential of the charged surface at a rate depending on their conductivity.

If a conductor particle and a nonconductor particle are just separated from contact with a charged plate, the conductor particle will be repelled by the charged plate and the nonconducting particle will be neither repelled nor attracted by it.

The charged plate must be balanced by other oppositely charged (or earthed) bodies to maintain overall neutrality. In electrostatic separation, this is usually accomplished by means of a single electrode of charge opposite in sign to that of the charged plate. The conductor particle is then in the electrical field between the two electrodes and experiences a net electrostatic force in the direction of the second electrode. The nonconducting particle, having no net charge, experiences no electrostatic force in a uniform electric field. Electrostatic separation of the conductor and nonconductor particles can be accomplished by movement of the conductors in the electric field.

***Ion Bombardment* (Fig. 19-55*c*)** The most positive and strongest method of charging particles for electrostatic separation is ion bombardment. Use of ion bombardment in charging materials of dissimilar properties may be visualized by considering conductor and nonconductor particles touching the grounded conducting surface of Fig. 19-55*c*. Both particles are bombarded by ions of atmospheric gases generated by an electrical corona discharge from a high-voltage electrode (usually a fine tungsten-alloy wire at 20 to 30 kV with respect to ground and several centimeters away from the particles). When ion bombardment ceases, the conductor particle loses its acquired charge to ground very rapidly and experiences an opposite electrostatic force tending to repel it from the conducting surface. The nonconducting particle, however, being coated on its side away from the conducting surface with ions of charge opposite in electrical polarity to that of the surface, experiences an electrostatic force tending to hold it to the surface. If the electrostatic force is larger than the force of gravity or other forces tending to separate the nonconducting particle from the conducting surface, the particle is held in contact with the surface and is said to be "pinned."

Electrostatic-Separation Machines The first electrostatic machines to be used commercially employed the principle of contact electrification. These were free-fall devices incorporating large vertical plates between which an electrostatic field was maintained. Triboelectric separation (contact charging) has experienced an increase in applications due to advances in mechanical self-cleaning and electrical design as well as the development of efficient precharging techniques.

Triboelectric Separators There are currently two industrial forms of triboelectric separators installed commercially to treat minerals and recycled plastics:

Tube-type. These separators are typically divided into two sections: (1) precharging and (2) separation. The precharging section is designed to create or enhance the charge difference between particles to be separated (typically by some form of contact mechanism or external pretreatment to render one constituent positive or negative in comparison to the other materials present. The separation section consists of two vertical walls of tubes opposing each other. Each tube

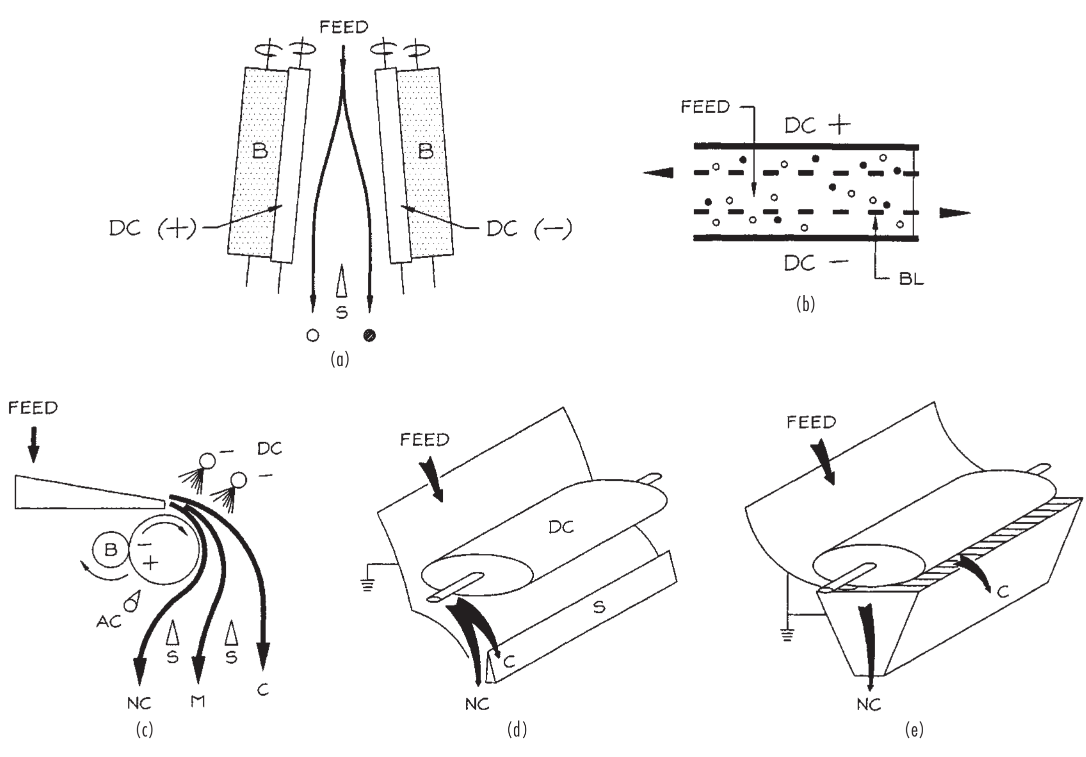


FIG. 19-56 Operating principles of electrostatic separators. *C* = conductors; *NC* = nonconductors; *M* = middling; *DC* = high-voltage dc electrodes; *AC* = high-voltage ac wiper (electrodes); *B* = brush; *S* = splitter; ○ = negatively charged particles; ● = positively charged particles; *BL* = belt.

"wall" is electrified with the opposite potential, and product splitters running parallel to the electrode walls at the base of unit separate materials attracted to the oppositely charged tube electrodes. This arrangement is illustrated in Fig. 19-57.

Belt-type. Figure 19-58 illustrates a horizontal belt-type separator equipped with fast-moving belts that travel in opposite directions adjacent to suitably placed plate electrodes of the opposite polarity. Material is fed into a thin gap between two parallel electrodes. The particles are swept upward by a moving open-mesh belt and conveyed in opposite directions, thus facilitating particle charging by contact with other particles. The electric field attracts particles up or down depending on their charge. The moving belts transport the particles adjacent to each electrode toward opposite ends of the separator.

The common types of other industrial electrostatic separators employ charging by conductive induction and/or ion bombardment. Figure 19-56 illustrates the principles of application.

Conductive-Induction Machines Electrostatic separators exploiting the principle of conductive induction will generally use the following electrode designs:

Plate separators. These separators introduce material typically by gravity onto a grounded-metal slide in front of which is placed a static electrode of large surface area. The elaborate contact electrification used in earlier free-fall electrostatic separators was avoided in later devices by incorporating the slide principle. Separation occurs by particles selectively acquiring an induced charge from the grounded plate and then being attracted in the direction of the charged electrode. Refer to Fig. 19-59.

Screen-plate separators. These include a metal slide at ground potential that is extended with a conducting screen of suitable screen-opening size to allow easy passage of the largest grains being treated. A stationary electrode is placed above the slide and screen sections as

FIG. 19-57 Triboelectric separators. V-Stat electrostatic separator for silica removal from industrial minerals. (*Courtesy of Carpco, Inc.*)

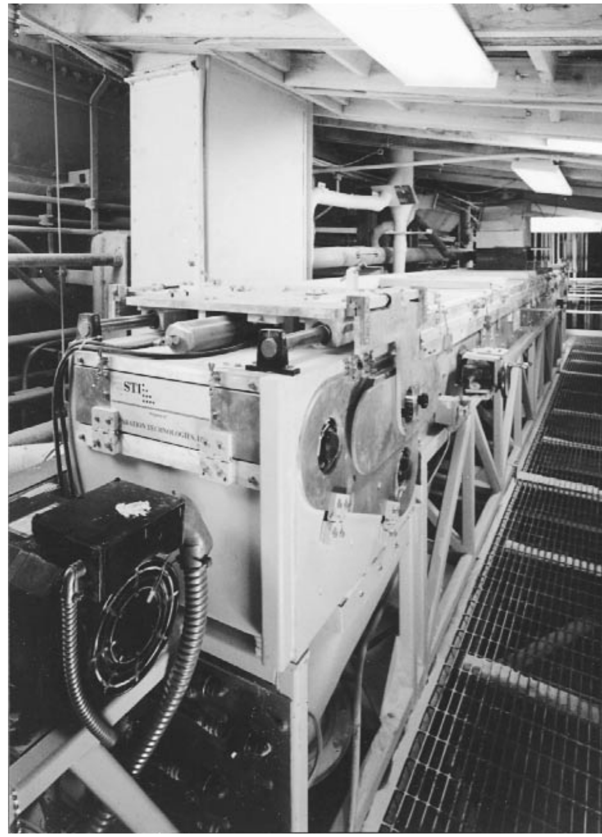


FIG. 19-58 Triboelectric separators. Belt-type electrostatic separator for separation of carbon from fly ash. (*Courtesy of Separation Technologies, Inc.*)

shown in Fig. 19-56*e.* Particles capable of assuming an induced charge from the slide are attracted by the electrode and prevented from passing through the screen grid; the other particles pass through the screen unaffected.

