MINERALS RECOVERY AND PROCESSING

1. Introduction

Minerals, critically important to the development and sustenance of civilization, are derived from the earth's crust, a relatively thin shell of siliceous material about 13 km deep. The art of minerals processing can be traced back to the beginnings of civilization as evidenced by distinct historic periods known as the Stone Age, Copper Age, Bronze Age, and Iron Age (see Table 1) (1). In the sixteenth century Agricola in his treatise *De Re Metallica* noted "The art of metal making is one of the most ancient, the most necessary and the most profitable to mankind. Without doubt, none of the arts is older than agriculture, but that of the metals is not less ancient; in fact they are at least equal and coeval, for no mortal man ever tilled a field without implements" (1). Mining can be considered the second largest industry in the world, next to agriculture. In terms of importance, mining is considered equal to agriculture. Its importance is well captured in the following statements (2): "If it can't be grown, it must be mined;" "Everything we have and everything we use comes from our natural resources." Every American born will need 3.6 million pounds of minerals, metals and fuels in a lifetime (see Fig. 1) (2). Minerals are used in most of the major industries in a variety of forms (see Table 2) (2,3).

The word mineral comes from *minera*, which once meant a mine or an ore specimen; it now has many meanings. Most commonly, it refers to *inorganic* substances (chemical compounds of elements and in a few cases native elements) occurring *naturally* in the earth. In a broad sense, however, mineral means any natural substance that is neither plant nor animal. Thus coal is described as a fuel or energy mineral. Modern meaning of mineral includes synthetic substances also. Minerals make up rocks and ores. Biotite granite, for instance, is composed of three principal minerals of different composition: light colored feldspar, grey quartz, and black mica (biotite). Magnetite, on the other hand, is virtually a mono-mineral, ie, composed of crystalline magnetite grains. Minerals are products of various physico-chemical processes occurring in the earth's crust including the life activity of various organisms. Although with some degree of approximation, natural minerals may be regarded as homogeneous crystalline masses, no mineral is perfectly homogeneous, either physically or chemically.

The occurrence of minerals or elements in the earth's crust is not uniform. Rather, minerals concentrate in particular areas (called deposits) as a result of geological conditions and activity. Such a deposit is referred to as an ore deposit, ore body, or simply as an ore when it is sufficiently large, ie, the percentage of useful components is high enough to make mining worthwhile for economic recovery of the mineral. Most minerals (or elements) are present in the earth's crust in very small amounts. The concentrations of selected metals are given in Table 3 (4). Eight elements, O, Si, Al, Fe, Ca, Na, Mg, and K, account for more than 99% of the crust. The abundance of a given element or a mineral often has no bearing on its industrial importance or its price (see also METALLURGY, EXTRATINE METALLURGY).

Ores which comprise a variety of minerals are as a rule, heterogeneous. An ore body is usually named for the most important mineral(s) in the rock, referred to as value minerals, mineral values, or simply values. Some minerals contain important metals (such as sulfide minerals containing one or more of Cu, Ni, Zn, Pb, Au, Ag, and platinum group elements), which are extracted from concentrated minerals using chemical methods such as leaching and smelting (see EXTRACTIVE METALLURGY). Other minerals (mostly non-sulfides) such as diamond, asbestos (qv), quartz (see SILICON COMPOUNDS), feldspars, micas (see MICA), gypsum, soda, mirabillite, clays (qv), etc. may be used either as-found, with some or no pretreatment, or as stock materials for industrial compounds or building materials (qv) (3).

Most ores as mined require some processing before they can be converted into usable metals or final mineral products. Processing of ores by physical or chemical methods is described as mineral processing. For convenience, it is often the practice to restrict minerals processing to physical methods, and to treat chemical methods under the realm of extractive metallurgy. This distinction is a fine one, however, and hydrometallurgy (see SUPPLEMENT), which comprises chemical methods of minerals treatment in aqueous medium, has much in common with minerals processing. Therefore, hydrometallurgy can often be considered as minerals processing. In such a broad sense, mineral processing comprises solid-solid separation (eg, flotation, gravity, magnetic etc.), solid-liquid (dewatering) and liquid-liquid separation (solvent extraction). Until the early part of the twentieth century, minerals processing was described as an *art* and focused primarily on engineering aspects rather than science. Rapid developments in the twentieth century in understanding of the fundamentals of processing have given minerals processing a sound scientific basis.

The size of the minerals industry is demonstrated by world production figures of the principal mineral commodities (Table 4) (5). The total tonnage of commodities listed (a mix of metals and minerals, and does not reflect tonnage of *mined ore*, which is much larger) amounted to approximately 11.7 billion metric tons for 2003 excluding petroleum. A conservative estimate of the total tonnage of earth moved in order to produce these 11.7 billion tons of mineral products is probably double that in one year. Total U.S mined tonnage of nonfuel materials in 2003 was ~5.5 billion tons. A large copper (qv) flotation (qv) plant alone treats from 100,000 to 270,000 t/d of ore. This ore may contain 0.3 to 1% copper value and may involve removing at least an equivalent tonnage in the form of overburden or waste rock. Vast amounts of energy, water, and chemicals are also used.

The minerals processing industry has made contributions to all areas of technology (see for eg, Table 2) in terms of scientific concepts, engineering principles, products and processing. Technologies developed in the mineral industry are used extensively in the chemicals industry as well as in municipal and industrial waste treatment and recycling industry, eg, scrap recycling, processing of domestic refuse, automobiles, electronic scrap, battery scrap, and decontamination of soils.

1.1. Ores

Deposits. Ore deposits (6) are classified based on a number of criteria, such as minerals or metals contained (eg, sulfides, oxides, metallic, nonmetallic), the shape or size of the deposit, host rocks, or the geologic processes that combined to form the deposit (ie, igneous, metamorphic, or sedimentary). In terms of processing technology, industry drivers, and end use products, a broad classification of ores into sulfide and non-sulfide ores is quite convenient (Fig. 2). Oxides can encompass oxides, hydrated oxides, sulfates, carbonates, and silicates. Sulfide ores contain metallic minerals and are processed to produce a metal as a final product. Non-sulfide ores or oxide ores contain largely non-metallic minerals. Although several of these are used for metal production (eg, bauxite for aluminum, iron ore for iron and steel, etc.), vast amounts of these are processed to make specific mineral products for end-use (eg, kaolin clays for paper coating, asbestos, titanium minerals for pigments, etc.). Considerable effort goes into estimating the size and geometry of ore deposits. These are important factors in economic assessment. The size of ore deposits is described in terms of proven or measured reserve which is well-outlined by extensive drilling; indicated reserve, implied by a limited amount of drilling; and potential reserve, based on geological data with no drilling information. Potential reserves could also include well-outlined deposits that are not economical to treat at present.

Ores are mined by open-pit, underground, alluvial, and solution mining methods. The first two are forms of hard rock mining. Open-pit mining is preferred because it is more economical and practical, especially when the ore grade is low and large tonnages have to be handled. It can, however, be practiced only when the ore body is close to the surface. Underground mining, the most expensive method of mining, has to be used when the ore body is at a considerable depth from the surface. Often a combination of open-pit and underground mining is used on the same ore body. Alluvial and solution mining are inexpensive, but can be used only on sedimentary, alluvial, or placer deposits such as river beds, dried-up oceans, and lakes. Dredges or water pressure are used to extract the ore. Unlike subsequent processing, the mining method has comparatively little dependence on the mineralogical nature of the valuable minerals in the ore, though the nature of the host rock may affect the actual method of hard rock mining used.

Mineralogy. Ores are inherently heterogeneous and unique. Any given element can exist in a variety of compounds or minerals of distinctly different composition. There is considerable variation in minerals composition within an ore body and from one ore body to another. For the purposes of minerals processing, any ore can be considered to be made up of valuable minerals and gangue (waste) minerals, and these are associated with one another intimately in the deposit. The valuable minerals may contain more than one valuable metal.

Ores may be designated simple or complex, depending on the ease and extent of mineral liberation and subsequent processing. The choice of a processing method and its efficiency are entirely dependent on ore mineralogy (8–10), ie, the specific minerals present, their relative proportions, composition, and the mode of physical occurrence of the various mineral constituents such as size and extent of liberation. A thorough knowledge of mineralogy is also important in monitoring the efficiency of day- to-day operation of the mill because mineralogical factors can change frequently. Often the chemical analysis of the ore may remain essentially the same, but the mineralogy may differ vastly. A simple example is that of a sulfide copper ore in which Cu is typically distributed amongst several sulfide minerals of varying amounts such as chalcopyrite, chalcocite, covellite, bornite, and sometimes significant amounts of non-sulfide (or oxide) minerals such as malachite, cuprite, chrysocolla etc. All of these minerals have their unique properties and responses to a given separation method such as flotation; for eg, the oxide minerals may not be recovered in the sulfide flotation circuit. Mineralogical information is critical in the optimization of processing steps such as grinding and concentration; without this it may be impossible to determine whether a poor separation is the result of inadequate grinding, mineralogical characteristics, or inefficient plant operation.

Mineralogical information is routinely gathered by collecting representative samples of the ore, or of the various flow streams in an operating plant, and studying polished or thin sections of these samples using optical microscopy (qv) in either the reflected light mode or the transmitted light mode. The electron microprobe and x-ray diffraction are also used extensively in conjunction with the optical microscope (see X-RAY TECHNOLOGY). More recent developments, such as QEM*SEM or QEMSCAN (11) and MLA (12), are fully automated image analyzers using EDS X-rays and back-scattered-electron signals in a scanning electron microscope. They are rapid and provide quantitative information on bulk mineralogy, detailed textural mapping of particles, and liberation characteristics. The type of information sought in mineralogical examination includes the grain size of the value minerals in the host matrix rock; associations or locking between the value minerals and gangue minerals and between value minerals; existence of trace elements in the lattice of the value minerals; presence of oxidation or alteration layers on mineral particles (say 0.5 μm or thicker); and the occurrence of minor amounts of potentially valuable metals or minerals. In the reflected light mode, mineral identification is made using many optical properties of the minerals, most notably color. Polarized light is used for obtaining additional information and confirmation because this reveals anisotropy of minerals. Chemical etching or staining techniques exploit differences in chemical properties of the minerals, and these can also reveal texture, cleavage, or the existence of trace elements. Dark-field illumination is used to obtain information on translucent minerals. Transmitted light microscopy is useful for the study of many non-sulfide minerals, such as silicates, which are difficult to identify by reflected light. Heavy liquid fractionation using organic and inorganic liquids of various specific gravities may be invaluable for physically separating the minerals in the laboratory to assess the degree of particle locking as well as the potential for separation by gravity concentration methods. Ferromagnetic fluids are also used for separation based on specific gravity (13). There are also physical separators, such as the Super-panner and Hydroseparator (14), which make fine separations based on size and specific gravity of mineral particles.

Liberation. In most ores, the value minerals are intimately locked or associated with gangue minerals. One of the most important prerequisites to the success of any ‘physical separation process is liberation of value minerals from the host rock (or ore matrix). Liberation, achieved by size reduction (qv) operations, can occur either because of intergranular or transgranular fracture (7,15,16). For some ores, such as kaolin clays, liberation is achieved by breaking up of aggregates in a concentrated slurry using high-intensity and high-shear agitation. The degree of liberation of a particular mineral species, ie, the percentage of that mineral occurring as free particles (in a macroscopic sense) in relation to the total quantity of the mineral is dictated by the type, performance, and economics of the processing operation, ie, throughput, recovery, product grade, processing costs, and profits. In many plants liberation and processing are performed in stages to optimize efficiency and economics. For example, coarse-grinding-separation is followed by fine-grinding-separation. Thus many concentration units deal with a significant proportion of middlings or composite particles which contain varying amounts of one or more value minerals still locked or associated with one or more gangue minerals. As can be expected, middlings decrease the sharpness of any separation. Both grade and recovery could be affected depending on the type of separation. Recovery of middlings constitutes one of the primary challenges in concentration separation. Middlings can be characterized by many types, but some of the common types are the occurrence of one mineral as veins, blebs, or inclusions in another mineral, or as two large areas of minerals making contact with each other. Mineral liberation has been studied extensively, and numerous liberation models have been developed (16–18). Microscopy of polished sections of sized fractions of crushed or ground ore is valuable and necessary in assessing the degree of liberation. This is at the heart of minerals separation.

1.2. Processing

Flow Sheets. All minerals processing operations function on the basis of a flow sheet depicting the flow of solids and liquids in the entire plant (7,18,19). The complexity of a flow sheet depends on the nature of the ore treated, the degree of liberation achieved, the specifications for the final product, and the economics of the operation. The basic operations in a flow sheet are: size reduction (qv) (comminution) and/or size separation (see SEPERATION, SIZE), solid-solid separation (minerals separation), solid-liquid separation, and materials handling. A schematic of the major unit operations and options in mineral processing is shown in Fig. 3. A highly simplified schematic flow sheet for a copper ore operation, from mined ore to metal, is shown in Fig. 4. The overall flow sheet depends on whether the specification for the final mineral product is size, chemical composition, ie, grade, physical properties of the mineral product, or combinations of these. Products from a quarry, for example, may have a size specification only; metal sulfide concentrates have a grade specification; and kaolin clays must meet both.

Each basic operation can be divided into one or more unit operations. Size reduction involves crushing and grinding depending on the size of material handled, and these may be carried out in stages. Size separation or classification is an integral part of any flow sheet, not only to meet product size specifications, but also to ensure a narrow size distribution for subsequent minerals separation circuits and to decrease the load and improve the efficiency of size reduction units which are energy intensive. Concentration separations can be either solids from solids, based on size or mineral composition, or solids from liquids, ie, dewatering (qv). [Liquid-liquid separation (ie, solvent extraction) occurs after the value mineral has been dissolved in an aqueous medium. This process separates and concentrates (into a small volume) one value metal from others in solution. Liquid-liquid separation is covered in HYDROMETALLURGY].

Concentration separations, such as gravity concentration, flotation (qv), and magnetic separation (see SEPARATIONS, MAGNETIC SEPARATION) are carried out most frequently in stages called roughing, scavenging, and cleaning. Roughing, the first stage in a separation, produces a concentrate (value product) and tails (rejects). Rougher tails are treated in scavengers to recover additional mineral values. Scavenger tails constitute the final plant tails or feed to another separation circuit. The rougher and scavenger concentrates can be combined or treated separately. For many ores, the scavenger concentrate represents the middlings or unliberated or partially liberated particles and, therefore, may require further grinding before treatment. Even the rougher concentrate may require further grinding before it enters the cleaning circuit. There may be additional stages such as cleaning of a scavenger product, scavenging of a cleaner product, or recleaning of a cleaner product, depending on the metallurgical objectives of the plant and the type of mineral product produced.

In addition to encompassing all of the unit operations in the plant, the plant flow sheets may also include materials handling operations associated with the transport and storage of materials in and around the mill. Typically, flow sheets provide quantitative information regarding water and slurry flows, tonnages, and assays; flow sheets form the basis for process control and automation.

Metallurgical Performance. Most minerals separations, whether by size or composition, are not perfect. There are invariably misplaced values and gangue minerals. The performance of a separation unit is, therefore, described by both recovery and grade (akin to yield and purity in chemical industry) (4,13,20). In addition, other parameters such as ratio of concentration, ie, ratio of weight of feed to weight of concentrate; enrichment ratio, ratio of grade of concentrate to grade of feed; economic efficiency (each plant can have its unique model); and separation efficiency, recovery of value minus recovery of gangue, are also used sometimes. Recovery is a measure of the efficiency, and the grade the measure of selectivity of a unit operation. Grade is expressed in a variety of ways including physical, eg, brightness of clays (qv) or of the calcium carbonate used in the paper industry, or chemical properties, eg, metal assays of concentrates or P_2O_5 content of a phosphate concentrate. Recovery and grade are not independent, and there is a tradeoff at every stage of the separation operation. Recovery is the primary objective in both roughers and scavengers; grade is the primary objective of cleaners. Each recovery/grade combination is associated with a cost. Thus the overall performance of a separation process can be represented as a triangle formed by recovery, grade and cost. Such a triangle implies a trade-off among the three drivers or goals, and is fundamental to minerals separations.

Material Balances. Material balances in the plant flow sheet provide an assessment of the efficiency of operation of individual unit operations and the plant on the whole (4,13,21). Information necessary for trouble-shooting is also provided. As for any process, materials balance involves calculation of input, output, and accumulation around any given circuit of a flow sheet. Many unit operations are characterized by high accumulations (circulating loads), sometimes by design. Most units are described as separation or junction points (nodes). Materials balance equations are presented as two-, three-, or n-product formulas depending on the number of products a flow stream consists of or is split into. The bases for materials balance are usually mass, assay (grade), volume, mineral composition, or energy. Balancing a circuit is usually quite a tedious task. There is either inadequate information about flow streams (leading to errors) or too much data (leading to conflicting results). Many computer programs have been developed to handle a complex set of nodes and inconsistent or excess data that typically exist in a plant (16).