Ion-Bombardment Machines *Conductive roll (drum) separators.* These separators are, by far, the most widely employed industrial-machine type. The electrostatic elements of these machines consist of a conductive rotating drum at ground potential coupled with one or more high-voltage ionizing electrodes. Suitably placed nondischarging (static) electrodes are often used in conjunction with an ionizing electrode to create a static field which aids centrifugal force in removing conductive particles from the drum surface.

Drum construction is typically of carbon or stainless steel when treating granular materials minus 1 mm in size. Drum diameter has ranged from 0.150 to 0.360 m, while drum length varies from 0.460 to 3.050 m in industrial ion-bombardment (high-tension) machines.

Feeding these separators is accomplished by vibratory, belt, rotary spline, or gravity methods, depending on the particle size being treated. Vibratory and belt feeding techniques are preferred for coarser sizes, and rotary spline and gravity methods are normally used for finer materials. Exceptions to this generalization can be observed in plant practice. The ionizing electrodes employed in these machines vary considerably in appearance, but all produce a corona.

An alternating-current electrode system referred to in the industry as a "wiper" is often installed in the nonconductor product-collection section behind each drum. The function of the wiper is to use an ac corona to neutralize the charge on the nonconductor particles pinned

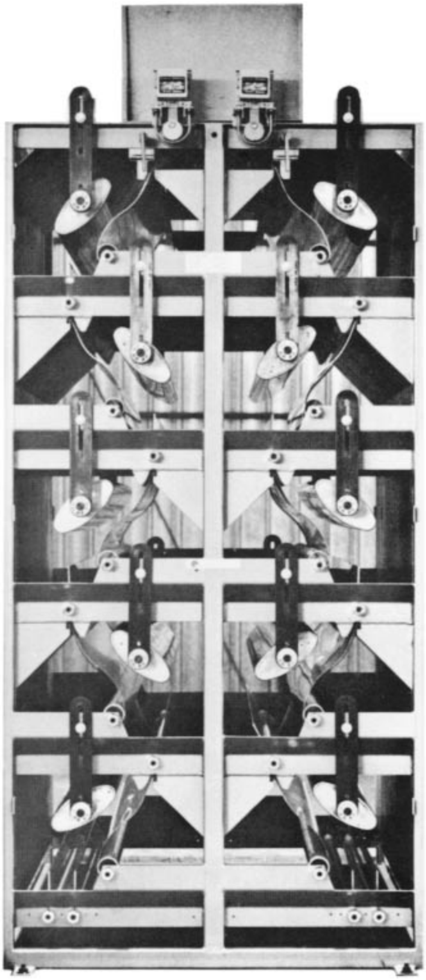


FIG. 19-59 Conductive-induction plate-type electrostatic separator. (*Courtesy of Mineral Technology, Ltd.*)

to the surface of the drum and thereby reduce the workload for mechanically operated brushing systems.

Ion-bombardment machines are available in horizontal and vertical (stacked) configurations. Horizontal units are preferred for large-tonnage applications in which machines are arranged in rows for ease of maintenance and operation. Stacked units up to four rolls high have been used to reduce material-handling costs in multipass treatment schemes when the material is capable of being passed vertically from one roll to another by gravity. Internal details of industrial roll-type separators are shown in Fig. 19-60.

Power Supplies High-voltage ac and dc power supplies for electrostatic separators are usually of solid-state construction and feature variable outputs ranging from 0 to 30,000 V for ac wiper transformers to 0 to 60,000 for the dc supply. The maximum current requirement is approximately 1.0 to 1.5 mA/m of electrode length. Power supplies for industrial separators are typically oil-insulated, but smaller dry-epoxy-insulated supplies are also available.

Features common to most high-voltage dc power supplies include reversible polarity, short-circuit and current-limiting protection, and automatic residual-charge dissipation to ground.

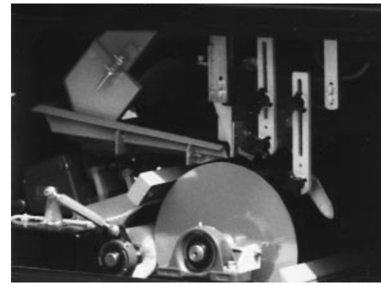


FIG. 19-60 Internal configuration of roll-type electrostatic separator, ionizing mode. (*Courtesy Carpco, Inc.*)

High-voltage controllers which regulate primary input voltage to the rectifier and wiper transformer and house primary current-limiting protection, meters, and instrumentation are designed for local or remote operation.

Machine Capacities Table 19-25 presents machine-capacity information for electrostatic separators.

Applications of Electrostatic Separation

Mineral Beneficiation Electrostatic methods are widely used in the processing of ores with mineral concentrates. Generally, electrostatic separation is used as a part of an overall flow sheet comprising various combinations of physical separation procedures. It is particularly well established in the processing of heavy-mineral beach sands from which are recovered ilmenite, rutile, zircon, monazite, silicates, and quartz. High-grade specular hematite concentrates have been recovered at rates of 1000 tons/h in Labrador. Applications also include processing tin ores to separate cassiterite from columbite and ilmenite. Refer to Fig. 19-61*a*.

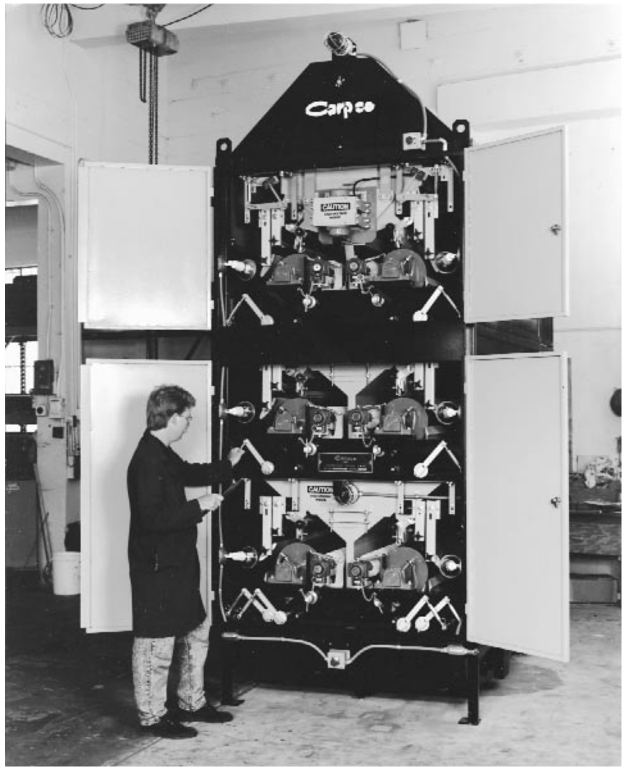
Charging by ion bombardment is the technique used in most mineral separations. The conductive-induction (nonionizing) plate types of separators have also been used. Applications of this device in the minerals industry include its use as a final cleaning step when concentrating rutile and zircon.

Generally, separators of the conductive-induction type have a lower capacity per unit length of electrode than the ion-bombardment (ionizing) type of apparatus, and multipass operation is typically required. This disadvantage is offset by the ability of these separators to (1) produce high-grade concentrates from ore materials that are otherwise difficult to process and (2) process a coarser material than competitive

TABLE 19-25 Machine Capacities of Electrostatic Separators for Mineral Applications

Type	Capacity, t/h
A) Triboelectric separators	
• Belt type (1 pass)	10
• Tube type (1 pass)	20
B) Conductive-induction electrostatic separators	
Plate separator: basis, 5 pass × 2 start; 1.42 m in length per start; capacity, 900 kg/(h·m)	2.0
C) Ion-bombardment electrostatic separators	
2-m units-basis:	
• 1 pass × 6 starts × 2 m × 2000 kg/(h·m)	24
• 3 pass × 2 start × 2 m	8

NOTE: Capacity information is based on the treatment of industrial minerals having a specific gravity of 2.6 to 4.0.



(a)

(b)

FIG. 19-61 (*a*) Mineral separation: high-capacity 4-roll (250 mm dia × 2000 mm long) separator featuring all-start × 2-pass separation. Siz-roll units also available. (*b*) Recycling separator for nonferrous metals and plastics: 4-roll (350 mm dia × 1500 mm long) separator featuring 2-start × 2-pass or 1-start × 4-pass separation. (*Courtesy Carpco, Inc.*)

processes such as froth flotation. These units are used as a final cleaning stage for rutile and zircon.

Electrostatic-type separation is being tested as an alternative to the presently used process of flotation of pebble phosphates for coarser-size fractions. Advantages sought include reduced reagent costs, a lower water requirement, and fewer tailings-disposal problems when a part of the flotation circuit is eliminated. The largest application of triboelectric separation is in the salt industry where sodium and potassium salts are separated after preconditioning.

Plastic and Metals Recycling Electrostatic separation has been increasingly applied to recover nonferrous metals from industrial plastics (telephone and communication scrap). It also is an important step in the recycling of beverage bottles to reject any remaining nonferrous metals. Both of these recycling applications make use of roll-type ion-bombardment separators (Fig. 19-61*b*).

A new application of triboelectric separation involves the separation of PVC from PET and other plastics. Recent developments in precharging technology permit PVC to assume a strong negative charge and be removed efficiently from properly protected mixed plastic feedstocks (Fig. 19-62).