There are many sources of errors in the plant. The principal ones are related to sampling (qv), mass flow rates, assaying, and deviations from steady state. Collecting representative samples at every stage of the flow sheet constitutes a significant task. Numerous methods and equipment are available for sampling (13,21,22). Sampling: is this a section in Encyclopedia?

1.3. Economic Aspects. The role of nonfuel (ie, coal and petroleum are excluded) minerals in the U.S. economy for 2003 is given in Fig. 5 (23). The value of mineral products mined in the U.S was $40.5 billion, but after processing the value was $432 billion, which is more than ten times. Value added to GDP by major processed-minerals consumer industries was $1.7 trillion. The U.S. mining industry uses about 290 trillion BTUs of energy (24).

The economics of minerals processing and production (7,25) are determined not only by supply and demand, but also by socio-political factors, including those tied into energy supply and environmental aspects, and by the need for a minimum product quality. The use of a mineral product has to justify all costs involved in producing it. Those costs include mining, processing, transportation, and marketing, as well as costs associated with meeting environmental regulations.

Mining of the ore deposit constitutes a significant cost, especially in hard rock mining. Mining costs vary considerably from ore to ore and from a few cents to well over $100/t mined. Underground mining is the most expensive; hydraulic mining of sedimentary deposits is the least expensive. Processing costs include those for size reduction, size classification, minerals concentration and separations, solid-liquid separation (dewatering), materials handling and transportation, and tailings disposal. Size reduction, one of the most expensive unit operations in minerals processing, could account for as much as 30–50% of the total energy consumed. An example of the distribution of power consumption in a separation plant, such as flotation, is given in Fig. . This cost varies considerably from deposit to deposit and quite often from one area of a deposit to another. Ore bodies are extremely heterogeneous and the associated minerals liberation is complex.

Minerals concentration operations also constitute a primary cost item. The grade (or value content) of the ore deposit has a strong influence on the size of the processing operation. When using a lower grade ore, a large-scale operation is required to produce a given amount of product economically. Operating cost per ton of ore treated is smaller for large-scale operation. Physical methods of separation consume less energy than do chemical methods. The location of the mineral processing plant or mill affects the costs of fuel, power, water, transportation, and labor, as well as the cost of the mill itself. Mills and tailings pond are usually located close to the mine site, although environmental issues and land resource management may require pumping of the plant tails to considerable distances from the plant.

The valuation of mineral products is complex because it is influenced by composition, which inherently lacks uniformity. credits are given for desirable features in the mineral product and penalties for undesirable features. For example, in a metal sulfide concentrate, credits are paid for contained gold and silver even if the amounts are very small, and penalties are paid for contained toxic elements such as As and Sb, or detrimental elements such as Mg and F which can cause problems or metal losses in smelting. Government legislation and environmental regulations are also becoming increasingly important in product valuation.

The economic importance of an ore deposit itself is largely affected by mineral or metal prices. Mine closures and re-openings are a common event in the mineral industry for this reason. Economics can also be affected by the ore composition, for example, by unacceptable levels of penalty elements in the ore. The assessment of overall economics of exploiting a given ore deposit is similar to that for any large-scale industry. The various cost components are those associated with equipment, labor, utilities, contingencies, operation and production, transportation, working capital, supplies, maintenance, depreciation, and increasingly important mine and tailings decommissioning and closure. Environmental awareness, concerns, and regulations have had a significant impact on mineral industry economies in the latter half of the 20th century and are expected to continue to play an increasingly important role in the twenty-first century. Similarly, water resource management will be a critical issue facing the industry in the near future. (see also ECONOMIC EVALUATIONS; ENERGY MANAGEMENT).

1.4. Environmental Aspects. Environmental and Safety issues, concerns, and management are a primary preoccupation of the mining and minerals industry. The overarching goal is to minimize the impact of the industry to the world ecosystem. The mining and mineral industry continue to face and deal with increasingly demanding challenges and liabilities from regulations pertaining to environmental matters which periodically change and increase in complexity.

Principal areas of environmental management (27,28) in minerals processing are: treatment and safe disposal of tailings; effluents and other waste; residual reagents in mine effluents; protection of ground and surface waters; acid mine drainage, land reclamation, restoration and abatement, mine subsidence, mine closure, dust control, air emissions, and the elements or minerals that cause environmental pollution in subsequent operations such as leaching, smelting or burning of coal. Some of the toxic elements of notoriety are As, Bi, Sb, Cd, and Hg. Diesel fuel, which is extensively used, is under scrutiny in many operations because of health hazards associated with it. Regulations such as the Clean Air Act have high impact on the industry. Laws in the 1990s required that permits for new mine openings be covered by a huge performance bond. The release of the bond is contingent on the completion of all required reclamation, restoration, and abatement work on the permit area.

A large amount of chemicals is used in minerals processing in a variety of separations, and many environmental regulations have been in effect for a long time for these chemicals. Regulations that govern the introduction into commerce of new chemicals are becoming more stringent, and each country has its own unique set of regulations (Ch.8 in Ref. 29, but updated). The legislation is different from country to country but outlines the tests (chemical/physical properties, mammalian toxicity, aquatic toxicity, mutagenicity and environmental fate) that are required in order to add a chemical substance to a country's inventory. Each distinct chemical species, including impurities, in the product "as sold" must be qualified. The following countries have implemented chemical control law legislation for new chemical substances and have established national chemical inventory lists of those chemical entities which have been approved for manufacture and/or import: USA (TSCA); Canada (DSL); EU (EINECS/ELINCS); Australia (AICS); Korea (ECL); Japan (ENCS; also known as METI); Philippines (PICCS) and China (IECSC). The cost of registering a chemical varies from country to country, but at a minimum it is ~\$65k per chemical for mutagenicity, acute mammalian toxicity, skin and eye irritation, aquatic toxicity, and preliminary environmental fate testing. EU requires more extensive testing linked to production volume, and typical costs are EU \$175–300k for each chemical. Testing Requirements for Europe will satisfy most other countries. Total cost for the above 8 countries could be as high as ~\$2 million per chemical.

A significant amount of research and development is being conducted to address the environmental issues in mineral processing. Many newer technologies have already become available.

1.5. Size Reduction. Size reduction (qv) or comminution is the first and very important step in the processing of most minerals (4,7,13,17,18,30–32). It also involves large expenditures for heavy equipment, energy, operation, and maintenance. Size reduction is necessary because the value minerals are intimately associated with gangue and need to be liberated, and/or because most minerals processing/separation methods require the ore mass to be of certain size and/or shape. Size reduction is also required in the case of quarry products to produce material of controlled particle size (see SIZE MESUREMENT OF PARTICLES). In some instances, liberation of valuables or impurities from the ore matrix is achieved without any apparent size reduction. Scrubbers and attritors used in the industrial minerals plants, eg, phosphate, rutile, glass sands, and clay.

Size reduction is conducted in stages, the first occurring during mining of the ore body by using either explosives, eg, for hard rock mining or mechanical means, eg, for hydraulic mining of sedimentary deposits such as clays (qv) and phosphates (see EXPLOSIVE AND PROPELLNTS). Subsequent size reduction stages involving crushing and grinding are grouped according to the particle size. General types of size reduction equipment include crushers, tumbling mills, impact mills, fixed-path mills, and fluid energy mills. A wide variety of crushers and grinding mills has been developed as necessitated by the ore type and rock hardness. The extent of size reduction achieved by any of these units is described by the reduction ratio, the ratio of the feed size to the product size. Particle size is defined as the size of the separator through which a certain percentage by weight, typically 80%, of the particles pass (referred to as d_{80} or p_{80}).

The three basic types of size reduction circuits used to produce a fine product are shown in Fig. 7 (7). The final stages of the grinding circuit are typically operated in closed circuit, at comparatively high circulating loads, so that the material has little chance of being broken a second time before it is removed from the circuit by a classifier. Rod mills are operated normally in an open circuit.

Size reduction is the most energy intensive unit operation in minerals processing. This is because most of the input energy is absorbed by the machine itself in vibration, generation of heat, etc, and only a small fraction (<1% to a few per cent) of the total energy is available for breaking the rock. The probability of breakage in comminution decreases, and the energy required per unit mass increases rapidly as the particle size decreases. In the late 1970s, approximately 29 billion kWh/yr (∼99 trillion BTUs) of electrical input was used for size reduction and an additional 3.7 billion kWh/yr in contained energy consumables such as grinding media and liners (31). This is roughly one-third of the total energy used in the mining industry. In some operations, energy used in size reduction could be as much as 30–50% of the total energy used. Of the total costs associated with size reduction, contribution from energy costs is typically in the range of 50–60%.

Crushing. Crushing handles the mine product (run-of-mine ore) and reduces the rock size from over 1 m to approximately 10–25 mm, often in stages termed primary, secondary, or tertiary. Crushing, carried out by compression or impact methods, is usually conducted dry, but water-flush crushers are emerging as a newer technology (30). There are many types of crushers: jaw, gyratory, roll, rotary, and impact. All are typically massive. The jaw and gyratory crushers dominate the primary crushing stage which is operated in open circuit, often in conjunction with scalping screens or grizzlies which remove undersize. In all crushers the energy is applied directly to the particles.

Jaw crushers are characterized by a fixed and moving jaw (Fig. 8a) (4). The latter allows for changes in the feed and discharge openings. Common types are the Blake crusher, which has the swing jaw pivoted at the top and a variable discharge opening; the Dodge crusher, used only in the laboratory, which pivots at the bottom and has variable feed area; and the Universal crusher, which pivots at an intermediate position and has variable feed and discharge areas. Jaw crushers are rated according to receiving areas, ie, the width (150–2100 mm) of the plates, and the gape, the distance (125–1600 mm) between the jaws at the feed opening. A large crusher can weigh up to 200 tons. A machine of size 1680 mm (gape) by 2130 mm (width) can crush ore (1.2 m) at a rate of 725 t/h at a 203-mm set (discharge opening) (4,33).

Gyratory crushers (Fig. 8b) (4) consist of a long spindle seated in an eccentric sleeve, housed in a fixed conical shell. Material is crushed as it gets nipped between the rotating spindle and the fixed crushing shell. The gyratory crusher has a much higher capacity than a jaw crusher and can be considered a continuum of jaw crushers. Common types are the suspended spindle (short or long shaft), the supported spindle (most used), and the fixed spindle. Large crushers having 0.2–1.0 MW (250–1300 hp) motors, weighing between 150–600 tons, have a gape of 1000–3000 mm, and can crush ore at a rate of 500–12000 t/h depending on the gape and discharge setting (13,18,33).

Cone crushers, which are modified gyratory crushers, perform the bulk of secondary crushing. These have a flat crushing chamber giving higher capacity and reduction ratio, a short spindle operated at much higher speeds, and a higher capacity. They are classified into standard (secondary) cone crushers, short-head (Symons design, tertiary crushing), and fine crushers (such as Gyradisc, Hydrofine, etc) (4,13). Crusher sizes are reported as the crushing-head (mantle) diameter; typical range is 610–3050 mm. Weights vary in the range of 5–200 t and connected power ranges from 7–500 kW (10–700 hp). A

3050-mm Symons Standard cone crusher can crush ore at a rate of up to 3000 t/h at a discharge setting of 64 mm (18).

Other types of crushers such as rolls and impact crushers (13), ie, hammer or stamp mills, have limited application in metal mining, but are more common in the quarrying industry, especially for relatively soft, friable, and sticky rocks such as phosphate, limestone (see LIME AND LIMESTONE), clay, asbestos (qv), coal, etc. These are characterized by larger wear and smaller (250−1000 t/h) capacity than the gyratory crushers. Some of the more recent crusher designs include the Tidco Barmac crusher, high pressure or high compression rolls, and the rotary coal breaker. High pressure rolls, for example, were introduced in 1985 and are used extensively in the cement (qv) industry. They are finding rapid acceptance in hard rock industry. They are reported to provide significant energy savings, from 20% to 50% depending upon configuration in the circuit, and considerably reduced wear of rolls, because they exploit interparticle crushing. The largest units are capable of using 100 kN of pressing force per cm of roll width and handling over 1000 t/h. Water flush cone crushers are an example of a wet crusher (product is 30−50% solids) (4).

Chemical and electrical, such as spark and microwave, methods of size reduction have also been suggested. Considerable effort has been made in crushing circuit control and automation to reduce capital and operating costs and to improve crusher efficiency (17,18,31). Larger crushing units are also used increasingly because these are considered to be more energy efficient and to provide savings in capital costs. Large gyratory crushers have replaced jaw crushers in many applications. Another trend in crushers is to provide mobile (in-pit) crushers to enhance flexibility (30). Continued improvements in raw material extraction and transportation have advanced the use of in-pit crushing. The use of mobile crushing units close to the working face in the mine provides efficient and versatile systems. The application of in-pit crushing significantly reduces the environmental concerns caused by truck haul operations.

Grinding. Grinding, which refers to size reduction of the crushed material, is carried out in stages, if necessary, until the required liberation is achieved (4,7,13,17,18,30–32). This liberation is dictated by the mineralogy of the ore deposit, the processing method, and prevailing economic conditions. Grinding is achieved by abrasion and impact in tumbling mills (Fig. 9) (4) using steel rods or balls, ceramic pebbles, or large pieces of the ore itself. The latter process is called autogenous grinding. Crushing circuits often provide media for autogenous milling. A variation used more commonly is the semiautogenous grinding wherein a small number (usually 10% of the total loading by weight) of large steel balls is used to break down certain intermediate sizes of particles that otherwise would build up in the mill.

Tumbling mills are classified into ball mills or rod mills, based on the type of tumbling media used. Tumbling mills vary considerably in size, autogenous mills being the largest, ie, 4−13 m diameter and 3−10 m in length. Nominally ball mills have a length-to-diameter ratio of 1.5:1 or less, and rod mills have a ratio 1.5:1 or greater. Ball mill sizes range from 1.5m × 2.4 m with 75 HP to 7.9 m × 12.5 m with 20,000 HP. Grinding balls range from 19 to 127 mm in diameter. Grinding rods range from 38 to 100 mm in diameter and 3 to 6 m long. Grinding media occupy less than half the volume of the mill and could weigh

5–500 tons. Typical drive motors have horse power in the range of 75 to 8000 or greater. Some of the large autogenous mills have 35,000 hp motors and handle 38,000 t/d of ore (33). Tumbling mills are lined with suitable abrasion-resistant material to minimize wear of the mill shell and to reduce slip between the shell and the grinding media. Liners are replaced periodically. The most commonly used materials are cast or rolled steel and rubber. Grinding media are usually forged or rolled high-carbon or alloy steel, or cast alloy steel, and consumption varies between 0.1 to 1 kg/ton of ore and liner wear is 0.05 kg per ton of ore ground (13). High chrome steels (>7% cr) and other high quality steel media are more expensive, and are used in niche applications because of their high wear and corrosion resistance. Normal feed size to a ball mill is 80% passing 6mm or finer for hard ores and 80% passing 25 mm or finer for soft ores. Product size is typically 420 μm or finer. As in the case for crushers, the trend in grinding is increasing the size of grinding mills. Much effort has been made in grinding circuit control and automation (17,18,31).

Grinding is a continuous process. It may be either wet or dry, but wet grinding (at 50–80% solids) is most common and requires less power per ton of material ground; however, media consumption is significantly higher. Wet grinding is also the choice when subsequent processing is wet. In certain instances, depending on use, dry grinding is necessary, especially in the case of certain industrial minerals or cement (qv). The feed to a tumbling mill enters through a hollow trunnion at the center of the feed end, and the product is discharged at the other end in a variety of ways, ie, hollow trunnion, grate discharge, peripheral discharge, etc. Particles are broken as a result of a combination of cascading and cataracting (free fall) motions of the media. At high tumbling speeds, cataracting dominates, and a large part of grinding occurs by impact; at lower speeds, cascading dominates, and grinding is by attrition. Typically, particles between 5 and 250 mm are reduced to between 10 and 300 μm. In rod mills the tendency is for large particles to be broken selectively and the fine particles to be passed selectively, whereas ball mills tend to grind both large and small particles. In tumbling mills most of the energy is consumed in keeping the mill shell, the media, and the mineral mass in motion; fracture occurs as a by-product of passage through the mill and is a statistical process. The average energy consumption for grinding can be about 11.6 kWh/t compared to about 2.2 for crushing and 2.6 for flotation (4). As a result of wear, all tumbling mills have a range of media sizes. Makeup media are added regularly. Wear is significantly greater in wet grinding. Autogenous and semiautogenous mills have a distinct advantage over conventional rod and ball mills in terms of savings in media consumption.

Other types of mills include fluid energy mills, vibratory mills, centrifugal mills, tower mills, roller mills, and attrition mills (4,13,30,34). These mills are specifically designed to produce a very fine product with high efficiency. In the fluid energy mills, the solids are entrained in high velocity gaseous or vaporous streams. Reducing particle size occurs by impact and attrition against other particles or a target, eg, a Micronizer. Vibratory mills consist of a nonrigidly supported chamber filled with grinding media (up to 80% of volume) and material to be ground, vibrated at frequencies up to 1800 min^{-1} by an eccentric mechanism. Attrition mills are of the rotating-disk (colloid mills), patterned-fluid (fluid energy mills), or abrading-sand (sand grinders) type. Size reduction is achieved by particles breaking each other after the particles have acquired the necessary energy from a solid or fluid impeller. In Stirred mills, such as tower mills, grinding occurs by attrition and abrasion in a vertical chamber containing steel balls or pebbles (35). An internal screw flight provides medium agitation. These are low-volume high intensity attrition mills. Although they were developed for fine grained ores that needed an economic grind to <10μ sizes to achieve sufficient liberation, they find applications even in coarser grinding, particularly when power efficiency, space and partile surface chemistry are important. Advantages of stirred mills are: narrower size distribution than conventional grinding, large unit size with high power input (up to 2.6MW), high grinding efficiency, short residence times, low operating costs, and the ability to use inexpensive, inert and finer size grinding media such as sand. A feed of 6 mm top size can be ground to a product in the size range of 74 to 2 μm or finer at 100 t/h.

1.6. Size Separation. Sizing of the crushed and ground product is a necessary step prior to any mineral processing operation, and in the production of a product having a specific size. Controlling the size of material fed to other equipment is important. All equipment has an optimum size range of material that it can handle most efficiently. Size separation can be achieved either by screening (for coarser particles) or by classification (for fines) (see also SEPARATION, SIZE).

Screening or Sieving. Screening or sieving (4,7,13,36) is accomplished by passing the crushed or ground ore through a mesh of perforated plates or woven metal wires/rods or profile bars providing a uniform distribution of fixed-size apertures. Screening is a continuous process, whereas sieving is a batch process used to determine the performance of crushing and grinding processes. Material retained on the screen is the oversize or the plus fraction, and the material passing is the undersize or the minus fraction. The product specifications are usually in terms of percentage of material passing a certain screen or sieve size. Although the principle and the process of screening and sieving are simple, separation is seldom perfect for any sizing operation. The most obvious contributing factor is the irregular three-dimensional shape of the particles. Each particle at any given time presents only two dimensions to the screen surface.

Laboratory sieves are made from woven wire of standard square apertures that follow a geometric progression from one screen to the next. The sieve size is designated in micrometers or mesh number. The latter denotes the number of openings per linear inch. In the laboratory determination of size distribution of a crushed or ground product, standard sieves in the desired size range are stacked. Successively smaller apertures are arranged from top to bottom, the crushed or ground material is placed on the coarsest sieve at the top, and the entire stack is shaken on vibratory equipment providing both vertical and circular motion to the particles. Material retained on each screen is then weighed, and the weight percent passing each screen is plotted against the sieve size. This is typically a log-log plot, but many other techniques have been developed (4). Wet screening is usually conducted before final dry sieving to remove very fine particles, which tend to adhere to coarse particles or to each other during dry sieving and lead to errors.

Industrial screening is used essentially for separations over 0.2 mm and in conjunction with crushers because the efficiency decreases rapidly as particle size decreases. The main objective is to remove undersize material that should not be circulated back to the crushers, or to remove (scalp) oversize material or trash that should not report to the subsequent processing step. Other applications of screening include production of a specification size material (as in quarries), dewatering, and trash removal from processed material.

Industrial screens can be classified as stationary or moving, which in turn can be either conventional (fixed aperture) or probability (variable aperture, statistical event). The applicable size ranges for screens are shown in Fig. 10 (7). The operating mechanism and the screen surfaces of probability screens are basically the same as that of conventional screens, only the design is different. Principal screen types include stationary and vibrating grizzly (Fig. 11a) (4), Hukki screen, roll grizzly, revolving screens, vibrating screens (Fig. 11b) (4), shaking screens, reciprocating screens, sieve bends (Fig. 11d) (7), and rotary sifters (4,7,13,36). Vibrating screens, the most widely used, have numerous designs, ie, inclined, horizontal, etc, and vibration can be generated either mechanically or electromagnetically. They can handle material from 250 μm to 250 mm. The probability type has a series of relatively small inclined screen surfaces. Separation is based on probability of a particle passing through an aperture that is larger than the diameter of the particle, rather than a fixed mesh size opening or physical constraint. This type of screen has higher capacity and efficiency than the other vibrating screens. The selection of a particular screen design depends on the application and the size cut-off. Grizzlies are used generally for very coarse material. Sieve bends have extended the applicability of industrial screening into fine sizes down to 50 μm. The upper size can be as large as 12,000 μm and capacities as high as 2000–5000 t/d. These bends consist of a curved or inclined screen surface having horizontal wedge bars. The size of the unit varies in height from 1.5 to 2.5 m and width from 0.5 to 2.5 m. They have relatively high capacity and efficiency. Among the revolving screens, the trommel is a slightly inclined, rotating cylindrical screen and it can be used wet or dry. High wear is a big disadvantage. Other types of revolving screens are the centrifugal and probability screens (Fig. 11c) (4). The former is a vertically mounted cylindrical screen; high wear is again a disadvantage. The probability screen consists of rotating radiating bars; the speed of rotation determines cut size. This high capacity screen can also produce fine separations. The rotary sifters can be either reciprocating or gyrating. In the former, a rectangular screen surface with a slight incline is used with a reciprocating motion at the discharge end, whereas in the gyrating type, a circular screen is used with a circular motion.

There are three basic types of screen surfaces: perforated or punched plate, woven cloth, and profile bars. Woven cloth surfaces are the most common. Perforated plates are made up of hardened steels, stainless steels, Monel, rubber, or plastic. Woven cloth can be made from high carbon steels, tempered steels, manganese steels, galvanized steel, Monel, copper, bronze, or reinforced synthetic cloths of polyurethane rubber. Screens can handle material ranging from the fineness of talc through boulders as large as 2 × 2 m weighing as much as 10–12 tons. Screen openings range from 0.1 mm to as large as 500 mm.

Capacity and efficiency are the two criteria used to assess screen performance. Whereas many empirical formulas for measuring these criteria are available, none is entirely satisfactory. The efficiency is usually expressed in the form of a partition or performance curve which is a plot of percent oversize vs. the geometric mean size on a log scale (4). For a perfect separation, such a curve would be a vertical line at the cut-off size, but in practice it is an S-shaped curve (see Fig. 12) (4) On such a plot, the separation or cut-off size is that corresponding to 50% oversize. A number of factors affect screen efficiency. These include feed rate, vibration rate or intensity, particle shape, screen design, the amount of moisture in the feed, and the amount of near-mesh particles which tend to blind or plug the apertures. Wet screening is more efficient than dry screening, but the decision to wet or dry screen depends largely on the application.

Classification. Classification covers a broad range of size separation methods that rely on the differences in settling or sedimentation velocities of particles in a fluid (air or water). Because these velocities are affected by size, shape, and density of the particles, classification is in reality a sorting rather than sizing method (4,7,13,3). Moreover, there is incomplete liberation of minerals in the ground product and the pulp invariably contains composite (locked) particles of varying specific gravities. Classification is more suitable for particles in the fine size range where the performance of screens is poor. The products are an oversize (or underflow, heavies, sands) and an undersize (or overflow, lights, slimes). An intermediate size can also be produced by varying the effective separating force. Separation size may be defined either as a specific size in the overflow screen analysis, eg, 5% retained on 65 mesh (212 μm) screen or 45% passing 200 mesh screen (75μm), or as a d_{50} (or p_{50}), defined as a cut-off or separation size at which 50% of the particles report to the oversize or undersize. The efficiency of a classifier is represented by a performance or partition curve similar to the one for screens (see Fig. 12) (4), which relates the particle size to the percentage of each size in the feed that reports to the underflow.

In a simplistic view, a classifier may consist of a column with rising fluid and falling particles. Particles either sink (sedimentation or settling) or are carried upward with the fluid depending on the terminal velocity of the particles as dictated by gravitational and frictional forces. Particles experience either free or hindered settling depending on the solid/fluid ratio. Density differences have a more pronounced effect on classification at coarser size ranges and size differences dominate at fine size ranges. Free-settling conditions accentuate the effect of size, and the hindered settling conditions accentuate the effect of density. Depending on the conditions, classifiers can, therefore, separate relatively coarse from relatively fine particles, or separate smaller heavier particles from larger lighter ones. They can also split a wide size distribution into manageable fractions. Classifiers can be grouped into horizontal current and vertical current types, or into mechanical, nonmechanical, sedimentation, and hydraulic or fluidized-bed types depending on the design of the equipment. The available equipment, their sizes, capacities, and their uses are given in Table 5 (7).

Horizontal current classifiers such as the mechanical classifiers are essentially of the free-settling type, whereas the vertical current classifiers are of the hindered-settling type. Sands discharge is by mechanical means against gravity in mechanical-type classifiers. Gravity or centrifugal forces are used for sands removal in the nonmechanical classifiers. In hydraulic or fluidized-bed classifiers, particles in the pulp settle under hindered settling conditions against a counter current flow of additional water in a series of columns. Products can be produced from each column: coarser and denser particles from the first column, finer and lighter particles from subsequent columns, and the slimes reporting to the final overflow. Hydraulic classifiers are used extensively in conjunction with gravity separators. In general, the fluidized-bed classifier is capable of giving a sharper separation and a more precise d_{50} than a sedimentation classifier, but this is achieved at the expense of capacity. Some of the fluidized-bed classifiers are essentially sedimentation classifiers using only small amounts of hydraulic water. In sedimentation classifiers, particles settle through a pool of water formed from the feed stream in an inclined trough (Fig. 13) (37). The settled mass is dragged upward by mechanical means, typically through a spiral, rake, or drag, and the fines go with the overflow. Widely used in closed-circuit grinding, these classifiers are also used to produce a clean, ie, free of slimes and fines, sized final product. The capacity and separation efficiency are influenced by feed rate, the speed of the rake or the spiral, the height of the overflow weir, and dilution of the pulp. One of the principal disadvantages is that the overflow tends to be very dilute for many subsequent operations. Mechanical sedimentation types of classifiers, such as the spiral classifiers, are large units and occupy substantial floor space, in contrast to the nonmechanical units.

The hydrocyclone, commonly referred to simply as cyclone, is a nonmechanical sedimentation-type classifier (Fig. 14) (4,7,13,37). It has no moving parts or power attachments directly connected to it. The hydrocyclone has become the workhorse of most mineral processing operations because of its simplicity, short residence time, compactness, and low cost of operation. It is, however, characterized by lack of sharpness of separation. Equipment consists of a cone having an open apex attached to a cylindrical section which has a tangential feed inlet at the top, through which the feed is introduced at a high pressure (Fig. 14a). The pressure feed generates centrifugal action to give high separation forces and discharge. Overflow is through a pipe mounted axially on the plate that covers the cylindrical section. A short extension of this pipe into the cylindrical section acts as a vortex finder which prevents short circuiting of the feed directly into the overflow. The flow pattern in a hydrocyclone can be described as a spiral within a spiral generated by the tangential feed. An air core exists in the center along the vertical axis extending through the apex. The centrifugal force developed accelerates the settling rate of particles, and particles are separated according to size and specific gravity. Faster settling particles exit through the apex and slower settling particles exit through the overflow (see also SEPARATION, CENTURIFUGAL).

The hydrocyclone is used in closed-circuit grinding and is efficient for separation at fine sizes. It can be used not only as a classifier, but also as a thickener, a desliming unit, or a concentrator (7). The disadvantages are the lack of sharpness of separation and the amount of material that passes through without classification. When the required cut-off size is coarse, the plant throughput may not be large enough to satisfy the cyclone feed requirements. It is also generally impractical to operate when cut points are $<2\ \mu m$. In spite of simple design and operation, the actual processes occurring in a hydrocyclone are far from simple. A complete rigorous analysis is unavailable, but numerous empirical formulations have been developed to describe its operation (4,7). The separation is not perfect and the performance curve is typically S-shaped (Fig. 14b). The slope of the central portion of this performance curve represents the efficiency or the sharpness of separation. The closer to vertical the slope, the greater the efficiency.

Empirical improvements have been made to the performance curve (4,7); for example, corrected performance curves, ie, corrected for part of the inlet stream passing out of the classifier without being classified, especially fines reporting in the underflow, and reduced performance curves, wherein the corrected performance curve is normalized by dividing the size scale by d_{50}. The latter is largely independent of the nature of particles, and is a characteristic of the type of classifier within a reasonable range of design. In a practical mineral slurry, which is invariably nonhomogeneous and consists of a diverse population of mineral particles, either as liberated or as middling particles, typically a number of different sized particles will have the same settling rate. Therefore, in a given classifier, each mineral has its own performance curve resulting from differing densities or shape or both. This has important implications, either beneficial or adverse depending on the type of ore being treated, in closed-circuit grinding and the subsequent concentration separation.

Although performance curves are valuable in assessing classifier performance, frequently the cyclone overflow size analysis is used more than the d_{50} of the cyclone. In practice, clusters of cyclones (in parallel) are used to handle large capacities. Cyclones are manufactured in sizes ranging from 0.01 to 1.2 m in cyclone diameter, ie, the cylindrical section at the top (4,13). Capacities run from 75 to 23,000 L/min. Materials of construction vary widely. Rubber-lined or all-polyurethane cyclones are used when abrasion is a problem.

1.7. Minerals Concentration. Although the size separation/classification methods are adequate in some cases to produce a final saleable mineral product, in a vast majority of cases these produce little separation of valuable minerals from gangue. Minerals can be separated from one another based on both physical and chemical properties (Fig. 15) (7). Physical properties utilized in concentration include specific gravity, magnetic susceptibility, electrical conductivity, color, surface reflectance, and radioactivity level. Among the chemical properties, those of particle surfaces have been exploited in physico-chemical concentration methods such as flotation, flocculation and oil agglomeration. The main objective of concentration is to separate the valuable minerals into a small, concentrated mass which can be treated further to produce final mineral products. Except for molybdenite (MoS_2), which is recovered from ores at high purity and used as-is in many applications, sulfide minerals in ores are concentrated for the sole purpose of producing the corresponding *metal*; for example, sulfide minerals of Cu, Ni, Zn, Pb. Pyrite and pyrrhotite, which are sulfide minerals of iron, are ubiquitous, but they are not concentrated to produce Fe metal. In base metal sulfide ores, pyrite and pyrrhotite are considered as nuisance minerals unless they carry valuable metals such as Au, Ag and platinum-group elements (PGE). Although many non-sulfide ores are processed for the purpose of producing a metal - for example, oxides of iron (hematite, magnetite), aluminum (bauxite) and titanium (rutile)–a vast number of non-sulfide ores are processed to produce saleable mineral products–for example, kaolin clays, glass sands, mica, talc, alumina, and foundry sands. In general, tonnages treated for sulfide ores are far greater than those for non-sulfides.

Selection of a particular concentration method depends entirely on the mineral and metal in question, the nature and mineralogy of the ore deposit, particle size at economic liberation, and the prevailing socio-economic factors. For many centuries, sorting by hand and gravity concentration were the only methods available. By the end of the nineteenth century magnetic and electrostatic methods had been introduced. Minerals processing underwent a revolution when the flotation method was developed in the early twentieth century because large tonnages of a wide variety of ores at various grades could be processed and complex mineral separations that were not possible by any other method could be performed.

Ore Sorting. Sorting methods rely on differences in the physical properties of the various mineral components in an ore (4,7,13,38,39). Such properties include optical characteristics, magnetic susceptibilities, x-ray fluorescence, electrical and thermal conductivity and charging, and radioactivity. Sorting techniques became successful once the sorter was able to analyze individual particles. Sorters can function efficiently only at coarse (typically >ca 10 mm) sizes, and in a narrow size range. Particles up to 160 mm can be sorted at capacities of 180 t/h . The extent of liberation of the valuable mineral from the gangue at such coarse sizes determines whether a sorter can be used. The nature of distribution of the value mineral in the ore is also important. For some sorters, the desired mineral must be exposed on the surface of the particle and not buried inside the particle. In many cases, limitations are imposed by the type and capacity of the mining equipment. The particle surfaces presented to the sorters must be clean of dust and slime-free not only to ensure that the surface physical properties of the minerals are not obscured, but also to protect the optics of the detection system. Such requirements are less critical for sorters that rely on the bulk properties of the mineral such as electrical and thermal conductivity, magnetic properties, and radioactivity. In these cases, a random distribution of the valuable mineral in the particle is preferred. Attempts to quantify ore sortability are limited.

The primary features of a typical sorting machine are shown in Fig. 16 (7). The detection system in a sorter is the most important unit and sorters are, therefore, grouped according to the detection system. Advances in the electronic industry have greatly facilitated the development of sophisticated sorting machines. The modern sorter has three distinct functions: singulation, detection, and ejection. Singulation ensures that individual particles are presented to the detection system. The detection system evaluates the selected physical property for the particle presented and sends a signal to a combination of electrical and mechanical ejection systems to classify the particle as valuable or gangue. The singulation and ejection systems determine the capacity of the sorter and the detection system determines the efficiency of the separation. A perfect synchronization between the detector and the ejector is necessary in the decision making process which incorporates the time delay between the detector and the ejector, and the size and position of the particle in the stream. After the ejection of one particle is complete, the ejector must return to its original position as rapidly as possible. Most sorters use solenoid controlled air valves to eject selected particles. Particles are presented to the detector either single file or single layer using a vibrating hopper and a conveyor belt. The single layer mode provides for increased capacity. Scanning speeds can be as high as $2000\,\mathrm{s}^{-1}$ for the ore stream, equivalent to one scan for every 2 mm of rock length (4). The most important advances in sorting, as in most other mineral concentration methods, are in automation and instrumentation.