Other Applications Electrostatic separators have been used to separate a number of different types of materials not only on the basis of differences in dielectric properties but also in combination with differences in surface conductivity and shape factors. Among these operations are seed sorting, cleaning of spices, separating of pill coatings from base materials, removal of textile from reclaimed plastics, and separation of paper and plastic. Electrostatic separation has also been adapted for use in classification and sizing when elongated particles or extremely fine sizes cause difficulty in conventional dry-screening applications.

Typical Operating Conditions Table 19-26 presents some typical values of important operating conditions for the separation of several different types of feed materials. In considering candidate processes for a given separation job, the table can sometimes be helpful in showing that materials of similar properties and/or economic value can be treated by electrostatic separation.

FIG. 19-62 New Triboelectric separator for separation of PVC from other plastics. (*Courtesy Carpco, Inc.*)

TABLE 19-26 Typical Operating Conditions for Electrostatic Separations

Type of particle charging	Feed	Separation	Type of separator	Feed temperature, °C	Feed size, mm	Feed rate, metric tons per hour per start*	No. of stages of separation
Triboelectric	Silica from limestone	Reduction of quartz by 80–90%	Tube type	80–100	−1.0 + 0.015	20	1
	Florida pebble-phosphate flotation conc.	Residual silica from pebble phosphate	Tube type	70–90	−1.0 + 0.10	10–15	1
Conductive induction	Zircon or rutile concentrate (eastern Australia)	Residual conductor minerals from rutile and zircon; upgrading of concentrate from 98.95 to 99.35% zircon at 92% recovery	Plate	50–80	−0.21 + 10.074	0.6–0.7	5–10
Ion bombardment	Heavy-mineral concentrate	Conductor minerals (ilmenite, rutile) from non-conductor minerals (zircon, monazite, aluminum silicates, quartz and others)	Roll	120	−1.0 + 0.04	2.5	3–6
	Iron ore	Iron oxides from quartz and silicates	Roll	120	−1.0	6–7	2–4
	Tungsten concentrate	Scheelite from iron oxides and other conductor minerals	Roll	150	−0.6	1.0–1.5	3
	Chrome ore	Chromite from silica and silicates	Roll	120	−0.85		
	Chopped wire	Metal from plastic insulation	Roll	Ambient	−12.5	1.5–2.0	2–4
	Metal powder	Removal of nonmetallic impurities	Roll	Ambient to 120	−0.20	1–2	2–4

*To convert metric tons per hour per start to kilograms per second per start, multiply by 0.2778.

FLOTATION

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INTRODUCTION

Mixed liberated particles can be separated from each other by flotation if there are sufficient differences in their wettability. The flotation process operates by preparing a water suspension of a mixture of relatively fine-sized particles (smaller than 150 micrometers) and by contacting the suspension with a swarm of air bubbles of air in a suitably designed process vessel. Particles that are readily wetted by water (hydrophilic) tend to remain in suspension, and those particles not wetted by water (hydrophobic) tend to be attached to air bubbles, levitate (float) to the top of the process vessel, and collect in a froth layer. Thus, differences in the surface chemical properties of the solids are the basis for separation by flotation.

Surfaces that do not have strong surface chemical bonds that were broken tend to be nonpolar and are not readily wetted. Substances such as graphite and talc are examples that can be broken along weakly bonded layer planes without rupturing strong chemical bonds. These solids are naturally floatable. Also, polymeric particles possess nonpolar surfaces and are naturally hydrophobic. By contrast, most naturally occurring materials are polar and exhibit high free energy at the polar surface. The polar surfaces react strongly with water and render those particles naturally hydrophilic. The relative wettability of the solids in a mixture can be enhanced by the addition of various surface chemical agents that are adsorbed selectively on the particle surface.

Mineral Applications. The flotation process is most widely used in the mineral process industry to concentrate mineral values in the ores.

A U.S. Bureau of Mines survey covering 202 froth flotation plants in the United States showed that 198 million tons of material were treated by flotation in 1960 to recover 20 million tons of concentrates which contained approximately $1 billion in recoverable products. Most of the world's copper, lead, zinc, molybdenum, and nickel are produced from ores that are concentrated first by flotation. In addition, flotation is commonly used for the recovery of fine coal and for the concentration of a wide range of mineral commodities including fluorspar, barite, glass sand, iron oxide, pyrite, manganese ore, clay, feldspar, mica, sponumene, bastnaesite, calcite, garnet, kyanite, and talc.

Other Applications. In addition to the minerals industry, flotation is finding a variety of new applications in other fields. The next largest application is for wastewater treatment to remove particulate, organic, and biological contaminants. Other applications include extraction of metallic values or removal of heavy metal compounds from hydrometallurgical streams by precipitate flotation, recovery of bitumen from tar sands, deinking of waste paper, recovery of solids from white water in paper making, recovery of glass sands from industrial wastes, removal of impurities from peas, removal of ergot from rye, separation of proteins from milk, and clarification of fruit juices. Ion flotation and foam fractionation are the slight modifications in the basic flotation process and are sometimes referred to as "adsorptive bubble separation." These methods are used for the extraction of soluble species.

GENERAL ASPECTS

Unit operation of flotation is based on two major steps: (1) *conditioning* and (2) *separation,* as is schematically depicted in Fig. 19-63. During the first step, the *slurry* or the *pulp,* consisting of particles to be separated, the particle size of which is already properly adjusted, is fed to the conditioning unit, to which the necessary flotation reagents are added. The main purpose of the conditioning step is to create physical-chemical conditions for achieving appropriate selectivity between particle species that are to be separated. The second step is then intended to generate and introduce air bubbles into the process vessel for contacting them with particulate species so as to affect their separation by flotation. Particles attached to the air bubbles are in most applications removed from the process vessel as froth. Accordingly,

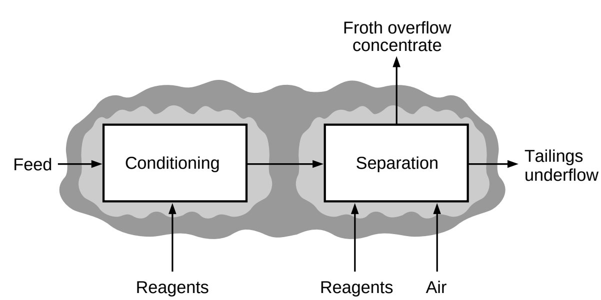

FIG. 19-63 Basic steps in a flotation system.

the unit operation of flotation is often referred to as *froth flotation.* The froth overflow stream is called a *concentrate* in the minerals industry, and the slurry underflow is termed *tailings.* Depending on the application, these two steps may be carried out in two distinctly different process units or in one combined unit.

Flotation Reagents. Three types of chemical reagents are used during the froth flotation process: collectors, frothers, and modifiers.

Collectors. These are surface-active agents that are added to the flotation pulp, where they adsorb selectively on the surface of the particles and render them hydrophobic. A convenient classification of the commonly used collectors is shown in Fig. 19-64. The nonionizing collectors (fuel oils and kerosene) are practically insoluble in water and cause the particles to become hydrophobic by covering them with a thin film. The ionizing collectors dissociate into ions in water and are made up of complex heteropolar molecules in that the molecule contains both a nonpolar hydrocarbon group with pronounced hydrophobic properties and a polar group with hydrophilic properties. The ionizing collectors adsorb either physically or chemically on the particle surface and can further be classified into anionic or cationic collectors depending on the nature of the nonpolar hydrocarbon group. Common examples of the ionizing collectors include fatty acids, long-chain sulfates, sulfonates and amines, xanthates, and dithiophosphates. Dosage requirements for collectors depend on the mechanisms by which they interact with the particle surface, but just enough is needed to form a monomolecular layer. As a rule, high dosages are required for nonionizing collectors and physisorbing ionizing collectors (in the order of 0.1 to 1 g of reagent per kg of solids) and low dosages for chemisorbing ionizing collectors (0.01 to 0.1 g of reagent per kg of solid). Addition of excess quantities of a collector is not desirable because it results in reducing the selectivity and increasing the cost.

Frothers. These are also surface-active agents added to the flotation pulp primarily to stabilize the air bubbles for effective particle-bubble attachment, carryover of particle-laden bubbles to the froth, and removal of the froth. The frother action is similar to the ionizing collectors except that they concentrate primarily at the air-liquid interface. Commonly used frothers are pine oil, cresylic acid, polypropylene glycol, short-chain alcohols, and 5- to 8-carbon aliphatic alcohols. Quantities of frothers required are usually 0.01 to 0.1 g per kg of solids.

Modifiers. Flotation modifiers include several classes of chemicals.

1. *Activators.* These are used to make a mineral surface amenable to collector coating. Copper ion is used, for example, to activate sphalerite (ZnS), rendering the sphalerite surface capable of absorbing a xanthate or dithiophosphate collector. Sodium sulfide is used to coat oxidized copper and lead minerals so that they can be floated by a sulfide mineral collector.
2. *pH regulators.* Regulators such as lime, caustic soda, soda ash, and sulfuric acid are used to control or adjust pH, a very critical factor in many flotation separations.
3. *Depressants.* Depressants assist in selectivity (sharpness of separation) or stop unwanted minerals from floating. Typical are sodium or calcium cyanide to depress pyrite (Fe_2S_2) while floating galena (PbS), sphalerite (ZnS), or copper sulfides; zinc sulfate to depress ZnS while floating PbS; sodium ferrocyanide to depress copper sulfides while floating molybdenite (MoS_2); lime to depress pyrite; sodium silicate to depress quartz; quebracho to depress calcite ($CaCO_3$) during fluorite (CaF_2) flotation; and lignin sulfonates and dextrins to depress graphite and talc during sulfide flotation.
4. *Dispersants and flocculants.* These are important for the control of slimes that sometimes interfere with the selectivity and increase reagent consumption. For example, soda ash, lime sodium silicate, and lignin sulfonates are used as dispersants, and starch and polyacrylamide are used as flocculants.

Quantities of modifying agents used vary widely, ranging from as little as 0.01 to 0.1 g/kg to as high as 1 to 2 g/kg of solids, depending upon the reagent and the metallurgical problem.

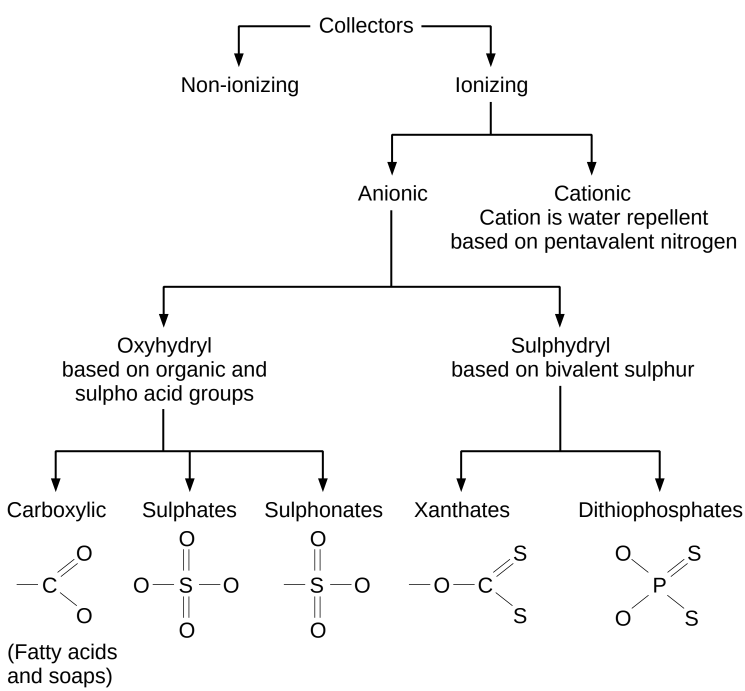

FIG. 19-64 Classification of collectors (after Glembotakii et al., 1972).

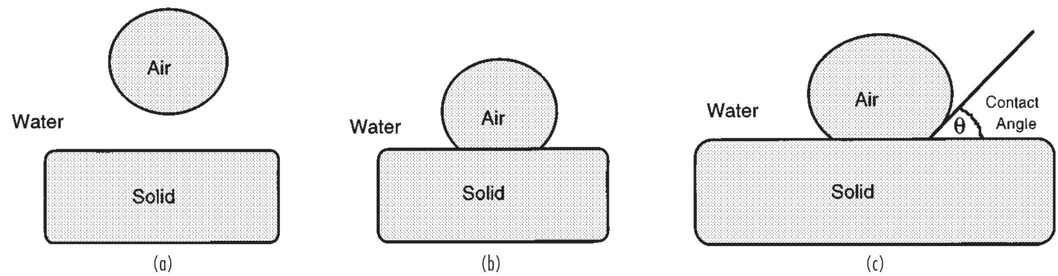

FIG. 19-65 Schematic representation of air bubble-water-solid particle system: (*a*) before, (*b*) after particle-bubble attachment, and (*c*) equilibrium force balance.

FUNDAMENTALS

Flotation is a physical process involving relative interaction of three phases: solid, water, and air. An understanding of the wettability of the solid surface, physical surface, and chemical phenomena by which the flotation reagents act and the mechanical factors that determine particle-bubble attachment and removal of particle-laden bubbles, is helpful in designing and operating flotation systems successfully.

Thermodynamics of Wetting. The fundamental objective of flotation is to contact solid particles suspended in water with air bubbles (Fig. 19-65*a*) and cause a stable bubble-particle attachment (Fig. 19-65*b*). It is seen that attachment of the particle to an air bubble destroys the solid-water and air-water interfaces and creates air-solid interface. The free energy change, on a unit area basis, is given by

$$\Delta G = \gamma_{AS} - (\gamma_{SW} + \gamma_{AW}) \tag{19-21}$$

where γ terms are the interfacial tensions of the air-solid (AS), solid-water (SW), and air-water (AW) interfaces, respectively. A force balance for the air-water-solid particle system (Fig. 19-65*c*) yields the familiar Young's Equation

$$\gamma_{AS} = \gamma_{SW} + \gamma_{AW} \cos \theta \tag{19-22}$$

where θ is the contact angle (measured through the water phase). It must be seen that the contact angle is an equilibrium measure of the interfacial energy of the air-water-solid system. Combining the above two equations, one obtains

$$\Delta G = \gamma_{AW}(\cos \theta - 1) \tag{19-23}$$

Thus, for any finite value of the contact angle, the free energy change becomes negative and particle-bubble attachment can take place. As mentioned above, polar solids have high surface energy and are wet by water. Therefore, the contact angle is zero. The wettability of solids can be controlled through adsorption of chemical reagents, which can change the interfacial tensions, so that the contact angle becomes finite and flotation can take place.

Physical-Chemical Phenomena. Several physical-chemical phenomena occur when chemical reagents are added to an air-water solid system due to the interaction of the reagents with the air-water, water-solid, and air-solid interfaces. This causes changes in the solution chemistry in which the particles are suspended. Some of the important phenomena that occur due to the addition of reagents include: solubility and dissociation of reagents in water, change of pH of the suspension, change of air-water surface tension, physical and chemical adsorption of the dissolved species on the solid surfaces due to hydrogen bond formation, electrostatic interactions, hydrophobic bonding, chemical bond formation, and fixation of reagent species in the solid lattice. All these phenomena in essence result in affecting the contact angle and flotation nature of solid particles and their attachment to air bubbles. A simplified pictorial representation of collector adsorption on particle surface, action of frother on the air bubble formation, and particle-bubble contact is shown in Fig. 19-66. An adequate understanding of the role played by the reagents and their proper choice to create the desired conditions is paramount to successful flotation.

Particle-Bubble Attachment. In the above, principles leading to creation of desired hydrophobicity/hydrophilicity of the particles has been discussed. The next step is to create conditions for particle-bubble contact, attachment, and their removal, which is simply described as a combination of three stochastic events with which are associated the probability of particle-bubble collision, probability of attachment, and probability of retention of attachment. The first term is controlled by the hydrodynamic conditions prevailing in the flotation unit. The second is determined by the surface forces. The third is dependent on the survival of the laden bubble by liquid turbulence and impacts by the other suspended particles. A detailed description of the hydrodynamic and other physical aspects of flotation is found in the monograph by Schulze (1984).

Process Variables. There are a number of variables that govern the flotation process. These include particle characteristics (size, shape, and chemical and mineralogical composition), chemical variables (type and amount of flotation reagents added), flotation machine variables (equipment size, internal geometry of the device, speed of operation, etc.) and operating variables (slurry feed rate and percent solids). A combined effect of all these variables can be represented by two independent variables—specific flotation rate (representing the rate of flotation of particles per unit time) and residence time of the pulp in the flotation device—and two dependent variables of grade (composition of the desired component) and recovery (ratio of the weight of the desired component in the froth product to that in the feed) of the froth product.

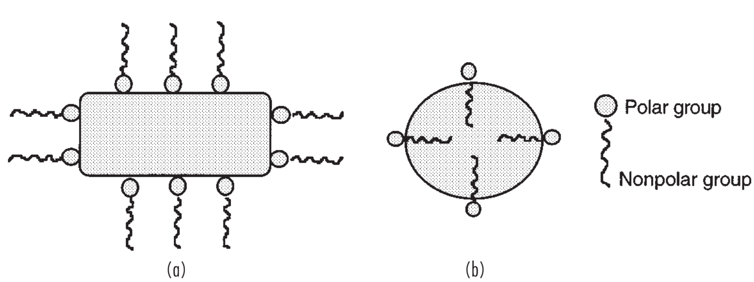

FIG. 19-66 Schematics of (*a*) collector adsorption at the particle-water interface and (*b*) action of the frother.

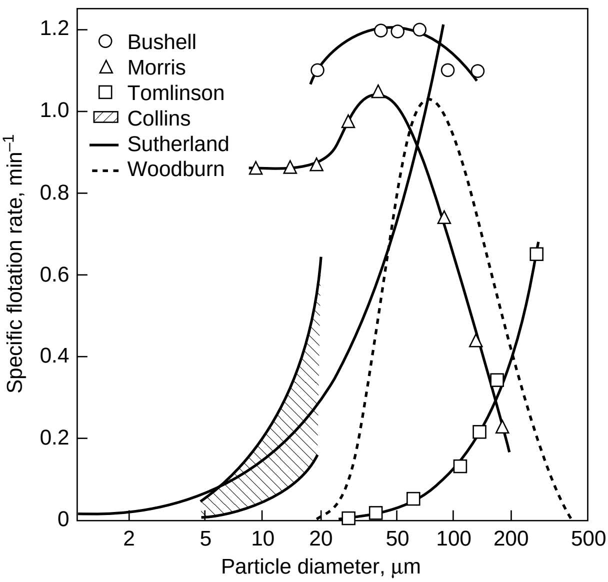

FIG. 19-67 Effect of particle diameter on specific flotation rate (Fuerstenau, 1980).