Optical and photometric sorters (13) are the most widely used. Optical properties in the detection system include reflectivity, color, transparency, and fluorescence. Photometric sorters have all of the features shown in Fig. 16 and utilize a laser light source in a scanning mode to detect reflected light using a sensitive photomultiplier (see LASERS; PHOTODETECTORS). Applications include upgrading rock salt, magnesite, barite, gypsum, marble, diamond, aggregates, asbestos, talc, and limestone. Color forms the basis for sorting white mineral-bearing quartz from quartzite of various colors. Other applications include separation of gold, cassiterite, and wolframite in quartz from quartzite, and white quartz containing tetrahedrite and other minerals, ie, Ag and Cu values, from quartzite. Scheelite sorting is achieved by sensing fluorescence under ultraviolet light. For minerals that do not exhibit fluorescence naturally, selective chemicals that adsorb on the value minerals and provide fluorescence have been considered in the laboratory. Asbestos (qv) is sorted using sequential heating and infrared scanning.

Radiometric ore sorting has been used successfully for some uranium ores because uranium minerals emit gamma rays which may be detected by a scintillation counter (4). In this application, the distribution of uranium is such that a large fraction of the ore containing less than some specified cut-off grade can be discarded with little loss of uranium values. Radioactivity can also be induced in certain minerals, eg, boron and beryllium ores, by bombarding with neutrons or gamma rays.

Other sorters such as X-ray, conductivity, and magnetic sorters have found only limited application (13). Sorting based on mineral conductivity has been used for iron ores and native copper. Diamonds emit visible light when bombarded with X-rays. This property has been used for the sorting of diamond ores in conjunction with the traditional grease tables. X-ray fluorescence has also been used in sorting. One commercial sorter measures electrical conductivity and magnetic susceptibility of individual particles and the detectors respond to only slight variations in such properties. It also uses an optical system to track the size and shape of the particle. A microprocessor estimates the grade of the particle from its size and measured bulk properties for the decision making process. Such a sorter would be suitable for a preconcentration step and would be applicable for a wide variety of ores because of bulk property measurement. Gamma radiation scattering analyses form the basis of another sorter (4). The most suitable metals are chromium, iron, cobalt, nickel, copper, and zinc or any combination of these.

Gravity Methods. Gravity methods date back to antiquity, and by sixteenth century these were well advanced (4,7,13,40–42). A considerable amount of gravity concentration occurs in nature in the form of concentration of heavy minerals in placer or beach sand deposits as a result of moving water. Gravity methods waned in popularity with the advent of flotation (qv) in the early part of the twentieth century. A revival, either as a primary technique or a preconcentration step, has occurred because of the simplicity, low cost, and advances made in the design of equipment . Gravity methods have been extended into the 50–10 μm range although capacity and throughput can still be a problem. These methods are also being evaluated for the treatment of large dumps of flotation tailings that contain heavy minerals. In 1975, approximately 490×10^6 t of coal and ore were treated by gravity methods compared to the 423×10^6 t treated by flotation .

Gravity methods, ie, gravity concentration and dense medium separation, rely on the differences in specific gravities between minerals in fluids. In gravity concentration, the fluid is water or air; in dense medium separation, the medium of separation is a liquid or fluid of specific gravity greater than that of water. Gravity methods are used to treat a variety of ores, including industrial minerals, coal, iron ore, phosphate ore, precious metals, tungsten ores, diamonds, and heavy metal sulfides.

Gravity Concentration. Gravity concentration devices or gravity separators have much in common with size separators and can be used interchangeably under certain conditions. The principles are similar except that in gravity separators care is taken to emphasize specific gravity differences of the materials being processed. The performance of gravity separators is best when a narrow range of feed size is used, just as the performance of size classifiers is best for a narrow range of densities. It is, therefore, important that a properly size-classified and consistent pulp sample be fed to the gravity circuit. Care is taken to eliminate or minimize slimes in the feed as these tend to decrease the sharpness of separation. The efficiency of gravity separators increases as particle size increases, but other factors, such as degree of liberation of values, is also important in determining the optimum size range of the feed to the gravity circuit. A gravity concentration criterion (Dh-Df)/(Dl-Df), where Dh, Dl, and Df are specific gravities of the heavy mineral, the light mineral, and fluid medium, respectively, is often used to assess the type of separation possible (4). If this ratio is 2.5 or greater, gravity concentration is very efficient, and efficiency decreases as the ratio decreases. Gravity concentration in some cases be used to produce a final saleable product; for eg, coarse coal. More typically, it is used as a preconcentration method; the final gravity concentrate requires cleaning by other methods, such as flotation, magnetic and electrostatic separation, and leaching, to remove impurity minerals. A typical gravity circuit may include roughing, and several stages of cleaning of the rougher concentrate with regrinding in between, although regrinding of middlings in the rougher stage may also be required.

The performance of gravity concentrators can be measured empirically by performance or partition curves similar to those for classifiers (Fig. 12) (4) except that a cut-off gravity is used instead of a cut-off size (7). Much of the available information, however, has been obtained only for coal cleaning (see COAL CONVERRION PROCESSES), because of the difficulty in carrying out measurements on high density minerals. The reduced performance curve is primarily a characteristic of the particle size being treated and of the device. Available empirical correlations are rather limited, however, unlike the case of size classifiers.

Gravity separators can be grouped according to either the operational feed size range or the manner in which the particles and fluid move relative to one another in the separator. Gravity separators fall into three groups: jigs, shaking concentrators, and gravity flow concentrators (7,13). Many of these are rather simple devices. The size, capacity, and uses of selected gravity separators are given in Table 6 (7).

Jigs, generally very effective for relatively coarse (typically 0.5–200 mm) material, have relatively high unit capacities. The sharpness of separation is a function of size distribution. Fine specific gravity separation is possible for a closely sized material. The principal usage of jigs is in coal beneficiation. Other applications include concentration of cassiterite, tungsten, gold, barites, and iron ores. The basic jig (Fig. 17) (4) has a large tank or hutch divided in the upper portion into two main sections. One section contains the stationary screen with the mineral bed on it, the bed depth being many times the thickness of the largest particle, and the other section contains the pulsating device, most commonly using air pressure rather than mechanical means. Mineral separation is achieved by applying a vertical oscillatory (pulsating) motion to the solids-fluid bed. This pulsating motion produces dilation of the bed and subsequent stratification. The denser and larger particles form a lower layer whereas the finer lighter particles are on top. The processes occurring in a full cycle of operation may be considered differential initial acceleration, hindered settling, and consolidation trickling. Several other theories have been developed, however, notably the center-of-gravity theory (7,13). The pulsing action is supplemented by using additional water in the hutch during the settling period. This extends the open state of the bed for a longer time. The dense minerals are collected either on the screen or under the screen depending on the screen aperture size. In the latter case, a layer of dense (ragging) particles larger than the aperture size are placed on the screen to regulate the collection of dense fraction. Examples are feldspar in coal cleaning, and hematite in cassiterite and scheelite separation. Several stages of jigging are used to achieve efficient separation. The commercial jigs have a variety of designs for the pulsating device and the removal of products. Some examples are shown in Fig. 18 (7). The Batac jig, which uses multiple air chambers under the screen, is the industry standard in coal cleaning. A more recent development is the Kelsey centrifugal jig (43,44). The circular or radial jig is a variation of the conventional rectangular design of hutches in series. The pulp is fed at the center and flows radially over the jig bed and exits at the circumference. A raking mechanism ensures an even bed depth throughout. It is mechanically simple and has very high capacity, up to 300 m^3/h for a maximum particle size of 25 mm. It achieves a fast compression/slow suction stroke with virtually no (additional) hutch water. The slow suction stroke allows more time for the fines to settle to the bed.

In the second group of gravity concentrators, eg, the shaking table, the vanner, the Bartles-Mozley concentrator, and the miners pan, a horizontal shear is applied to the solids-fluid stream by vibrating the surface under the stream. Despite early popularity, conventional tables have limited usage in the minerals industry mainly because of low capacities and the large floor area they occupy, and because of the availability of many alternative concentrating methods. Tables are employed for processing coal and, to a lesser extent, many industrial minerals. In general, tables are efficient separators used for difficult flow streams and for producing finished concentrates from products of other separators. The shaking table is effective in separating small dense particles from coarse light particles, and is capable of treating finer particles than jigs, but at the expense of capacity.

In a typical shaking table (Fig. 18c) the feed enters through a distribution box along part of the upper edge and spreads out over the table as a result of the differential shaking action and the wash water. Product discharge occurs along the opposite edge and end. The surface of the table is a suitably smooth material, eg, rubber or fiber glass, and has an appropriate arrangement of riffles on it, which decrease in height along their length toward the discharge end. Various riffle arrangements can be used to emphasize grade or recovery. The table also has an adjustable slope or tilt of about 0–6 degrees from the feed edge down to the discharge edge to regulate distribution of material. During tabling, minerals are subjected to lateral (table motion) and longitudinal (flowing film of water) forces. The net effect is a diagonal movement from the feed end. Separation of minerals is effected as a result of flowing film concentration, hindered settling, consolidation trickling, and asymmetrical acceleration. Small dense particles move more slowly than coarse light ones and depending on the size and density, the particles fan out on the tables. Several products can, therefore, be collected.

Double- and triple-deck shaking Tables 4,7 have much higher area/capacity ratio at the expense of some flexibility and control. The Bartles-Mozley concentrator is an example of a multiple-deck device developed to recover fine cassiterite that cannot be recovered by other devices. It consists of a suspended assembly of 40 fiber glass decks arranged in two sandwiches of 20, each deck separated by a 13-mm space that also defines the pulp channel. Feed is distributed to all 40 decks for a period of up to 35 minutes, after which the flow is interrupted briefly while the table is tilted for concentrate removal and the feed resumes. The concentrator has high capacity and is capable of recovering a majority of fine (ca 10 μm particles) cassiterite in low grade slurries. Other devices include pneumatic tables and the duplex concentrator (4).

In gravity flow concentrators, eg, sluices, troughs, spirals, and the Reichert cone, a layer of slurry flows under gravity down an inclined surface (4,13,41). The separation is one of both size and density and has been used for centuries, including the natural processes in placer deposits such as those of tin, gold, and beach sand minerals. The simplest device is the pinched sluice, an inclined, tapered launder on which pulp flows gently and stratifies as it descends. Although relatively simple and inexpensive to operate, usage is limited because of the development of more efficient, higher capacity concentrators. The Reichert cone (Fig. 19) (37) is a high capacity device developed initially to treat titanium-bearing beach sands. The efficiency is rather low, but this cone is effective as a roughing device for large tonnages. The principle of operation is similar to that of a pinched sluice. It comprises several cone sections stacked on each other which permit separation in stages as the feed descends from the periphery of the cone to the center. It is essential to maintain a high feed density (55–70% solids). These concentrators are most efficient in the 100–600 μm size range (4).

Another gravity flow concentrator is the spiral (Fig. 19b), initially known as the Humphrey's spiral (introduced in the 1950s), which has also been used largely for mineral sands containing rutile, ilmenite, and zircon, and for iron ore. As the slurry flows down a spiral surface, particles are stratified owing to the combined effect of centrifugal force, differential settling rates, and trickling through the flowing particle bed. The primary process is hindered settling. The addition of secondary wash water at the center of the spiral aids the already present river bend action resulting from the circular path of the slurry. This results in the larger lighter particles reporting to the periphery of the spiral. The denser material goes near the center, where it is removed through the discharge ports. The spiral requires lower (15–45%) pulp densities than the Reichert cone and can handle a size range of 3 mm to 75 μm (4). Modern spirals are made of fiber glass and may comprise two separate spirals intertwined to save floor space. High capacity spirals are a more recent development and operate at high pulp densities using no wash water. These have a flatter profile and have no splitters. Other advantages are smaller plant footprint and simpler plant layout, lower capital cost, and easier control. These newer designs have increased spiral usage.

More recent advances in the centrifugal gravity separators are the Mozley multigravity separators (MGS), the Knelson concentrator, the Falcon concentrator, and the Knudsen bowl centrifugal concentrator (43–45). These separators are particularly suitable for fine particles, and applications include precious metals, coal, many industrial minerals, coal desulfurization, soil remediation, and harbor silt clean-up. They are all characterized by a high capacity/area ratio. The MGS is essentially a conventional shaking table that has been wrapped into a drum. By rotating the drum, forces many times the gravity can be exerted on the particles in the film flowing across the surface. The large-scale unit comprises two drums mounted back to back, rotating at speeds 90–150 rpm with shaking oscillations of 4–6 cps, and enabling forces of 5 to 15 G. The Knelson and the Falcon concentrators have both gained popularity for precious metal ores (45). The Knelson concentrator is a compact unit with an active fluidized bed to capture heavy minerals. It consists of a tapered bowl in which a ribbed inner cone is rotated at high speeds. Capacities up to 40 t/h (for 760-mm dia unit) and forces up to 60 G are possible. A more recent model claims a nominal capacity of 650 t/h with an installed power of 200 to 500 HP, rotor speed in the range of 225 to 450 rpm generating 50 to 200 G. The Falcon concentrator, which can generate 50–300 G centrifugal field, can have capacities of 1.0 to 100 t/h, with a capital cost below $4000 per t/h capcity, and total operating cost of >$0.4 per ton.

Dense Medium Separation. Dense medium separation (DMS) (4,7,13) is a gravity concentration method in which a medium of density higher than that of water, but between the densities of the minerals to be separated, is used. It is also referred to as heavy media separation or the sink-float process. DMS is used in the concentration of a variety of minerals, essentially as a preconcentration step. The principal use is in the cleaning of coal to produce a final product. The conventional dense medium separators, not highly efficient for the separation of fine particles, are restricted in use to ores in which either the valuable or gangue minerals liberate at a relatively large size. The advantages of this method include sharp separations, high efficiency even in the presence of large amounts of near-density material, ease of changing the separation density, and ease of control. This method, however, is relatively expensive.

In the simplest form of the method, heavy organic or inorganic liquids, the latter usually dissolved salts in water, of appropriate (1.5–5) specific gravities are used (4,13). Higher density minerals are collected in a sink product; lower density minerals are collected in a float product. A third middlings product is also sometimes collected. Examples of heavy liquids are tri- and tetrachloroethane, tri- and tetrabromoethane, di- and triiodomethane, and aqueous solutions of sodium polytungstate, and thallium formate-malonate. The specific gravities of the medium can be varied by mixing in liquids of lower specific gravities such as carbon tetrachloride or triethyl orthophosphate. In view of the toxicity and high cost of most of these liquids, use is restricted to laboratory testing of ore to assess suitability of ores for gravity concentration and to determine the economic separation density and minerals liberation.

The medium used in industrial separations is a suspension in water of finely divided high density particles, most commonly fine magnetite (sp gr 5.1), ferrosilicon (sp gr 6.7–6.9), or a mixture of the two, although quartz (sp gr 2.65, Chance process for coal) and other solids have been used (13). Medium specific gravities in the range of 1.5–3.4 can be obtained readily. Both magnetite and ferrosilicon are physically hard and chemically stable, form low viscosity fluids, and can be recovered readily using magnetic separation. The size of the medium particles is generally in the range of 95% −150 to −40 μm. Magnetite is used mainly for coal, and ferrosilicon for metalliferous ores.

Industrial separations are conducted in gravity or bath separators for a coarse feed, and in centrifugal separators for a fine feed (4,7,13). In gravity-type separators the feed and medium are introduced to the surface of a large quiescent pool of the medium. The float material overflows or is scraped from the pool surface. The heavy particles sink to the bottom of the separator and are removed using a pump or compressed air. The drum separator (Fig. 20) (4), up to 4.6 m dia and 7 m long, processes approximately 800 t/h, and treats feed of size up to 30 cm dia, operates in the gravity or the static bath mode. Feed enters at one end of the drum and the floats exit from the other end. The sink product is removed continuously from the rotating drum through the use of lifters attached to the drum which empty into a launder as they move to the top. A modification of the simple drum separator is the two-compartment drum separator which allows a two-stage separation. In the cone-type separator (up to 6.1 m in dia and 450 t/h) feed is introduced at the top. The medium in the cone is kept in suspension by gentle agitation. The sink product is removed from the bottom of the cone either directly or by airlift in the center of the cone. The maximum particle size that can be separated is limited to 10 cm. Other separators include the Drewboy bath and the Norwalt bath (4).