Several laboratory procedures are available to investigate the flotation response of any solid-solid system and in generating basic data for the selection and sizing of the flotation units and circuits. Also available are various process models for flotation with varying degrees of sophistication and representation. These process models can form a quantitative basis during all stages of engineering flotation systems.

Finally, numerous types of flotation reagents and flotation equipment with different design details are currently available, and their proper choice has to be made depending on the kind of separation task one has on hand. Monographs listed in the general references provide a good starting point towards an understanding of these aspects.

Effect of Particle Size. Particle size is the most significant variable of flotation separation. The effect of particle size on the flotation rate is shown in Fig. 19-67. Particles in the size range of 20 to 60 µm have the highest flotation rate. Larger particles, being heavy, cannot be easily levitated and recovered, even though proper thermodynamic conditions might exist. In contrast, as particles become small, they become lighter and their surface-to-volume ratio becomes large. There are several factors that enter into making the flotation of small particles quite inefficient (Fig. 19-68).

EQUIPMENT

Various types of flotation machine designs can be classified into different categories based on the methods used for the generation and introduction of air bubbles into the equipment (Fig. 19-69). Each of the techniques of air bubble generation and particle-bubble contact along with the special features associated with different kinds of equipment has its own advantages and limitations. These must be considered carefully in selecting the equipment for a specific application. Individual manufacturers can provide basic help in selecting the equipment.

Electrolytic Flotation Units. Electrolytic or electroflotation is based on the generation of hydrogen and oxygen bubbles in a dilute aqueous solution by passing direct current between two electrodes. Choice of electrode materials include aluminum, platinized titanium, titanium coated with lead dioxide, and stainless steel of varying grades. Figure 19-70 illustrates the basic arrangement of an electrolytic flotation unit.

Electrical power to the electrodes is supplied at a low voltage potential of 5 to 10 volts. The power consumption is in the range of 0.5 to 0.7 kW/m^2 of flotation tank surface area depending on the conductiv-

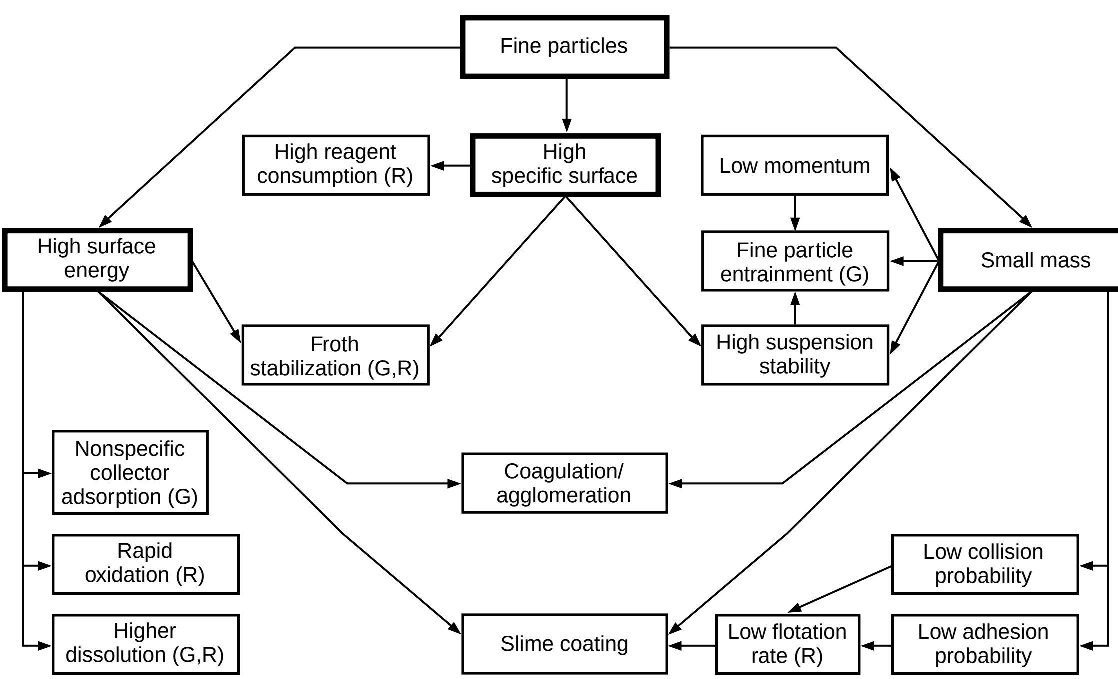

FIG. 19-68 The schematic diagram showing the relationship between the physical and chemical properties of fine particles and their behavior in flotation. (G) and (R) refer to whether the phenomena affects grade and/or recovery. The arrows indicate the various factors contributing to a particular phenomena observed in flotation of fine particles (Fuerstenau, 1980).

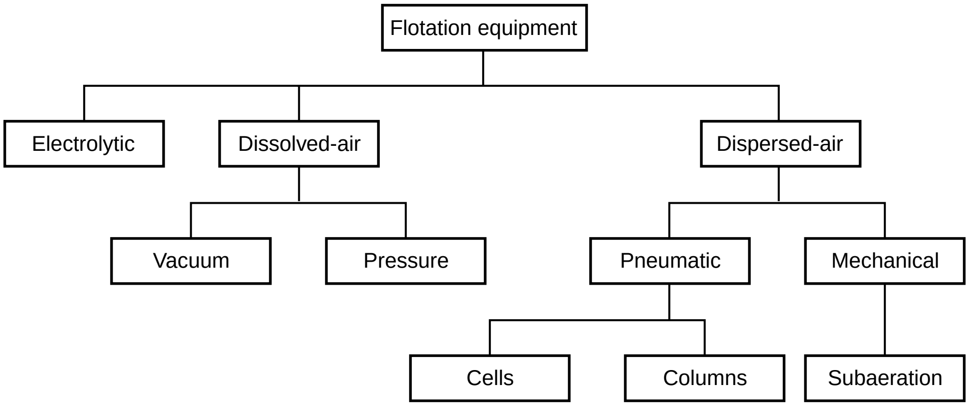

FIG. 19-69 Classification of flotation equipment based on the generation and introduction of air bubbles.

ity of the liquid and the distance between the electrodes. Such a unit produces approximately 50 to 1 of gas/h/m^2 of tank area. The main drawback of the electroflotation units is associated with the electrodes in terms of their fouling requiring mechanical cleaning devices and their consumption needing replacement at frequent intervals.

The bubble size in these cells tends to be the smallest (10 to 50 μm) as compared to the dissolved-air and dispersed-air flotation systems. Also, very little turbulence is created by the bubble formation. Accordingly, this method is attractive for the separation of small particles and fragile flocs. To date, electroflotation has been applied to effluent treatment and sludge thickening. However, because of their bubble generation capacity, these units are found to be economically attractive for small installations in the flow-rate range of 10 to 20 m^3/h. Electroflotation is not expected to be suitable for potable water treatment because of the possible heavy metal contamination that can arise due to the dissolution of the electrodes.

Dissolved-Air Flotation Units. Dissolved-air flotation entails saturating the process stream with air and generating air bubbles by releasing the pressure. Particle-bubble contact is achieved by the direct nucleation and growth of air bubbles on the particles, and very little mechanical agitation is employed. The dissolved-air precipitates in the form of fine bubbles in the size range of 20 to 100 μm. This method of air bubble generation does not require the addition of frother-type chemical reagents and often limits the total quantity of aeration possible. As such, dissolve-air flotation systems are used to treat process streams with low solids concentration (0.01 to 2 percent by volume). Vacuum flotation and pressure flotation are the two main types of dissolved-air flotation processes, with the latter being most widely used.

In vacuum flotation, the process stream is saturated with air at atmospheric pressure and introduced to the flotation tank on which a vacuum is applied, giving rise to the generation of the air bubbles. The process can be run only as a batch process and requires sophisticated equipment to produce and maintain the vacuum. By and large, the amount of air released during flotation is limited by the vacuum achievable.

In contrast to vacuum flotation, dissolved-air flotation units can be operated on a continuous basis by the application of pressure. This consists of pressurizing and aerating the process stream and introducing it into the flotation vessel that is maintained at the atmospheric pressure. The reduction of pressure results in the formation of fine air bubbles and the collection of fine particulates to be floated and removed as sludge.

Pressurization could be carried out on the entire feed stream (full-flow pressure flotation) or a fraction of the feed stream while the remainder is introduced directly without aeration into the flotation tank (split-flow pressure flotation). The split-flow system offers a cost saving over the full-flow units, since only a portion of the influent needs to be pressurized. In both cases, however, if the solid particles in the feed stream are flocculated before introducing to the flotation tank, the high shear during pressurization, aeration, and pressure release can destroy the flocs. Also, if the particle loading in the feed stream is high, both systems are susceptible to blockage of the air release devices. To minimize these problems, recycle-flow pressure flotation is often practiced (Fig. 19-71). In this process, the feed stream, flocculated or otherwise, is introduced directly into the process vessel, and part of the clarified effluent is pressurized, aerated, and recycled to the flotation tank in which it is mixed with the flocculated feed. The air bubbles are released as they attach to the flocs and float to the tank surface. The recycle-flow devices are found to offer the highest unit capacities.