The settling rate for smaller particles would be too low for efficient separation in gravity separators. This is overcome by the use of centrifugal forces. Acceleration up to 20 times that of gravity can be obtained. Hydrocyclones, similar to those used for fine feed classification, are used as the centrifugal separators after certain design modification. Separations can be made with feed sizes in the range of 0.5–30 mm at capacities approximately 75 t/h. Hydrocyclones are Used for the separation of a wide variety of feeds, most notably coal, where many advantages are clearly realized because the separation is extended into finer size ranges. Pyrite removal from coal and upgrading of oxidized coal are also achieved more effectively than using flotation. Separation can be extended down to 0.1 mm. The Dutch State Mines cyclone is able to treat ores and coal in the size range 40–0.5 mm (4). The ore suspension in fine ferrosilicon or magnetite is fed tangentially to the cyclone under pressure. The sink product is removed through the central vortex finder. In the Vorsyl separator the feed makes an involute entry under pressure (13). The lighter, clean coal exits through the vortex finder, and the heavier particles move in a spiral path against the wall and are removed at the base of the vessel. The large coal dense medium separator (LARCODEMS) (Fig. 21) comprises a cylindrical vessel (1.2-m dia and 3-m long) inclined at 30° to the horizontal (4). Feed medium makes an involute entry. It can handle coal in the size range 0.5–100 mm, and the capacity can be as high as 250 t/h. The Dyna whirlpool (13), the precursor to the LARCODEMS, is used to treat fine coal, diamonds, fluorspar, manganese, tin, and lead-zinc ores. It has many advantages over the DSM cyclone and has a high sinks capacity. The Tri-Flo separator (4) is similar to the Dyna whirlpool but has two separators in series. It can be operated with two media of differing densities in order to produce sink products of individual controllable densities, or with a single medium in which case a two-stage separation is possible resulting in increased recovery. There are also autogenous dense medium separators wherein fine particles of high and intermediate density from the ore to be treated form the autogenous dense medium in the cone. These can separate coal particles in the size range of 600–150 μm. They have lower capital and operating costs because of the elimination of medium separation and recovery costs.

Desliming of the feed to the dense medium separators is important because slimes interfere with the separation. Low amounts of clay contaminations have been known to stabilize the suspension, however. The particle size and rheology of the medium are two important factors. Significant advances have been made in automatic control of the medium consistency. Density of the medium can be controlled to within ±0.005 specific gravity units (7). Because DMS does not depend on surface properties of particles, this could often be a significant advantage over flotation and other methods. Also the energy requirement for grinding, flotation, and dewatering is sufficiently high to make dense medium separator an attractive concentration method, at least as a preconcentration step.

The efficient recovery and recycle of the medium is a factor in the economics of the DMS process. Typical losses of medium per ton of feed treated are about 0.5 kg for magnetite and 0.25 kg for ferrosilicon (13). Losses occur because of inadequate washing of the separated products and the need for periodic replacement owing to buildup of fine mineral particles resulting in lower medium density. The recovery involves screening of feed to remove fine particles, screening of both sink and float products in two stages, and concentration of diluted medium by wet, low intensity magnetic separation of the drum type using permanent magnets. The feed to the drum separator contains 10–15% solids which are 75–90% magnetics. A 760-mm diameter concurrent single-drum separator can give magnetic recovery of 99% or better (13). Double-drum units are used when feed volumes are large.

Dry dense medium (pneumatic fluidized-bed) separation has been used, but has not received wide attention by the industry. An area of promise for future developtnent is the use of magnetically stabilized dense medium beds by using ferro or magnetic fluids (4,13). Laboratory and pilot-scale units such as Magstream are available. In this unit, material is fed into a rotating column of water-based magnetic fluid. Particles experience centrifugal forces and opposing buoyant forces which are magnetically derived. These forces are balanced so that lighter fraction is biased toward the center and the heavies are biased outwardly, and these fractions are collected separately. The unit can treat particles in the size range 1 mm to 53 μm and the specific gravity split point can be in the range of 1.3–21.

Magnetic Separation. Magnetic separation, based on the differences in magnetic susceptibilities between minerals, has been used since the early 1800s (4,7,13,34,40,46,47). Usage has increased steadily in the latter twentieth century because of significant advances made in applications technology and design of equipment. One application is in the concentration of iron ores from nonmagnetic, nonvalue minerals. Another is the removal of iron and iron-bearing nonvalue minerals from valuable nonmagnetics and for many nonferrous minerals (see MAGNETIC MATERIALS).

Magnetic susceptibility, a bulk property, is the ratio of the intensity of magnetization (M) produced in the mineral to the magnetic field (H) which produces the magnetization. In addition to field intensity, field gradient, ie, the rate at which field intensity increases toward the magnet surface, is also important (4). Minerals may be divided into ferromagnetic, paramagnetic, and diamagnetic depending on how strongly they interact with an applied magnetic field. Iron and magnetite are ferromagnetic; hematite, ilmenite, pyrrhotite, wolframite, and chromite are paramagnetic; quartz and feldspar are diamagnetic. Magnetic susceptibility is ca -0.001 for quartz and ca 0.01 for hematite (4). It is much higher for ferromagnetic materials and is a function of the magnetic field.

Magnetic separation is carried out either wet or dry in low or high intensity separators depending on the type of separation. A steep field strength gradient is maintained in both types of separators by using different pole designs. The field intensity is regulated by changing either the space between poles or the current (in electromagnets). Belts or drums are used to transport the feed through the field. Some examples of the equipment used are shown in Table 7 (7) and Fig. 22 (7) and 23 (4,7). In Table 5, at bottom, change 10^{-4} to 10^{4}, and change Ref. 6 to Ref. 7

High magnetic susceptibility minerals, ie, ferromagnetics and paramagnetics of high susceptibility, are primarily separated in wet low intensity separators (<0.2 T (2 kG)) using permanent magnets of the Alnico (limited use now) or barium-strontium ferrite. Rare earth magnets (containing one of the rare earth, or lanthanide series, elements, eg, Sm or Nd) have replaced the ferrites because of the high magnetic force they produce and cost effectiveness (7,13,48). Electromagnets are sometimes used when high field strengths are required. Wet separators are more common. These electromagnets are typically used for removing ferrous metals in heavy industrial applications like coal, limestone, sand and other aggregates. These large scale magnets are available in a variety of configurations including oil and air-cooled; round, hollow and rectangular core; explosion proof; even cryogenic superconducting magnets capable of producing very strong magnetic force.

Dry low intensity separators, such as the magnetic pulleys, drum-type, suspended magnets, and plate and grate magnets, are used for coarse tramp removal from many feed systems to protect downstream equipment such as crushers, screens, and conveyors (13). The magnetic pulley (permanent or electromagnets) is one of the simplest devices (Fig. 22d). Installed at the head end of a conveyor belt, it provides continuous removal and automatic discharge of tramp iron. Electromagnetic pulleys having dc power supply up to 5000 W are used for handling large volumes and removal of larger pieces of tramp iron. Tramp iron drums from either permanent or electromagnets handle material in a reasonably thick layer. These are built with radial poles capable of projecting a deep magnetic field and holding tramp on the drum face until discharged. Suspended or over-the-belt rectangular magnets are used when belt speeds and tramp burden are much greater. These can also be located over chutes or at the end of shaking screen discharges. They can be made self-cleaning when a moving belt is incorporated over the rectangular magnet. Plate magnets are placed in the bottom of chutes and hoppers. Grate magnets consist of parallel tubes or grids of permanent magnets, and are mounted over chutes or troughs handling dry or wet streams. These are gaining popularity because they can handle any volume passing through chutes and are easy to install.

Dry low intensity separators are extensively used in concentration (or cobbing) of coarse, strongly magnetic material (4,13). For example, magnetite, nickel ore, recovery of dense medium from DMS circuits, recovery of iron values from blast furnace and steelmaking slags, and for processing low grade taconites. Wet separation using drums is more common, especially in the finer (<5 mm) size range. The drum separator (Fig. 22) consists essentially of a rotating non- magnetic drum equipped with several stationary, permanent or electromagnets having field intensities up to 0.7 t (7 kG), of alternating poles (4). Variations of this design are available. Feed enters on one side of the drum; magnetic particles travel through the field close to the drum and are discharged at the end, and the nonmagnetics or tailing discharge at the bottom of the drum. The feed flow can be either in the same direction as, or the opposite of, drum rotation. In countercurrent design, the tailings are made to travel in the direction opposite that of drum rotation. Magnetic flocculation often occurs when ferromagnetic particles are present because the particles can behave as tiny magnets and attract each other (7). Magnetic flocculation is used in the concentration of magnetite and the cleaning of steel plant waste water. Such flocs can, however, entrain nonmagnetic particles and thus lower separation efficiency. The flocs can also cause problems in subsequent mineral processing operation. A similar problem exists in polymer-assisted flocculation of minerals (see FLOCCULATIN AGENTS). Demagnetizing coils are used for the depolarization of the particles.

In order to process paramagnetic minerals of low magnetic susceptibility, both a high magnetic field intensity, 1-2 T (10–20 kG), and a high field gradient, 1 T/mm (10 kG/mm), are required (4,13). Dry separation is common, although wet high intensity and high gradient separation are gaining importance rapidly and have widened the application of magnetic separation. Modem units have capacities as high as 120–180 t/h. The common separators are the induced roll, carousel (Fig. 23a) (13), the cross-belt, and the canister (Fig. 23b) (7). The induced roll separator consists of a series of revolving laminated rolls of alternating magnetic and nonmagnetic disks, of slightly different sizes, compressed together on a nonmagnetic stainless steel shaft. These rolls are magnetized by induction using a high intensity stationary electromagnet. Very high field strengths are generated in the gap between the pole and the roll, and the gap size is adjustable. They are typically used for coarse feed (>75 μm) and find application in the concentration of beach sands and other weakly magnetic minerals, and removal of iron contamination from many industrial minerals.

In the well-known Jones high intensity separator (Fig. 23a) a carousel of grooved plates revolves in a magnetic field generated by electromagnets enclosed in air-cooled cases (4,13). The grooved plates concentrate the magnetic field at the tip of the ridges and increase the collection area for magnetic particles significantly compared with the grooved rotor of the induced rolls. Feed enters at two points on the carousel. Paramagnetic particles remain on the plates and the nonmagnetic particles flow through the plate grooves. The paramagnetics are collected at points where the magnetic field is essentially zero. Variations of this design include the use of steel balls, steel wool, or sheets of expanded metal as the ferromagnetic matrix. These separators are used extensively in processing low grade hematite ores. Other uses include processing of ores containing siderite, ilmenite, chromium, manganese, tungsten, etc, and removal of magnetite impurities from cassiterite concentrates, glass sands, asbestos, scheelite, kaolin, talc, removal of pyrite from coal, and processing wolframite, some sulfide ores, and beach sands.

The cross-belt separator is one of the oldest types of separators (13). The feed flows over a conveyor belt, and the magnetics are picked up by another belt perpendicular to the feed belt and moving over the sharp edged upper poles of an electromagnet. The lower poles are situated below the belts and are flat. The disk separator is a modification of the cross-belt and consists of a series of disks containing concentrating grooves and revolving above the feed belt. Electromagnets are used to magnetize the concentrating grooves by induction. This design lends itself to excellent control, sharpness of separation, and selectivity. It is capable of producing a separate middlings product.

Very high (1 T/mm (10 kG/mm) field strength gradients are achieved in the canister-type separators (Fig. 23b) and particles of very low susceptibility can be concentrated (4). Canisters are used typically for very fine particles. A solenoid is used instead of the conventional magnetic circuit design. The core is filled with a matrix of secondary poles such as steel wool in a canister, and these poles produce high field gradients. The weakly magnetic particles are captured in the matrix and are removed periodically. They are used to remove very fine iron-containing impurities from kaolin clays.

Many improvements have occurred in magnetic separators, especially in the high intensity and high gradient units. Many powerful magnet materials have also been developed, eg, neodymium-iron-boron. Magnetic separators using a superconducting magnet have been used (Fig. 23c) (4), eg, processing of kaolin clays (4,48). The industrial units operate at temperatures near absolute zero, achieve magnetic fields of 5 T, and are characterized by greatly reduced power costs (up to 90% over conventional units), high capacities (20–40 t/h), and superior performance. These types of separators are expected to gain in importance as advances in high temperature superconductors are made (see SUPERCONDUCTING MATERIALS). Progress has also been made in the use of magnetohydrodynamic and magnetohydrostatic methods in minerals processing (4,7,34,49). These methods can separate minerals by density, magnetic susceptibility, and electrical conductivity simultaneously (see MAGNETONYDRODYNAMICS).

Many other devices are available for laboratory use. These include the Davis tube, Frantz isodynamic separator, laboratory drum-type separators, low intensity rotating field separator, and superconducting high gradient separator (4).

Electrostatic Separation. Electrostatic separators exploit the differences in electrical conductivities, triboelectric effects, and polarizabilities between minerals (4,7,13,18,46). These have a limited number of applications in minerals processing. They are used extensively in dust removal from gas streams, and are quite successful where applied. They are frequently used in combination with gravity and magnetic separation. Whereas early usage was in the separation of high conductivity gold and metallic sulfides from low conductivity siliceous gangue and separation of sphalerite from galena, principal use in the 1990s is in processing beach sands and alluvial deposits containing titanium minerals (rutile and ilmenite are separated from zircon and monazite) (4). Other concentration methods have been less successful in these latter systems because of the similarities in surface properties and specific gravities of the minerals to be separated. Other uses include beneficiation of cassiterite, columbite, and ilmenite; iron ores; separating halite and sylvite; shape separation of vermiculite and gangue minerals; and industrial waste recovery, eg, plastics from scrap. Minerals pinned on a rotor in an electric field are apatite, barite, calcite, corundum, garnet, gypsum, kyanite, monazite, quartz, scheelite, sillimanite, spinel, tourmaline, and zircon. Those thrown from the rotor are cassiterite, chromite, diamond, fluorspar, galena, gold, hematite, ilmenite, limonite, magnetite, pyrite, rutile, sphalerite, stibnite, tantalite, and wolframite (4).

Electrostatic separations can be either the electrophoresis type, involving charge transfer to or from a particle, depending on the differences in conductivities and triboelectric properties; or the dielectrophoresis type, involving induced polarization owing to differences in dielectric constant, shape, and structure (50). Electrophoresis is the basis of all of the applications in minerals processing. Dielectrophoresis is an emerging technology, but limited to laboratory and pilot plant (see also ELECROSEPARATIONS).

The basic processes in electrophoresis electrostatic separation are partial charging and subsequent separation of charged particles either using a grounded surface or an attracting electrode. Particles can be charged by contacting dissimilar particles, which occurs during bulk movement of particles; ion bombardment or charging in an ionizing field; or induction in a non-ionizing field. Usually a combination of charging mechanisms exists in separators, but the last two are the most important. The separation that occurs at the grounded surface results from the combination of electrical, centrifugal, and gravity forces.

Electrostatic separators are either of the electrodynamic or the electrostatic type (7). Early machines were of the electrostatic type and based on electrostatic processes in charged fields. Charged particles were attracted to an electrode of opposite charge and were lifted from the particle stream toward the electrode (lifting effect). An example is the separation of the negatively charged quartz from other nonconductors using a positively charged electrode. Such separations are inefficient and are sensitive to humidity and temperature.

Most of the separators used are the high tension or electrodynamic type (Fig. 24a) (7), based on the principle of corona discharge in an ionizing field utilizing the focusing or the beam-type electrode design (4,7). The particle stream is fed on to a rotating metal drum or rotor which is grounded. Particles enter a field of charged ionizing electrode assembly which spans the entire length of the drum and supplied with a dc voltage of up to 50 kV and a negative polarity (current flow is usually 5–15 mA/m rotor length). The electrode assembly itself comprises a fine wire electrode to produce the ionizing field and a large-diameter electrode to produce a dense non-discharging field. Particles are charged by ion bombardment. Particles that have high conductivity lose their charge to the grounded drum and therefore are thrown from the drum surface by centrifugal force and aided by a non-discharging static electrode that is placed after the ionizing electrode. Particles having low conductivity and nonconductors retain their charge and are pinned to the drum surface by their own image charges. These particles slowly lose their charge and falloff the drum, the middling particles losing their charge faster than the nonconductors. Any nonconductors still pinned to the drum are removed by a brush, in some cases aided by an additional electrode. The voltage supplied to the electrode assembly is sufficiently high to promote ionization but no arcing. The voltage range of stable corona discharge is narrow. The separation is influenced not only by the conductivities of the particles, but also by the degree of liberation and the particle size distribution because of differences in the charge per unit mass on fine and coarse particles. There is a tendency for the fine particles to be pinned to the drum for a longer time and report to the middlings or the nonconductors fraction. Several stages of cleaning are therefore required.

Plant capacity is a function of feed size distribution and liberation. Separators can accept a size range as wide as 50–1000 μm. Capacities are typically 1000–2500 kg/(h.m) based on rotor length which could be up to 3 m and have dia 150–250 mm. The feed should be as dry as possible because moisture interferes seriously with separation. Heaters are usually provided before the feed enters the charged field. Final cleaning is often conducted in electrostatic-type separators. Electrostatic shape separation, a newer form of ion bombardment separation, involves separation of particles based on shape and density without consideration to conductivities (50).

Electrostatic separators are either rotor or plate type (7). The former is similar in appearance to the high tension separator. However, there is no ionizing electrode in the electrode assembly, instead there is a large single electrode producing an electric field. Particle charging is by induction in this case. The paths followed by conductors and nonconductors are similar to that in high tension separators. Modem plate-type separators are either plate (Fig. 24b) or screen electrostatic types. The particle stream passes over a sloping, curved, grounded plate into an electrostatic field induced by a large curved electrode. The conductor particles acquire a charge opposite to the electrode and are lifted toward it. The particles go over a splitter or a screen. The nonconductor particles continue down the plate or through the screen. Because the action of the field is mainly on the conductors, a sharp separation can be made. Fine particles are more affected by the field than the coarse particles; therefore the latter are readily rejected, unlike in high tension separators. The plate-type separators are used for removing small amounts of nonconductors from large amounts of conductors and vice versa for the screen-type separators. The main usage of electrostatic-type separators is in final cleaning of concentrates produced by other methods.

Electrostatic separation is sensitive to humidity and moisture, temperature, and any organic coatings. The feed is usually cleaned by washing, attrition scrubbing, or caustic scrubbing and then dried. Desliming is also practiced where necessary because slimes interfere with the charging of particles. Improved selectivity can often be achieved at elevated temperatures. Normal practice in rutile separation from zircon is to operate at 90°C or higher. Electrostatic separators are equipped with heating coils and lamps.

Froth Flotation. Flotation (qv) is the most extensively used primary mineral concentration technique, and the tonnage treated is the largest among physical separation methods (51–55). Its introduction in the early 1900s revolutionized the industry (see Ref. 53 for a history of the development of flotation). The Centenary of Flotation was celebrated in a special conference in June, 2005 in Brisbane, Australia.

The attributes of flotation are its applicability to a wide range of minerals systems, ore types, and size ranges, and its high versatility and selectivity. Flotation is used to treat all of the sulfide ores, precious metals ores, coal (qv) and the majority of non-sulfide ores. In flotation, one or more types of particles are separated from others by actually floating the particles against gravitational forces with the aid of air bubbles introduced into the separator. The science of flotation is complex though the process itself is seemingly simple.

Most minerals are naturally hydrophilic. The surface chemical properties of the minerals to be floated are changed selectively to make the surfaces hydrophobic (water repelling) by the use of organic reagents called *collectors*. A simple example is the flotation of a small (<5%) amount of copper sulfide minerals, eg, chalcopyrite, chalcocite, etc, and molybdenite from a copper ore containing siliceous gangue minerals constituting >95% of the ore. Small, organic sulfur-containing compounds, such as dithiophosphate or xanthate, are used as collectors to coat metal sulfides so that these particles are rendered hydrophobic and, therefore, can attach to air bubbles. The bubble-particle aggregates then float to the surface of the pulp where they form a stable froth (Fig. 25) (4). This froth is removed continuously by either natural overflow or mechanical means. A stable froth is ensured by the addition of suitable chemicals, called *frothers*, which also facilitate the production of a fine dispersion of bubbles in the pulp. The froth must be stable only long enough to complete the removal of the floated particles and break readily in the launders where it is collected. In order to sharpen the mineral separation (ie, improve selectivity) reagents called *modifiers* are used. The main function of the modifiers is to modify the surfaces of the minerals and/or the chemistry of the aqueous phase in order to achieve desired selectivity of separation; in other words, they modify the functions of collector and frother by changing the interfacial properties of S/L, L/G and S/L/G interfaces. Modifiers frequently have many roles and they are involved in complex interactions with minerals and the aqueous phase. Modifiers can be sub-divided into *activators*, *depressants, pH modifiers*, and *dispersants*, depending upon their primary role. Activators modify the surfaces of the desired mineral directly to promote collector adsorption which otherwise would not occur. Depressants act on other (undesirable) minerals in order to prevent collector adsorption on these; their function is, in a broad, sense, opposite of that of collectors. Dispersants have many roles to play. One of them is to ensure that very fine gangue minerals (or slimes), such as clays and other silicates, do not interfere with collector adsorption on desired minerals and their subsequent transfer to the froth phase; they can also control slurry rheology. pH modifiers are common acids and bases such as lime, caustic, soda ash, and sulfuric acid (see HYDROGEN ION ACTIVITY). A wide variety of flotation reagents is used in the industry to make impressive separations in non-sulfide and sulfide flotation (56).

The three types of flotation reagents–viz. collectors, frothers and modifiers–can be represented as a flotation reagent triangle, implying that all of these are of critical importance and the flotation outcome is determined by the complex interactions between these reagents and the various minerals in the pulp. In terms of flotation practice, separation schemes and the reagents used, sulfide ore flotation is distinctly different from non-sulfide ore flotation, and the differences are readily understood by the fundamental differences in the chemical and surface properties between sulfide and non-sulfide minerals (as also differences between S and O (57).

It is the buoyancy of the bubble-particle aggregate that determines flotation outcome, not the specific gravity of the particles, particle size, or particle shape. Thus even the heaviest, eg, native gold, sp gr 19.3, and coarsest, eg, ≤ 3 mm for sylvite flotation, of the particles can be made to float if they can attach to a large enough bubble or if sufficient number of bubbles can be attached to each particle. In a loose sense, flotation can be considered a variation of a gravity separation technique or chemically-assisted gravity separation. Large differences in gravitational forces acting on mineral-laden bubbles and other (unattached) particles are exploited. It is generally preferable to float a small mass of particles, either value minerals or impurities, away from the rest of the ore. As for any other concentration method, flotation efficiency is affected by particle size, falling off at both very coarse and very fine sizes. The size range of optimum flotation is different for different mineral systems, but successful separations can be made down to 1 μm (4,7). The 10–150 μm range is considered the best in sulfide mineral flotation for example. Flotation is a delicate balance between buoyancy and gravitational forces. It is a probabilistic (or more accurately, pseudo-probabilistic) event, in that each of the sub-processes leading to flotation has a finite probability, which is influenced by chemical, physical-mechanical and operational factors (or variables) acting simultaneously (see Fig. 26) (58). More than 20 variables have been found to influence flotation outcome, which in a plant operation is highly variable, and is almost always a trade-off between the chemical, physical and operational factors.

Collectors. Collectors play a critical role in flotation (54). These are heteropolar organic molecules characterized by a polar functional group and a nonpolar hydrocarbon group, usually a simple 2-18 carbon atom hydrocarbon chain, that imparts hydrophobicity to the minerals surface after the molecule has adsorbed on the mineral via the functional group. The functional group, or ligand, comprises one or more of the important donor atoms, N, O, S attached to a central atom of C, P, N or S. The donor atoms are involved in bonding with metal or other active sites on the mineral via chemical or physical interactions. The type, number and geometric positioning of the donor atoms influence strength and selectivity of adsorption of collectors on minerals. Adsorption strength and selectivity are also influenced by the hydrocarbon groups; for eg, the type (alkyl vs. aryl), chain length, branching, presence of hetero atoms (sometimes additional donors) such as N, O & S. Most collectors are weak acids or bases or their salts, and are either ionic or neutral. The mode of interaction between the functional group and the mineral surface may involve a chemical reaction, for example, chemisorption, or a physical interaction such as electrostatic attraction. Pearson's Hard-Soft Acid-Base (HSAB) concepts are quite readily applicable to the interactions between minerals and collectors (57).

In general, collectors for the flotation separation of sulfides and precious metals contain at least one sulfur atom in an appropriate bonding state. Examples of the important families of collectors are given in Table 8 (56,57). The functional group in collectors for non-sulfide minerals is characterized by the presence of either a N (amines) or an O (carboxylic acids, sulfonates, hydroxamate, etc.) as the donor atoms (See Table 8). In addition to these, straight hydrocarbons, such as fuel oil, diesel, kerosene, etc, are also used extensively either as auxiliary or secondary collectors, or as primary collectors for coal and molybdenite flotation. The chain length of the hydrocarbon group is generally short (2–8 C) for the sulfide mineral collectors, and long (10–20 C) for non-sulfide collectors, because sulfides are generally more hydrophobic than most non-sulfide minerals (54). In the case of sulfide flotation, a wide variety of collectors, many of which exhibit a high degree of selectivity for a given sulfide, is available for the relatively small number of sulfide minerals floated. In contrast, only a small number of collectors, many of which are rather non-selective, is available for the large number of non-sulfide minerals floated.

Of the 15 distinct families of S-containing collectors for flotation of sulfide minerals, xanthate discovered in 1925 used to be a dominant collector in sulfide flotation until about 25 years ago. Collectors from the other 14 families have gradually replaced or augmented xanthate in most applications, driven primarily by industry needs and increasingly complex technical challenges (29). Search for novel, high-performance flotation reagents is an active area of research. Fatty acids, amines, and sulfonates dominate the non-sulfide flotation usage. The fatty acids are by-products from natural plant or animal fat sources (see FATS AND FATTY OILD). Similarly petroleum sulfonates are by-products of the wood (qv) pulp (qv) industry, and amines are generally fatty amines derived from fatty acids. Collectors used for non-sulfide minerals are generally non-selective; therefore, modifiers are invariably used to provide selectivity, and they play a very critical role. Many of the collectors discovered or used until the early 1950s were chemicals that had been developed in other industries such as rubber, textile, tanning and consumer products. The majority of the sulfide collectors, for example, were developed or used in the rubber industry as vulcanizers (21). Targeted design of flotation reagents, ie, reagents specifically developed and manufactured for flotation applications, began in early 1950s. Since then several novel flotation reagents have been developed that are used almost exclusively in flotation (59).

The amount of collector used is necessarily very small because surface coverage of an equivalent monomolecular layer or less is required to impart sufficient hydrophobicity to the mineral (in research contact angles are used as a measure of hydrophobicity). Even a small hydrophobic patch on the mineral is sometimes sufficient to promote attachment to bubble and cause flotation. The usages typically range from 1−100 g of collector per ton of ore treated for sulfide flotation (typically 0.2−10% value metal content in the ore) and 100−1000 g/t for non-sulfide flotation (1–30% value mineral content) (13).

The adsorption of sulfide collectors on sulfide minerals can be best described by electrochemical reactions wherein the mineral, the collector, or both are known to undergo redox reactions (40). This process is unique to sulfide mineral systems. Most sulfides exhibit metallic properties and undergo electrochemical reactions that are much like corrosion reactions exhibited by metals. The majority of the sulfide collectors adsorb on sulfide minerals by forming a metal complex on the surface. Direct unequivocal evidence for surface complexes has been obtained recently from SIMS (60). Redox reactions are not relevant in non-sulfide mineral flotation systems. In these latter systems, the adsorption is generally a chemisorption, surface chemical reaction, or a physisorption, eg, electrostatic attraction between oppositely charged mineral surface and collector, and is often a combination of these processes (40). Surface charge on the minerals, as approximated by zeta potentials, is therefore of relevance in non-sulfide systems. Because surface charge on oxides and silicates is strongly dependent on pH, the adsorption mechanism for a given collector, the choice of a collector, and the flotation selectivity are all influenced strongly by pH.

Sulfide collectors in general show little affinity for non-sulfide minerals, thus separation of one sulfide from another becomes the main issue. The non-sulfide collectors are in general less selective and this is accentuated by the large similarities in surface properties between the various non-sulfide minerals (55). Some examples of sulfide flotation are copper sulfides flotation from siliceous gangue; sequential flotation of sulfides of copper, lead, and zinc from complex and massive sulfide ores; and flotation recovery of extremely small (a few ppm) amounts of precious metals. Examples of non-sulfide flotation include separation of sylvite, KCl, from halite, NaCl, which are two soluble minerals having similar properties; selective flocculation-flotation separation of iron oxides from silica; separation of feldspar from silica, silicates, and oxides; phosphate rock separation from silica and carbonates; colored impurity minerals from kaolin clays; and coal flotation.

Frothers. Frothers are generally aliphatic alcohols (C5-C6), polyglycols (qv), or polyethylene or polypropylene glycol monoethers (54). They are, therefore, heteropolar molecules having surface-active properties. Until about 30 years ago, reagents such as pine oils (eg, alpha-terpineol, or mixtures of terpineols) and cresylic acids were used widely. They have very limited use today, mainly due to cost and environmental issues. Methyl isobutyl carbinol, a C6 aliphatic alcohol, is one of the widely used frothers. The polyglycols and their ethers (methyl to butyl ethers) have a MW in the range of 200−500. Frother requirements are strongly dependent on the ore being treated and the other flotation reagents (collectors and modifiers) used. A typical range for sulfide flotation is 5−100 g/t. The non-sulfide mineral collectors such as fatty acids, petroleum sulfonates, and amines exhibit frothing properties that are sufficient in some applications so as not to warrant the use of a separate frother; however, in several non-sulfide flotation operations, a separate frother addition is made to improve efficiency.

The role of a frother in flotation is rather complex despite its seemingly simple function. The froth phase in flotation was largely neglected in flotation research to some extent in flotation practice, in stark contrast to its importance. This is not surprising, however unjustified, given the complexity of the froth phase, which indeed is the ultimate separator in froth flotation process as practiced in most operations using mechanical cells or columns; in other words, "froth phase = flotation". And indeed froth phase characteristics are influenced by chemical, mechanical and operational factors, which is a statement synonymous with "flotation outcome is influenced by chemical, mechanical and operational factors." The froth phase properties are affected by the type, size, and hydrophobicity of the various mineral species present as well as the composition of the aqueous phase (containing various inorganic ions released from the minerals, and all the flotation reagents). Essentially, anything that is put into the flotation cell should affect froth characteristics, in addition to physical and operational factors. There is now a concerted effort in many research institutes to understand the froth phase in flotation in order to develop techniques to manage and control its properties to maximize flotation performance (58,59).

Modifiers. Modifiers assume a critical role in many separations (29,54). Unlike collectors and frothers, which are predominantly designed and targeted synthetic reagents, modifiers are dominated by readily available inexpensive inorganic compounds and natural products. Modifiers fall into three major categories from purely chemistry point of view: Inorganic, small organic and polymeric molecules. Most of the modifiers used are inorganic, and were developed a long time ago. Table 9 (56) shows the important modifiers used in flotation. Small organic molecules have found limited commercial use; cost is a major factor. Until the early 1980s, the use of polymeric depressants was limited to polysaccharides and other natural products and byproducts. Several synthetic, water-soluble, low MW polymers are now in commercial use as depressants (for gangue iron sulfides, phosphate and silicate minerals) (61).

The use of dispersants has been sporadic in sulfide flotation. Their main usage is in addressing slime coating of sulfides and precious metals, and in dealing with high viscosity in pulps caused by certain gangue silicate minerals, such as serpentines, chlorite and sericite. The important modifiers used in these applications are low MW polyacrylamide-based polymers (eg, polyacrylates), soda ash, lignin sulfonate, CMC, polyphosphates, low MW polyethylene oxides. It is well known that dispersants can have significant influence on behavior and properties of the froth phase. This aspect has not received much attention in the past. The use of dispersants in non-sulfide flotation is very significant. In some systems, dispersants greatly facilitate in liberation of the value minerals (eg, beneficiation of kaolin clays) and are absolutely essential. The most commonly used dispersants are sodium silicate, polyphosphates, soda ash, lignin sulfonates, tannins, CMC, and polyacrylates. Industry choice of modifiers is predominantly from what is readily and naturally available products which do not provide much room for structural modifications and performance improvement. In order to exploit the role of modifiers in flotation, it is perhaps necessary to develop synthetic modifiers, such as polymeric depressants, to tailor them for specific separations. Research in this area is sporadic and insignificant. Environmental issues may also force research into developing acceptable alternatives to several of the modifiers used currently (29).

In addition to modifiers added deliberately, significant amounts of unavoidable ions and species released from the minerals are also present in flotation pulps, and they act as modifiers, albeit inadvertently. The unavoidable ions and species formed in this process react with water, with each other, and with other reagents forming complex ions and molecules and also colloidal and coarsely dispersed suspensions and precipitates. These species in turn adsorb on, or react with, the mineral surfaces and air bubbles and change their surface compositions. The direction and kinetics of these processes depend on the concentration of the acting components, the temperature (seasonal variations), and to a large degree on hydrodynamics. The situation is complicated by large recirculating loads and recycled water in plant circuits. A major consequence of unavoidable species released from the various mineral components is inadvertent activation of certain value minerals. For example, inadvertent activation of sphalerite by Cu or Pb causes significant amounts of Zn reporting to Pb and Cu concentrates; similarly inadvertent Cu activation of pyrite and Ca activation in non-sulfide mineral separations. Inadvertent activation is often difficult to predict since it is strongly influenced by mineralogy and pulp conditions which exhibit significant variability in a plant. Indeed one of the main functions of modifiers added deliberately (for eg, lime, soda ash, sodium silicate etc.) is to control the effects of unavoidable species (59).

Of the three major reagents forming the flotation reagent triangle, modifiers have perhaps the greatest effect on rate of flotation of particles. This effect can be either positive (enhancing flotation rate) or negative (decreasing flotation rate), and it can be temporary (ie, the mineral can be re-floated in another stage with relative ease, eg, non-ionic polymer) or permanent (ie, true depression; the mineral can be re-floated only after drastic measures have been taken; eg, a change in chemical environment; eg, NaCN depression of pyrite). Thus one modifier may merely decrease the flotation rate of one mineral selectively and sufficiently so that a selective flotation of another mineral is possible (59).