Figure 19-72 illustrates a dissolved-air flotation plant flowsheet for water treatment. The flowsheet shows that the incoming raw water is

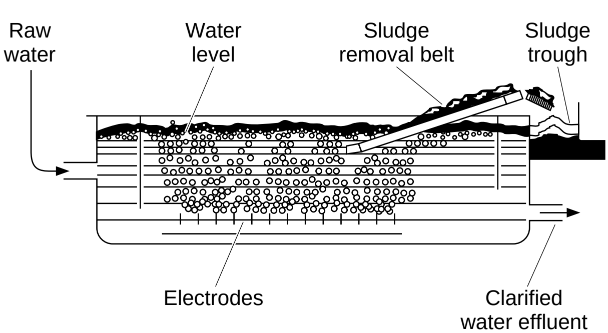

FIG. 19-70 Schematic diagram of an electrolytic flotation plant.

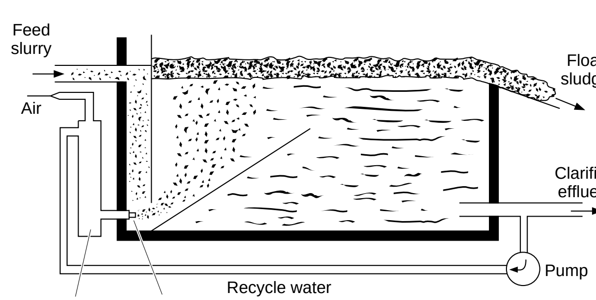

FIG. 19-71 Schematic diagram of a dissolved air flotation plant.

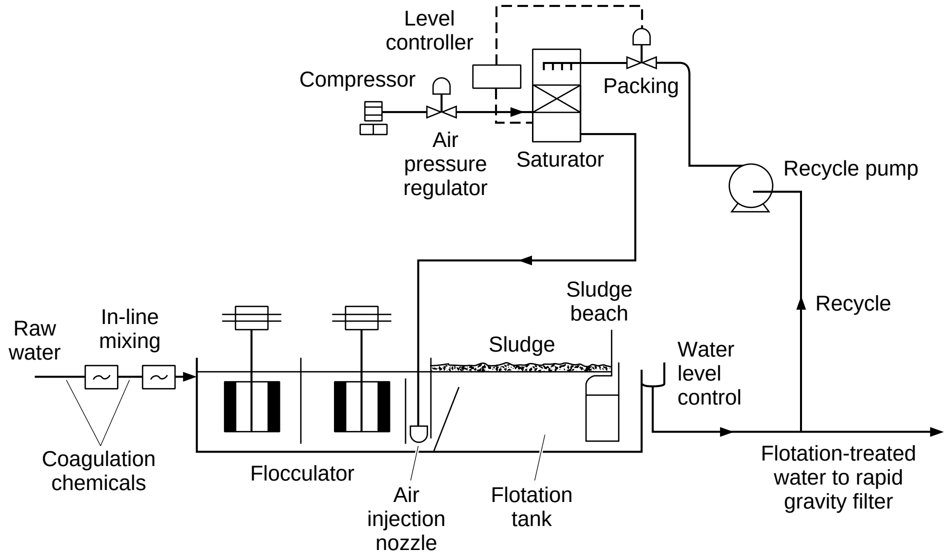

FIG. 19-72 Schematic diagram of a recycle dissolved-air flotation plant for water treatment.

conditioned with the addition of coagulation chemicals in a flocculator. This device pressurizes and aerates part of the treated water and recycles it to the flotation unit.

The dissolved-air flotation process is most commonly used for sewage and potable water treatment. It is also gaining popularity for the treatment of slaughterhouse, poultry processing, seafood processing, soap, and food processing wastes (Zoubulis et. al., 1991).

Dispersed-Air Flotation Units. Dispersed-air flotation involves the generation of air bubbles, either pneumatically or by mechanical means. In both cases, relatively large air bubbles (at least 1 mm in size) are generated. In order to control the size and stability of air bubbles, frothers are added to the flotation devices. These devices represent the workhorses of the minerals industry in beneficiating metallic and nonmetallic ore bodies and cleaning of high-ash and high-sulfur coals in which feed streams contain relatively high percent solids (5 to 50 percent by volume), and high throughputs are maintained (in excess of 4000 t/h). Handling of large quantities of solids in these flotation devices requires such special design considerations as maintaining the solids in suspension, promoting particle-bubble collisions leading to attachment, providing a quiescent pulp region below the froth to minimize pulp entrainment, and finally providing sufficient froth depth to permit washing and drainage of hydrophilic solids entering the froth region.

Mechanical flotation machines are most commonly used in the mineral industry, while pneumatic column-type units are gaining popularity in recent years. Surveys by Harris (1976), Young (1982), Barbery (1982), and Mavros (1991) provide a detailed overview of the process-engineering aspects of mineral flotation devices in particular and systems in general.

Mechanical Cells. Figure 19-73 presents a schematic representation of a typical mechanical device commonly known as a *flotation cell.* It is characterized by a cubic or cylindrical shape, equipped with an impeller surrounded by baffles with provisions for introduction of the feed slurry and removal of froth overflow and tailings underflow. The machines receive the supply of air through a concentric pipe surrounding the impeller shaft, either by self-aeration due to the pressure drop created by the rotating impeller or by air injection by means of an external blower. In a typical installation, a number of flotation cells are connected in series such that each cell outputs froth into a launder and the underflow from one cell goes to the next one. The cell design may be such that the flow of slurry from one cell to another can either be “restricted” by weirs or unrestricted.

The mechanical cells that are most widely used today in sulfide, coal, and nonmetallic flotation operations in the western hemisphere are made by Fagergren (by WEMCO Division of Envirotech Corporation), D-R Denver (by Denver Equipment Corporation of Sala International), Agitair (supplied by Galligher Ash Company), and Outokumpy (by Outokumpu Oy).

These machines provide mechanical agitation and aeration by means of a rotation impeller on an upright shaft. In addition, the Agitair and Denver cells also utilize air from a blower to help aerate the pulp.

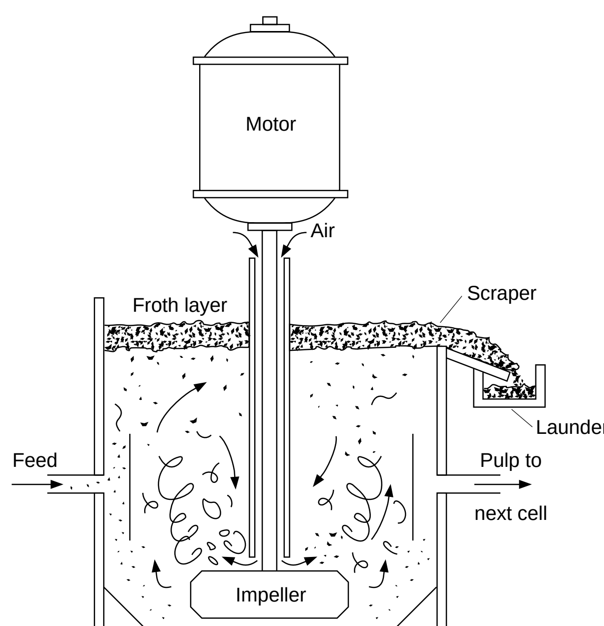

FIG. 19-73 Schematic of a mechanical flotation cell.

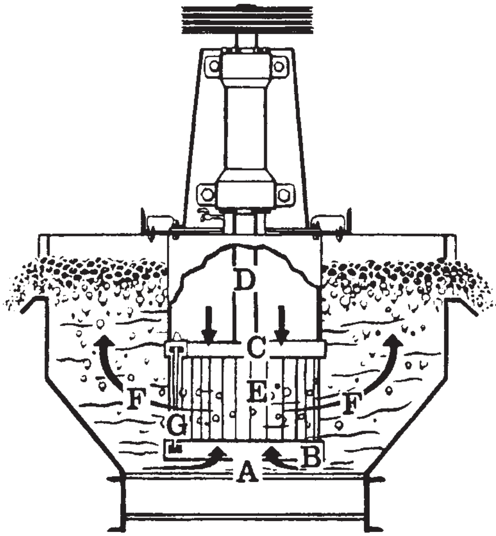

FIG. 19-74 Fagergren flotation machine.

In the Fagergren machine (Fig. 19-74), pulp is drawn upward into the rotor *A* by the rotor's lower portion *B.* Simultaneously the rotor's upper end *C* draws air down the standpipe *D* for thorough mixing with the pulp inside the rotor *E.* The aerated pulp is then expelled by a strong centrifugal force *F.* The shearing action of the stator *G,* a stationary cage fitting closely around the rotor, breaks the air into minute bubbles. This action uniformly distributes a large volume of air in the form of minute bubbles in all parts of the cell.

In the D-R Denver machine (Fig. 19-75), the pulp enters the top of the recirculation well *A,* while the low-pressure air enters through the air passage *B.* Pulp and air are intimately mixed and thrown outward by the rotating impeller *C* through the stationary diffuser *D.* The collector-coated mineral particles adhere to be removed in the froth product.