The chemistry of modifiers in flotation pulps is quite complex in comparison to that of collectors; and it is more complex in non-sulfide systems than it is in sulfide systems. Modifiers can affect multiple factors simultaneously in flotation pulps and, therefore, have multiple functions. Thus, depending upon the type, modifiers can change the pH, E_p (pulp redox potential), composition of the aqueous phase and surface composition of the minerals and bubbles. They can affect properties of all three interfaces. They are also known to affect dispersion and rise-velocity of bubbles. Same modifier in different circuits could have very different results, or different modifiers used for pH control can have very different results even at the same pH (59).

In addition to the use of modifiers, numerous other techniques such as high intensity and high solids conditioning (attrition, scrubbing), desliming, selective flocculation of slimes, etc, have been developed, especially for non-sulfide mineral systems.

Flotation Equipment. Numerous designs of flotation machines are available (4,7,40,44,62,63). Mechanical flotation machines are the most widely used; however, flotation columns have seen a rapid acceptance in the industry. Pneumatic machines, popular in the early years of flotation, are receiving renewed interest. Most flotation is carried out in banks of flotation cells. The primary functions of the flotation machine are to maintain all particles in suspension, to disperse fine air bubbles throughout the pulp and promote particle-bubble collision, to provide a quiescent pulp region close to the froth phase to minimize mechanical entrainment of unwanted particles into the froth phase, or to prevent turbulent disruption of the froth layer. The froth phase should be of sufficient depth to permit drainage of entrained particles. The selection of a flotation machine is dictated by metallurgical performance (grade and recovery of valuable minerals), capacity, operating costs, and ease of operation.

The main feature of a mechanical flotation machine is the impeller surrounded by baffles, both designed to provide adequate suspension of pulp, in a rectangular tank. Aeration (qv) is via either self-induced or more commonly forced air, in both cases through the hollow impeller shaft or through a standpipe surrounding the impeller shaft, to provide good dispersion and particle-bubble collisions. There are numerous designs of impeller/baffle assembly, each claimed to be better than others (40,55,62). All provide the same basic requirements: a turbulent zone for particle-bubble collision and a quiescent zone below the froth phase which allows for efficient transfer of the mineral-laden bubbles to the froth. A dramatic increase in the size of the flotation cells has occurred such that 14−150 m^3 cells are typically used (44,63). Recently a 300 m^3 cell was introduced. The main advantages of these large cells are better efficiency, smaller space requirement, higher throughput, and lower power consumption. Froth collection from the mechanical cells is via either direct overflow or using paddles. Pulp flow from cell to cell down a bank is most commonly accomplished by gravity. Individual cells may or may not be separated from the others by baffles and overflow weirs. Flotation cells can handle pulp densities in the range of 10−50% solids, and are dictated by the type of ore treated and the metallurgical objectives.

In pneumatic machines (Fig. 27a) (4) air is either mixed with pulp by turbulent pulp addition or blown or induced in cells. In any case air is used not only to produce froth but also to keep the pulp in suspension. One of the more recent designs is the Ekoflot pneumatic cell, very similar to the Jameson cell (4) in which aerated pulp is introduced near the bottom of a cylindro-conical vessel at a controlled speed. Froth overflows at the top and the tailings are collected at the bottom. The largest unit has a 6-m diameter vessel and can handle up to 1500 m 3/h and 100−700 t/h depending on application. Dissolved air flotation (7), which can be grouped under pneumatic flotation, utilizes air dissolved under pressure in the pulp to form bubbles on fine solids, and it is used primarily in the treatment of industrial effluents. In froth separators (4) developed in Russia, aerated pulp is introduced on top of a froth bed in an inverted pyramidal tank. Hydrophobic particles are retained in the froth and the remaining pulp descends and is discharged at the bottom.

In a flotation column (Fig. 27b) (4), feed slurry is introduced in the top half of the column, and air, sometimes premixed with pulp, is introduced in the bottom half of the column. Numerous variations of feed and air inlet designs have been used. Flotation columns can be up to 18 m high (4,44), having diameters of up to 3.5 m. Columns having much smaller height/diameter ratios have also been developed, eg, the Jameson cell. In this cell, air and feed are mixed at the head of a long vertical pipe and the aerated pulp is introduced near the bottom of a cylindrical vessel with a conical bottom (4). Flotation columns generally have no agitators or moving parts inside. Water sprays are used invariably above the froth phase in order to provide efficient drainage of the mechanically entrained particles. A vast amount of research and development has been conducted on column flotation. Columns have become an integral part of a flotation circuit in most operations. One of the attractive features of a column cell is its ability to make excellent separations even in the very fine ($<20\,\mu m$) particle size range.

A considerable amount of research on novel flotation systems has been carried out. Several have been developed, eg, the air-sparged or air-injected hydrocyclones, and the rapid flotation device (44). The air-sparged hydrocyclones have a high throughput per unit volume. Significant advances have been made in research and development of modeling, automation, and control of flotation circuits and froth phase management (16,20,18,55,63). Numerous on-line monitoring, measurement, and control systems are part of a flotation circuit. Froth cameras to monitor and manage froths in plants are becoming popular. In a sense these cameras are replacing the experienced eyes of flotation plant operators whose numbers are on the decline. The basic flotation circuit comprises roughers, scavengers, and cleaners. Recovery of the values is the primary emphasis in the roughers and scavengers; grade is the primary emphasis in the cleaners. The most common method of assessing flotation circuit performance is via the mass and metallurgical balance which provides the recovery and grade of value minerals recovered at each stage of the circuit. Several excellent techniques have been developed to perform the metallurgical balances. Much effort has been made to ensure a proper estimation of recirculating loads and the mass balance of the various flow streams. Grade-recovery relationships are most commonly used to assess the efficiency and selectivity of separation.

Flotation is now used increasingly to remove all potentially hazardous minerals (eg, all sulfide minerals which could contribute to acid mine drainage, or minerals containing toxic elements such as As) from ores and tailings, in order that the tailings (which constitute the bulk in many plants) are disposed safely.

Flocculation and Agglomeration. Selective flocculation and selective agglomeration are two other processes based on surface chemistry changes of mineral particles (4,7,34). Both are used commercially to a limited extent, the former for hematite, clay, and potash ores, and the latter for coal and fine metallic oxide minerals. In selective flocculation, water-soluble polymers (qv) such as polysaccharides or synthetic polymers are used to selectively flocculate fine particles of one mineral. The non-flocculated minerals are removed subsequently by desliming or flotation. In selective agglomeration, also referred to as hydrophobic coagulation, one type of mineral particle is rendered hydrophobic, or is naturally hydrophobic, eg, coal, and is forced to agglomerate by using a combination of hydrocarbon oils and high shear conditions. The agglomerate size is increased until a subsequent size separation is possible either by flotation or sedimentation technique.

1.8. Solid-Liquid Separation. Most minerals processing operations are conducted in large quantities of water (see Fig. 28) (26) which shows volume %

water in various unit operations. A typical copper ore flotation plant uses about 3800 L/t of ore treated. Water usage can be as high as 23,000 L/t in glass sands flotation. Dewatering (qv) of mineral slurries to varying degrees becomes necessary for a variety of reasons; for example, subsequent treatment by pyrometallurgical operations (such as roasting, pelletizing and smelting), transportation, disposal, water recovery for recycle, etc, require dewatering.

Dewatering is performed by using one or more of the following methods: sedimentation, also known as settling or thickening; filtration (qv); and thermal drying (qv) (4,7,13,18,34,64,65). Frequently all three are used, in that order, on the same slurry to ensure that the final product has a low moisture content. Thickening is generally the most economical method for tailings dewatering before disposal in the tailings pond.

Sedimentation. Sedimentation is used to remove the bulk of water from streams (thickening). The thickened product is usually 55–65% solids. Sedimentation is also used to remove suspended solids from a relatively dilute stream to produce a liquid phase that is as clear as possible (clarification). Although the same equipment can often be used as a thickener or a clarifier, the distinction is based on operation. Thickeners operate with a clear solid-liquid interface and their capacity is dictated by the underflow conditions. In clarifiers there is no well-defined interface, and the capacity is dictated by overflow clarity. Polymeric flocculants are used in most of the thickeners to enhance the settling rates of particles by flocculating the solids, especially the fines. Synthetic flocculants, such as polyacrylamides and polyacrylates, are used extensively (see FLOCCULATIVY AGENTS). Polysaccharides are also used to a significant extent in certain systems. The synthetic flocculants are available in a wide range (several thousands to over 20 million) of molecular weights and charge (0–100 mol %) (4). The type of flocculant, ie, anionic, cationic, or nonionic, depends on the type of minerals involved. Often a combination is used. Anionic polymers having very high molecular weights can be produced; whereas cationics tend to be of much lower molecular weight. Electrostatic interactions between the charged minerals and the polymers and particle bridging are two important mechanisms leading to flocculation of solids. The physical characteristics of the flocs formed, ie, floc density, size, etc, have a strong influence on the settling and consolidation of the sludge. Flocs are generally quite fragile. Although flocculation leads to significant increase in settling rate or improves filtration rate, it is often detrimental to final consolidation of the sludge or to producing low moisture filter cakes.

There are three types of thickener designs: cylindrical, lamella, and deep cone. The cylindrical design is the most common (Fig. 29) (4). It is also continuous. It comprises a large (up to 200 m dia, 1–7 m deep) cylindrical tank, a shallow conical base (80–140 mm/m), and a central structure carrying sludge-raking arms. Feed enters the thickener through a central feed-well and clarified liquor overflows into a launder around the periphery of the cylindrical tank. Thickened sludge collects in the shallow conical base and is raked by the slowly revolving mechanism to a central discharge point. There are many designs of the feed-well: conventional, bottom-fed, counterflow, perforated, and deep feed-well. The rakes help to move the sludge toward the central discharge and help improve settling and consolidation of the sludge. Typical tip speeds are 5–8 m/min and torque ratings up to 13,000 kNm (890 lbf/ft) (65). Several designs are available for the rake support and drive mechanism.

The other types of thickeners include the high capacity/rate/compression tray, lamella, and the deep cone type (18,65). These are designed to reduce floor area requirements and take up only 5–20% of the floor area required for a conventional thickener and produce denser sludges and clearer overflow. In the high capacity thickener, feed and the flocculant enter the hollow drive shaft and are mixed by staged mechanical mixing. The flocculated feed is injected into the sludge blanket where further flocculation occurs. Settling occurs along inclined plates. The lamella thickener has packs of sloping parallel plates in the settling area which reduce the settling distance and increase the effective settling area. The unit can also be vibrated to enhance dewatering. The same unit can function as both a thickener and a clarifier. The tray thickener is a series of independent conventional thickener units stacked vertically having a common central drive shaft (4,13).

Filtration. In many mineral processing operations, filtration follows thickening and it is used primarily to produce a solid product that is very low in moisture. Filtration equipment can be either continuous or batch type and either constant pressure (vacuum) or constant rate. In the constant pressure type, filtration rate decreases gradually as the cake builds up, whereas in the constant rate type the pressure is increased gradually to maintain a certain filtration rate as the cake resistance builds. The size of the device is specified by the required filter surface area.

Most continuous vacuum filters are the constant pressure type. Their main use is in dewatering concentrated slurries such as concentrates. They belong to three classes: the disk, drum, and horizontal filters. Disk, and to a lesser extent, drum filters, are the mainstay for most final dewatering. These filters remove most fine particles from a process stream.

The drum filter consists of a horizontal cylindrical drum, having from 1 to 5 m diameter, that rotates while partially submerged in an open feed slurry tank. The filter medium is wrapped tightly around the drum surface. The drum shell is divided into compartments and drain lines are connected to the central valve system allowing either vacuum or pressure dictated by the cycle. A normal cycle comprises filtration, draining, and discharge by a blast of air or by mechanical means. Cake washing and filter cloth cleaning can also be part of a cycle. Several variations of the standard drum filters exist, including hyperbaric filters, ie, pressure filtration, up to 600 kPa (87 psi), which are continuous and give high filtration rates and drier cakes (44).

The disk filter is similar to the drum in operation, but filtration is conducted using a series of large diameter filter disks that carry the filter medium on both sides of the disk. They are connected to the main horizontal shaft and partly immersed in the feed slurry. The central shaft is connected by a set of valves which serve to provide vacuum and air as in drum filters. As the disk sections submerge during rotation, vacuum is applied to form a cake on both sides of the disk. The cycle of operation is similar to that in a drum filter. One unit can have as many as 12 disks of up to 5-m diameter. Disk filters, both compact and cost effective, are used extensively in the iron ore industry to dewater magnetite concentrates.

In the horizontal continuous vacuum filters, also called belt filters (Fig. 30) (4), filtration, washing, and drying occur on a traveling belt filter cloth which is provided with suction boxes underneath (4,13). Many variations are available. The belt can be linear or circular. Slurry is fed on the belt at the beginning. Filtration is by both gravity and suction. Filter cake is discharged from the belt using scrapers before the belt reverses. Some advantages of belt filters are relatively low capital and operating costs and excellent washability, but they require high flocculant dosage and greater floor area for a given filter area.

Pressure filters or filter presses are commonly of the batch type (4,66). These are characterized by high filtration rates, smaller floor area, and lower capital cost. Dryer cakes are produced. The plate and frame presses (Fig. 31) (4) and the chamber presses are the most common types of pressure filters used. These consist of a series of vertical, alternating parallel frames and plates. The filter cloth is held against the plate. Cake formation occurs in the hollow frame. The fully automatic Larox chamber filter, reported to reduce dewatering energy requirements, is a more recent development (66). Other developments are the tube press or the pinch press which can operate at 10,000 kPa of dewatering pressure. These units produce very low moisture filter cakes even in the absence of dewatering acids (67,69).

Flocculants and surfactants (qv) are used frequently as filter aids, particularly when slimes are present or when the particles to be filtered are very fine and difficult to filter. Low molecular weight polymers are more commonly used. These form small, dense flocs which provide higher cake porosity. Blinding of the filter cloth by fine particles or slimes is reduced. Surfactants are also used to enhance flow through the filter cake pores.

Centrifugal Separations. Centrifuges are used when conventional dewatering methods are not applicable because the settling rates are too slow, as in the case of clays, or when low moisture levels are required before the next unit operation (4,7). The centrifugal force is used in these devices to enhance solid-liquid separation. These are high capital cost and high maintenance units, but can perform many functions, eg, as classifiers, thickeners, clarifiers, and filters. The hydrocyclones are an example of centrifugal devices used for classification or thickening that are simple and inexpensive. These do not, however, produce very high solids concentration in the underflow and their efficiency falls off rapidly at fine size ranges, ie, $<10\,\mu m$ particles invariably appear in the overflow. Hydrocyclones are still valuable for initial dewatering before thickening. Many other classifiers can also be used for primary dewatering.

Centrifuges are either the solid bowl or the perforated basket type. The solid bowl centrifuge consists of a horizontal bowl rotating at a high (1000–6000 rpm) speed. Slurry enters in the center of one end of the bowl, the liquid discharges at the bottom of the same end, and the solids are discharged at the other end by means of a rotating helical scroll. Further dewatering occurs as the solids move along the scroll. Washing can be incorporated at this stage. The size of the bowl is dictated by the extent of dewatering required and the application. These are particularly suited for clarification.

In the perforated basket centrifuge, material transport is through vertical vibrations in the basket. These vibrations also loosen the bed of particles, aiding drainage, thus allowing lower (550–750 rpm) speeds than are necessary for solid bowl centrifuges. These are commonly used in dewatering coal. Because of the perforations in the basket, these centrifuges are not suitable for feeds having a significant proportion of fines.

Thermal Dewatering. Thermal drying is typically the last stage in dewatering (4,7,13). It is an expensive operation because energy is wasted in heating solids, but is necessary in many cases because mechanical dewatering cannot reduce the moisture level below a certain limit. The extent of reduction in moisture by thermal drying is dictated by economics and specifications required for the mineral product in terms of flow properties of particles, dust prevention, etc. A moisture level of 5% is often acceptable and complete drying is not necessary.

The common types of dryers are rotary, hearth, flash (spray), and fluidized beds (13). Hot gases are used invariably to remove moisture. The gas flow can be either cocurrent or countercurrent to the flow of solids, the former tends to be more efficient. In the hearths, the gas flow is countercurrent as the solids are raked down from one hearth to the next below. Flash dryers are very rapid because the solids are exposed only briefly to the hot gases. Fluidized bed dryers, which use hot gases to suspend the solids, are rapid and efficient, but require elaborate dust collection systems. These are preferred when fine solids are involved, and are used commonly for drying fine coal. Indirect-fired dryers are used when the solids are heat sensitive or combustible.

1.9. Materials Handling. The management of transportation, storage, feeding, washing, and packing of processing streams to final products in a mineral processing operation constitutes a significant effort and a cost factor. Materials handling comprises dry solids and slurry handling, and tailings disposal.