In the Agitair flotation machine (Fig. 19-76), the impeller is a flat rubber-covered disk with steel fingers extending downward from the periphery. A rubber-covered stabilizer eliminates dead spots in the agitation zone and improves bubble-ore contact. The degree of aeration is controlled by regulating air volume on each cell with an individual air valve. Air is supplied at 10×10^3 Pa (1.5 lbf/in^2).

Modern mineral-processing plants are being designed with capacities on the order of 500 to 1000 kg/s (2000 to 4000 tons/h). The unit capacities of flotation machines now being manufactured are 10 times greater than those in common use 15 to 20 years ago (Fig. 19-76). Examples of large flotation cells that are currently available on the market include Denver Equipment (36.1 m^3), Agitair (42.5 m^3), and Wemco (85 m^3). Larger-scale flotation machines offer advantages of lower installed cost, lower operating cost, and lower floor-space requirements. However, it should be noted that large flotation cells do not permit a reduction in the number of cells in a series. The use of large flotation cells does enable a fewer number of parallel rows and thereby permits a reduction in pumps, piping, and other auxiliaries.

Flotation Columns. Flotation columns belong to the class of pneumatic devices in that air-bubble generation is accomplished by a gas-sparging system and no mechanical agitation is employed. Columns are built of long tubes of either circular or square cross sections that are commonly fitted with internal baffling. They are usually 10 or even 15 m high with a cross sectional area of 5 to 10 m^2. Figure 19-78 presents a schematic of a typical flotation column unit. Inputs to the column include preconditioned slurry feed and air and washwater spray, which are introduced at about two-thirds of the height from the bottom, in the bottom region, and at the top of the column, respectively. The outputs are froth overflow, consisting of hydrophobic particles from the top, and underflow from the bottom of the column, carrying the nonfloatable hydrophilic particles. Flotation columns

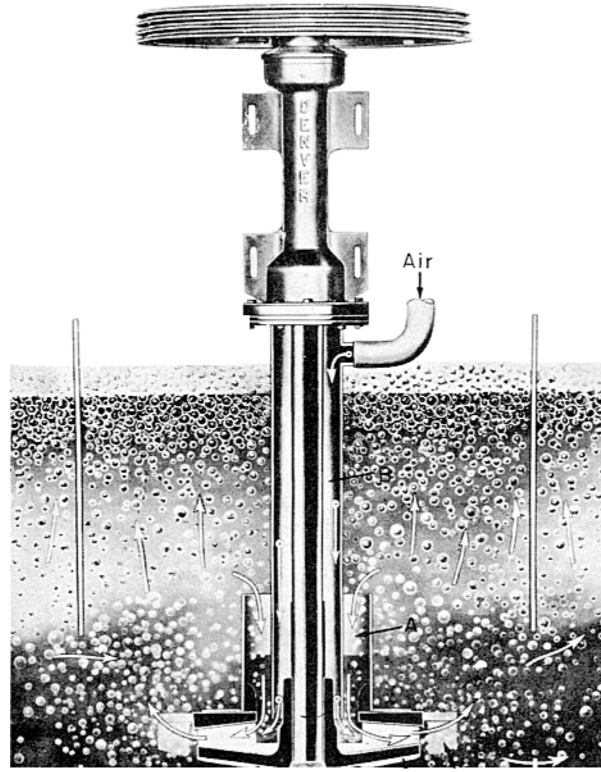

FIG. 19-75 D-R Denver flotation machine.

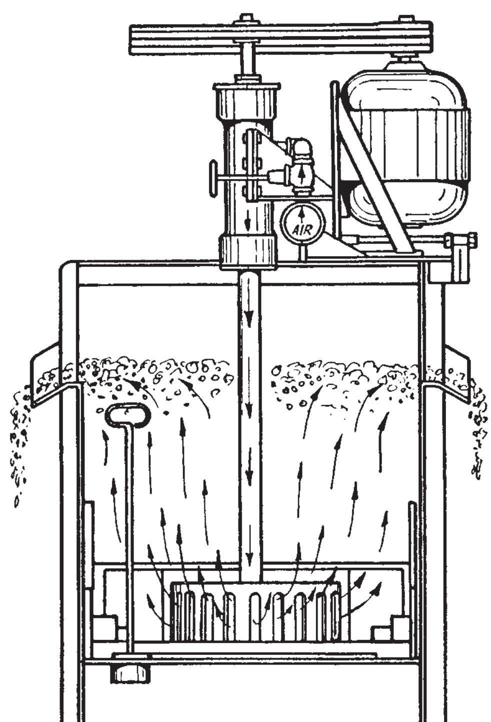

FIG. 19-76 Agitair flotation machine.

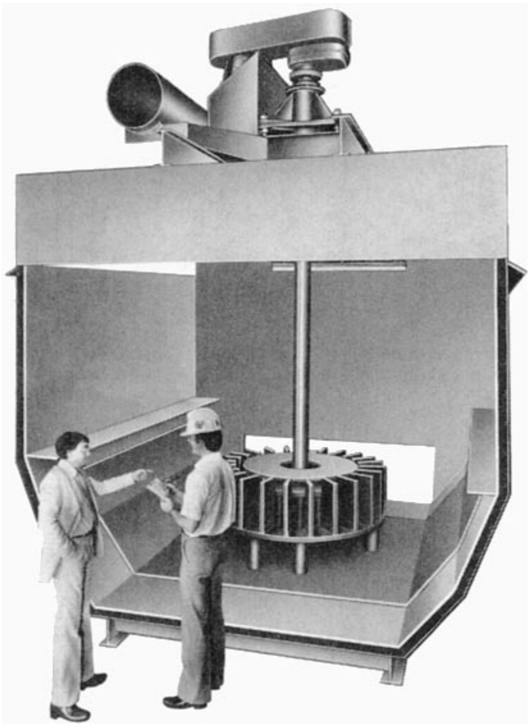

FIG. 19-77 Large flotation cell No. 165 AX 1500 Agitair, 42.5 m^3 (1500 ft^3). (*Courtesy of Caligher Ash Company.*)

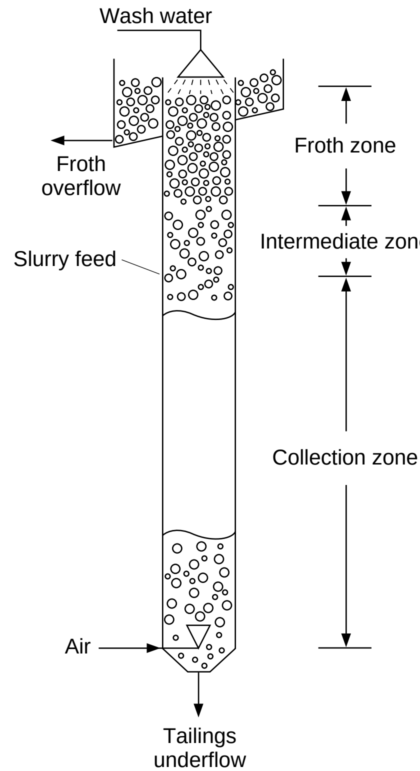

FIG. 19-78 Schematic of a flotation column.

make use of the countercurrent flow principle in that the swarm of air bubbles rises through the downward-flowing slurry during which time transfer of hydrophobic particles occurs between the slurry and bubble phases. The particle transfer process occurs in three distinct zones known as collection, intermediate, and froth zones. Properly designed baffles reduce short circuiting and promote better bubble-particle contact. Recovery of hydrophobic particles by the air bubbles takes place in the collection zone. Underflow removal rate and washwater addition rate are regulated such that there exists downward flow of slurry throughout the height of the column, thus ensuring that there is no bypass of the feed slurry in the upward direction. Further, the downward pattern of the flow of liquid helps in minimizing the entrainment of hydrophilic particles with the uprising air bubbles in the collection zone and in stripping the hydrophilic particles attached to the air bubbles in all three regions. All in all, the performance of the columns in terms of the recovery of hydrophobic particles and the grade of the froth concentrate is determined primarily by the slurry feed rate, air flow rate, and the surface area of the air bubbles.

Because of their inherently simple design, it is fairly common for flotation columns to be constructed in-house except for using the patented air-sparging systems. Several sparger designs are available that include simple porous plugs made from glass, stainless steel, and rubber, or more sophisticated venturi or in-line mixer configurations (Finch and Dobby, 1990). Some of the advantages claimed with flotation columns include improved separation performance, particularly for fine materials; low capital and operating costs; low plant floorspace requirements; and easy adaptability to automatic control. Flotation columns are being used in iron ore, copper, lead, zinc, and coal flotation applications and are expected to become even more popular because of their simplicity in construction and flexibility of operation.

Several modifications to the basic column design have become available over the years. Figure 19-79 shows three such designs. The first design variation is a packed column (Fig. 19-79*a*), which represents a minor variation to the basic column design in that it provides for corrugated plate-type packing. The packing feature enables uniform bubble size throughout the height, intimate particle-bubble contact, increased residence time of the slurry and bubble phases in the column, and a deeper froth zone (Yang, 1988). The Jameson cell (Fig. 19-79*b*), by contrast, is a combination column-cell design. It includes a vertical downcomer column in which the air and pulp are dispersed into a dense foam of fine bubbles creating a favorable environment for particle-bubble contact. The bubbly mixture is then discharged into a cell, which allows the separation of particle-laden bubbles from the pulp (Clayton et. al., 1991). Air-sparged hydrocyclone (Figure 19-79*c*) is a distinctly different design consisting of two concentric tubes with a conventional cyclone header at the top, providing for a tangential entry of the feed. As the feed slurry swirls down the inner porous tube through which air is sparged, collision between centrifuged particles and air bubbles takes place, leading to the recovery of hydrophobic particles. The bubble-hydrophobic particle aggregates are transported into the overflow stream as froth, while the nonfloating particles are removed with the underflow (Miller et al., 1988). Each of these and other design variations to the basic pneumatic flotation column concept is found to offer process improvements to specific applications.

Monographs by Sastry (1988), Finch and Dobby (1990), and Rubinstein (1994) provide an overview of the design and operational aspects of flotation columns.

FLOTATION PLANT OPERATION

Ores must be ground to a point of complete or nearly complete liberation. Even though this might possibly be accomplished by coarse crushing, grinding to finer than 10 mesh in all cases and finer than 48 mesh in most cases is necessary prior to flotation. Grinding is done in closed circuit with classifiers.

In many instances, superior flotation results are obtained by conditioning the ore with the reagents before the flotation step. Oily-type collectors are sometimes added to the grinding circuit to ensure dis-

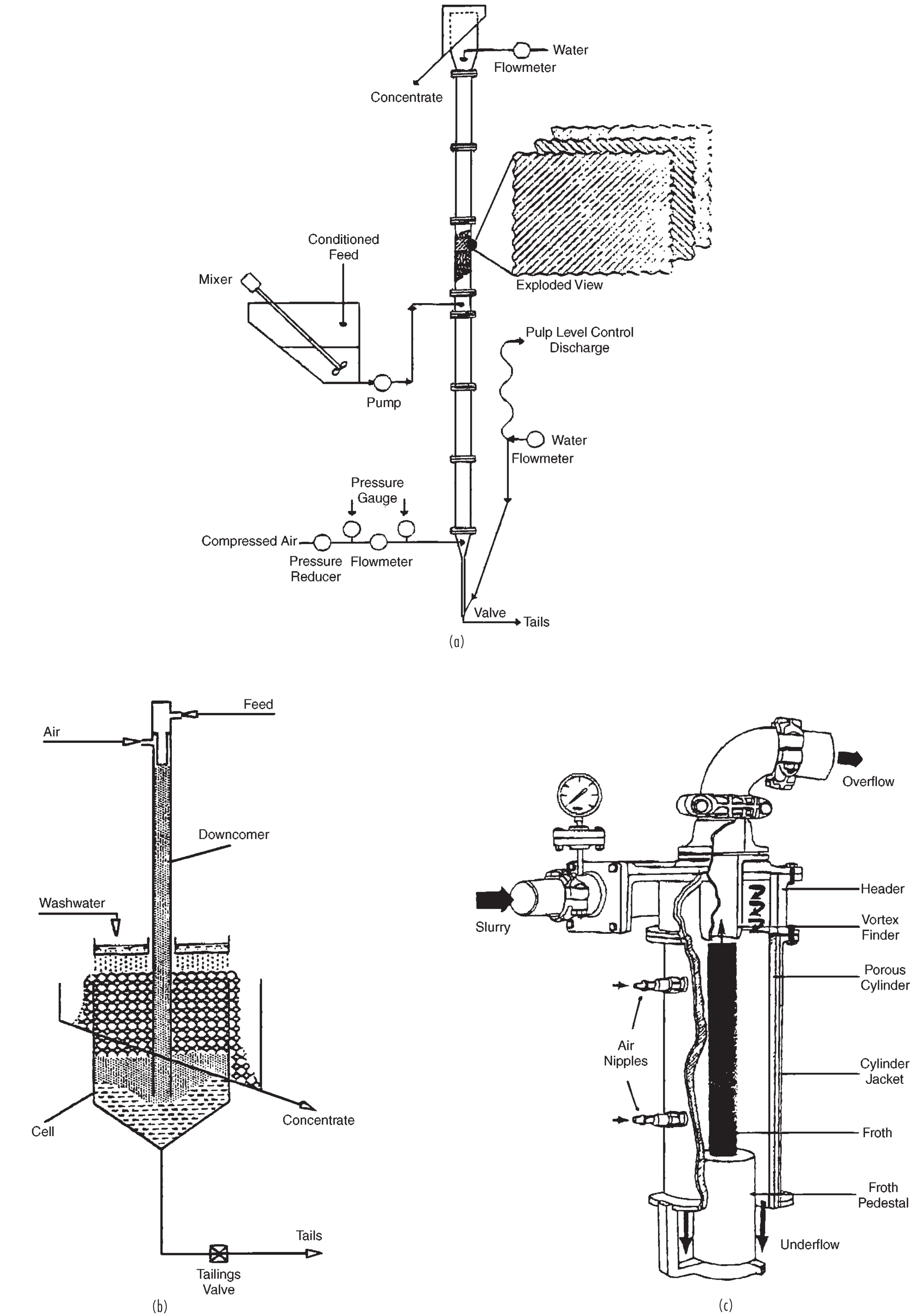

FIG. 19-79 Variations in the basic column design: (*a*) packed column, (*b*) Jameson cell, and (*c*) air-sparged hydrocyclone.

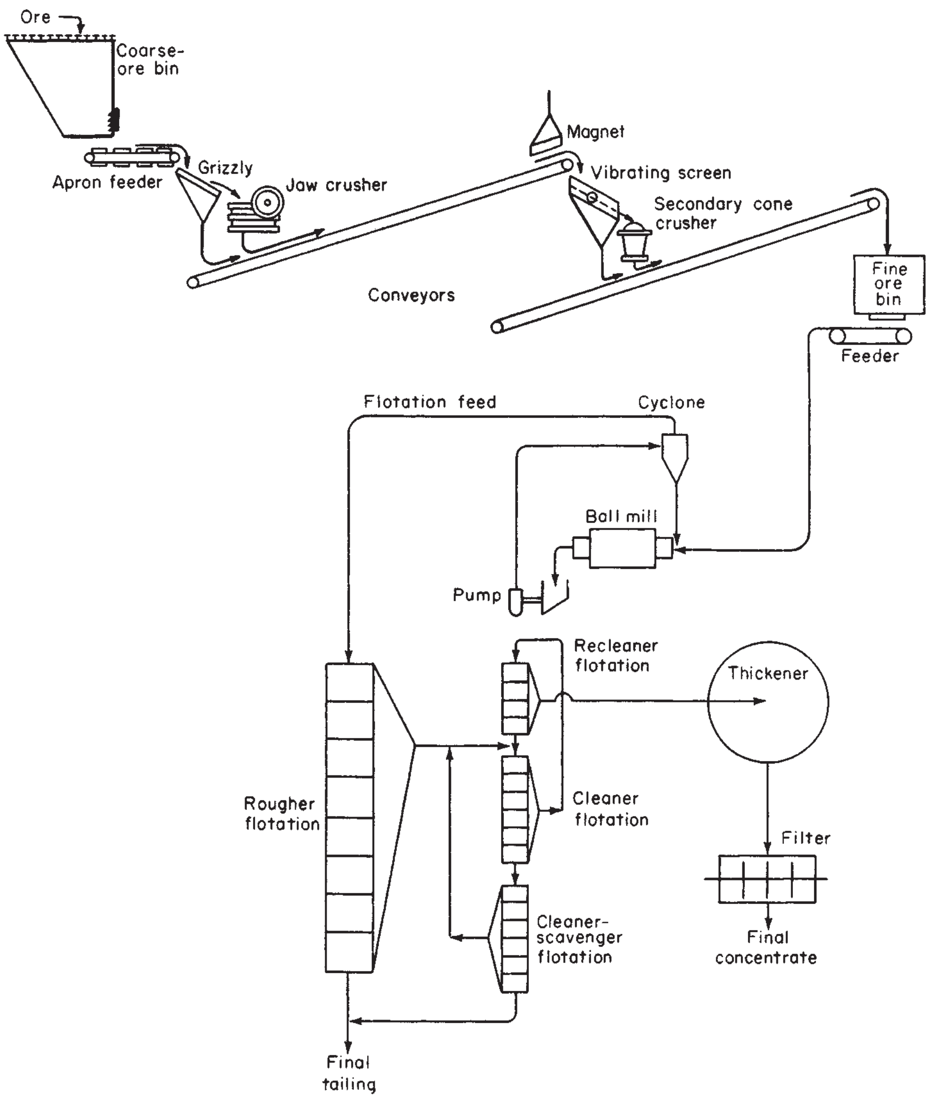

FIG. 19-80 Flowsheet of a typical flotation plant.

persion. For proper selectivity, a definite contact time is sometimes required between reagent and ore, and this is usually secured by mixing the reagent and the ore pulp in a conditioner consisting of a cylindrical tank with a vertical impeller.

Flotation machines are built in multiple units, and the flow of the pulp through the 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 low-grade concentrate. The concentrate is treated, sometimes after regrinding, in *cleaner* cells and *recleaner* cells for final concentration. The tailing 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.

Important auxiliary equipment in a flotation plant includes feeder and controls, sampling and weighing devices, slurry pumps, filter and thickeners for dewatering solids, reagent storage and makeup equipment, and analytical devices for process control.

Figure 19-80 is a flowsheet of a typical flotation plant.