Dry solids, such as as-mined ore, crushed ore, and dried concentrates, are transported using trucks, rail cars, ore passes, conveyor belts (see CONVEYING), or slurry pipelines (qv) as dictated by the logistics, distances involved, and capacity. Within the mill, conveyor belts are more common, but slurry transportation is more typical for fine particles, tailings, and coal (4,7).

Storage of dry solids provides for surge capacity between unit operations and for blending ore types. Stockpiles are preferred for coarse material and large capacity such as crushed ore, sized products, and dried concentrates ready for shipment. Bins are preferred for smaller capacities. Stockpiles are formed on a concrete or earthen pad using a variety of equipment such as fixed stackers, tripper conveyors, reversing shuttle conveyors, traveling winged stackers, and radial stackers. Material from a stockpile is reclaimed using bottom tunnels, bucket-wheel reclaimers, scraper trucks, or front-end loaders. Bins and hoppers provide a greater flexibility in terms of transportation, storage, blending, and controlled feed to subsequent stage and their design is crucial. Improper design for a specific application may result in total system failure or extensive operation problems associated with intermittent feed. Feeders are usually part of the bin and hopper unit. Feeder design types are belt or apron, screw, rotary table, vibratory, star, and rotary plow. Belt and apron and the vibratory types are the most common. Material from a stockpile or a bin is transported using chutes; mechanical conveyors, ie, belt, screw, chain, and vibratory type, that are horizontal or inclined; pneumatic conveyors; skip hoists, used widely to haul ore from underground; and bucket elevators (see CONVEYING).

Much effort is made to obtain a representative sample from bulk dry solids. This can be a difficult task for very coarse material and from large stockpiles, bins, and hoppers as segregation is inevitable. It is, therefore, preferable to collect samples or subsamples when the material enters the stockpile. Cross-chute and the rotary samplers are the most commonly used (see SAMPLING).

Slurry handling in minerals processing includes transportation and suspension in tanks during processing. Transportation can be from mine site to the processing plant, as for hydraulically mined ores such as clays, phosphates, beach sands, and cassiterite; within the processing plant, eg, from grinding to dewatering; and from the processing plant to another location, eg, transport of tails to a tailings pond, or transport of concentrate slurries through a pipeline to a dewatering plant or a shipping port. Powerful pumps (qv) are required at every stage, and the selection of appropriate pumps and pipeline sizing constitutes a large task in the design of a plant. Agitated tanks are used within the operating mill essentially as sumps for collection, holding, scrubbing of mineral surfaces, conditioning of reagents, and distribution of slurries as dictated by the logistics and the circuit design. These are typically large cylindrical tanks provided with an impeller or a propeller and baffles. Efficient suspension of solids is an important criterion in the design of a tank (see TANKS AND PRESSURE VESSALS).

Tailings Disposal. In many operations the bulk of the ore becomes tailings (4). In porphyry copper ores for example as much as 90–98% of the ore is discarded as tailings. In other operations tailings could be 20–50% of the ore. Tailings must be treated in an economical and environmentally safe manner. In a majority of operations tailings are collected in a tailings pond (Fig. 32) (7), the design of which constitutes a significant task requiring long-term planning. Tailings disposal is receiving much attention globally, especially in view of the significant increase in demand for and production of minerals, and the serious concerns with management of water and land resources (28). Similarly, mine closure in an economically and environmentally safe manner is receiving much attention.

Tailings are usually thickened before discharging into a pond. This provides rapid water recycle and reduces the volume of tailings transported to the pond. Coarse tailings are used for the construction of a dam which is built up continuously (Fig. 32c). Hydrocyclones are used at the pond site to effect a rough size separation so that coarse stream discharges onto the dam and the fines report to the pond. Several measures are taken to prevent a dam break. Waterproof pads, ie, plastic sheeting and clay are used in some ponds to prevent seepage of contaminated water and potentially hazardous chemicals, such as cyanide from gold leaching plants or flotation chemicals, into the underground water system or to rivers. In some plants around the world tailings are still discharged into rivers, though such practice is rapidly becoming extinct. Coarse solids are also used for enhancing the settling of fine solids in the pond.

In many plants, the main purpose of the tailings pond is for water management and recovery. The requirements today in the majority of plants is to recycle all of the process water, ie, achieve zero discharge. A schematic diagram showing water balance for a typical mineral processing plant is shown in Fig. 33 (26). Another purpose is to collect the solids and use them for backfill of an excavated site or an abandoned underground mine as part of the overall land reclamation plan. Mine overburden is also used for this purpose. An emerging trend is for tailings co-disposal, which involves mixing safe-to-dispose mill tailings with waste rock (or overburden) from the mine. Many techniques are available for mixing tailings with waste rock. In many mines the overburden could be a much larger volume than the ore deposit itself. Tailings disposal in many operations could cost in the range of $0.2 to $1.20 per ton of tailings. Tailings ponds also provide adequate time for biodegradation of potentially hazardous chemicals and for precipitation of heavy metals which, if present in high concentrations in the recycled water, may be detrimental to minerals separations. Settling of solids in a tailings pond is not a problem generally, unless clays and other silicate slimes are involved. Extreme cases of slow settling ponds are those of clay tailings of Florida phosphate operations and the Kimberly diamond operations in Africa.

1.10. Process Control. The most significant developments in minerals processing in the latter 1900s have been in the area of automation and computer control of minerals processing plants (4,7,68–70). Rapid advances in the electronics area are largely responsible for the introduction of on-stream analyzers and sensors (qv). Process control (qv) is an extremely difficult task because of the heterogeneous and complex nature of ores, the extreme variability in any given circuit, and the severity of conditions in the operation. Frequently data needed for process control are available only after the fact. Obtaining representative samples from various flow streams is a prerequisite to obtaining reliable data.

The primary goals in process control are to improve the efficiency and/or selectivity, to reduce the operating cost of each unit operation, and to reduce plant variability. Overall goals are also strongly influenced by economic factors prevailing in the market. Flexibility is therefore a key factor in process control. As for any process control system, the key elements are measurement or sensing, comparison to a target value, manipulation of the variable value, and feedback to the controller. Separate algorithms are developed for each control unit based on empirical factors and experimentation. Continuous improvements and corrections are often made as data are accumulated and as ore characteristics change. Control strategies become more effective when predictions can be made for any unit operation at a high degree of confidence. The more modern control systems are based on multivariable control and model-based concept and digital instrumentation. Present trends are toward total systems using knowledgebase and artificial intelligence (see EXPERT SYSTEMS), which optimize overall performance rather than performance of individual unit operations. These are rule-based systems that attempt to implement human expert knowledge or a rule-of-thumb approach and the uncertainty inherent in human decision making involving linguistic variables (fuzzy terms) and subjective interpretation. Expert system programs or shells are extensively used in plants today.

A development in the 1960s was that of on-line elemental analysis of slurries using X-ray fluorescence. These have become the industry standard. Both in-stream probes and centralized analyzers are available. The latter is used in large-scale operations. The success of the analyzer depends on how representative the sample is and how accurate the calibration standards are. Neutron activation analyzers are also available (6,69). These are especially suitable for light element analysis. On-stream analyzers are used extensively in base metal flotation plants as well as in coal plants for ash analysis. Although elemental analysis provides important data, it does not provide information on mineral composition which is most crucial for all separation processes. Devices that can give mineral composition are under development.

Dry ore tonnage is measured by belt scales mounted along the conveyor belt line. These are continuous weighing devices and take into account the belt load and the belt speed. The output is a flow rate. An integrator in the scale calculates the total tonnage. Slurry flow rates are measured using magnetic or ultrasonic flow meters (Fig. 34) (7). Slurry densities are routinely measured by batch operation by collecting a representative slurry sample in a liter vessel and weighing it on a density scale. Continuous pulp density measurement is made using a nuclear density meter (Fig. 35) (7) or a gamma-gauge which measures the transmission of gamma rays from a radioactive source through the slurry using an ionization chamber-type detector. Transmittance is inversely proportional to the slurry density. Particle size is measured routinely, in a batch operation, by collecting a representative sample and using laboratory standard sieves. Numerous devices are also available for continuous particle size measurement. One of the more common devices uses the attenuation of ultrasonic energy that occurs on transmission through a slurry, eg, the PSM systems for on-line measurement (4) (see ULTRASONICS). For fine particles, laser-based size analyzers, which are light-scattering devices, can be used, eg, the PARTEC system which operates as a scanning laser microscope (70) and the Microtrac system (44). Various other components of plant control and automation include the elaborate alarm systems and shutdown mechanisms for crushers based on bearing pressure and temperature; crusher power and ore level; grinding and classifier controls comprising ore feed rate, water addition rate, classifier feed rate and pulp density, particle size distribution, mill power, and load; flotation controls comprising aeration rate, pulp and froth level; reagents addition rates; and pH and lime addition. Considerable effort is going into improving the performance of existing measurement devices and sensors, and developing new ones. One area of active work is in the development and implementation of reliable redox control systems for sulfide flotation circuits (69). There has been much advance made in Color (or vision) sensors for on-line analysis of flotation froths and slurries, and several plants have already installed such sensors. There is significant research activity in this area. More robust sensors are expected in the near future. (63,68,70).

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Table 1. **History of Minerals Processing and Metallurgy**

• Gold, Copper, Bitumen	Prior to recorded Civilization
• Tin, Bronze, Iron, Soda	Prior to 3500 BC
• Silver, Lead	Prior to 2000 BC
• Bellows in Furnaces	Prior to 1500 BC
• Steel	Prior to 1000 BC
• Base Metals (Hydrometallurgy)	Prior to 500 BC
• Sulfide Ore Smelting	Prior to 500 BC
• Mercury & White Lead	Prior to 300 BC
• Brass (Cementation)	Prior to 100 BC
• Copper Refining (Pyrometallurgy)	Prior to 100 BC
• Stamp Mills, Roasting, Smelting, First Book on Assaying, Co/Mn Pigments, Bi, Sb, Zn, Selective Leaching	Prior to 1500 AD

Ref. 1.

Table 2. **Major Industries Where Minerals are Used**

- Abrasives
- Ceramic Raw Material
- Chemical Industry
- Construction Materials
- Consumer Products
- Electronic and Optical Components
- Electrical
- Fertilizers
- Fillers, Filters and Absorbents
- Fluxes
- Food Industry
- Foundry Sand
- Gem and Jewelry
- Glass
- Medical
- Pharmaceuticals
- Pigments
- Refractories
- Oil Well Drilling Fluids

Ref. 2&3.

Table 3. **Abundance of Metals in the Earth's Crust**[a,b]

Element	Abundance, %	Amount in 3.5 km of crust, t	Element	Abundance, %	Amount in 3.5 km of crust, t
silicon	28.2		chromium	0.010	10^{14}–10^{15}
aluminium	8.2	10^{16}–10^{18}	nickel	0.0075	
iron	5.6		zinc	0.0070	
calcium	4.1		copper	0.0055	10^{13}–10^{14}
sodium	2.4		cobalt	0.0025	
magnesium	2.3	10^{16}–10^{18}	lead	0.0013	
potassium	2.1		uranium	0.00027	
titanium	0.57		tin	0.00020	
manganese	0.095	10^{15}–10^{16}	tungsten	0.00015	10^{11}–10^{13}
barium	0.043		mercury	8×10^{-6}	
strontium	0.038		silver	7×10^{-6}	
rare earths	0.023		gold	$<5 \times 10^{-6}$	
zirconium	0.017	10^{14}–10^{16}	platinum metals	$<5 \times 10^{-6}$	$<10^{11}$
vanadium	0.014				

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[a] Ref. 4. Courtesy of Pergamon Press.

[b] By comparison, oxygen has an abundance of 46.4%.

Table 4. **2003 World and U.S. Production of Selected Mineral Commodities**

Metric Tons **

Mineral or product	World total	United States	US, % of World
Metals:			
Aluminum (Primary)	27,900,000	2,791,900	10.0
Antimony (Mine prod)	145,000	–	–
Arsenic trioxide	35,100	–	–
Bauxite	155,000,000	200,000	0.1
Beryl	3,300	2,100	63.6
Bismuth, refinery	4,630	–	–
Cadmium	16,900	670	4.0
Chromite	16,400,000	–	–
Cobalt (Co content, Mine Output)	48,400	–	–
Columbium (niobium)-tantalum concentrates	80,600	–	–
Copper (Refined Cu)	15,200,000	1,310,000	8.6
Germanium	33	12	36.4
Gold	2,590	277	10.7
Iron ore	1,238,000,000	50,000,000	4.0
Iron and steel:			
Direct-reduced iron	44,100,000	210,000	0.5
Pig iron	647,000,000	40,600,000	6.3
Raw steel	962,000,000	93,700,000	9.7
Ferro-alloys	167,208	–	–
Lead:	6,820,000	1,390,000	20.4
Lithium Minerals (Li content)	14,100	1,500	10.6
Magnesium	508,000	NA	NA
Manganese ore	23,200,000	–	–
Mercury	1,530	15	10.0
Molybdenum, Mo content	125,000	33,500	26.8
Nickel, Ni content:	1,400,000	–	–
Niobium minerals (concentrates)	214,000	–	–
Platinum-group metals	453	18	4.0
Selenium	1,430	NA	NA
Silver	18,700	1,240	6.6
Strontium minerals	539,000		
Tellurium	149	50	33.6
Tin (smelter prod)	279,000	5,420	1.9
Titanium concentrates	6,284,000	500,000	8.5
Tungsten, W content	62,100	–	–
Uranium (mine prod, U3O8 equiv.)	42,800	788	1.8
Vanadium (mine prod)	40,800	–	–
Zinc (mine prod)	9,010,000	738,205	8.2
Industrial minerals:			
Alumina	58,600,000	4,834,000	8.2
Asbestos	2,160,000	–	–
Barite	6,520,000	468,313	7.2
Boron minerals	4,800,000	1,150,000	24.0
Bromine	548,000	216,000	39.3
Celestite	367,000	–	–
Cement, hydraulic	1,950,000,000	94,328,695	4.8

Table 4. (*Continued*)

Metric Tons **

Mineral or product	World total	United States	US, % of World
Clays, Bentonite, Fuller's Earth, Kaolin	55,950,000	15,220,000	27.2
Diamond, natural	30	–	–
Diatomite	1,950,000	620,000	31.7
Feldspar	15,000,000	800,000	5.3
Fluorspar	4,740,000	–	–
Graphite, natural	1,660,000	–	–
Gypsum	102,000,000	16,700,000	16.3
Iodine, crude	24,700	1,750	7.1
Lime	120,000,000	19,200,000	16.0
Magnesite, crude	20,800,000	NA	NA
Mica, including scrap and flake	275,000	78,600	28.6
Nepheline syenite	713,000	–	–
Nitrogen, N content of ammonia	109,000,000	8,770,000	8.0
Peat	26,100,000	634,000	2.4
Perlite	1,630,000	493,278	30.3
Phosphate rock, gross weight	137,000,000	35,000,000	25.6
Potash, K_2O equivalent	28,500,000	1,100,000	3.9
Pumice	14,300,000	870,000	6.1
Rare Earth Minerals (concentrates)	136,000	NA	NA
Salt	210,000,000	43,700,000	20.9
Sand and gravel, industrial, silica	110,000,000	27,500,000	24.9
Sillimanite Minerals	405,000		
Soda ash, natural and manufactured	37,800,000	10,600,000	28.1
Sulfur, all forms	61,800,000	9,600,000	15.5
Talc and pyrophyllite	8,920,000	869,000	9.7
Vermiculite	347,000	100,000	28.8
Wollastonite	220,000	NA	NA
Zirconium minerals	1,155,000	150,000	13.0
Carbon Black	8,000,000	NA	NA
Coal	5,080,000,000	970,247,000	19.1
Coke, metallurgical	392,811,000	21,830,000	5.6
Crude Petroleum	3,596,000,000	285,600,000	7.9
Grand Total, tons	15,324,896,553	1,762,166,331	

NA = Not available; – = Zero.

**Refs. (Minerals Yearbook & BGS; Some are estimates because there is discrepancy in numbers from various sources; numbers in this table are indicated only for the purpose of providing an estimate of the magnitude of Mineral production.

Table 5. **Types of Classifiers**[a]

Equipment	Type[b]	Size, m Width	Size, m Dia	Size, m Length	Feed rate, t/h	Uses
sloping tank classifier (spiral, rake, drag)	M,S	0.3–7	2.4	14	5–850	closed-circuit grinding, washing, dewatering, desliming
log washer	M,S	0.8–2.6	0.6–1.1	4.6–11	40–450	removing trash, clay from sand; break down agglomerates
bowl classifier	M,S	0.5–6	1.2–15	12	5–225	closed-circuit grinding, washing
hydraulic bowl classifier	M,F	1.2–3.7	1.2–4.3	12	5–225	closed-circuit grinding, washing
cylindrical tank classifier	M,S		3–45		5–625	primary dewatering
hydraulic cylindrical tank classifier	M,F		1.0–40		1–150	washing, desliming, closed-circuit grinding
cone classifier	N,S		0.6–3.7		2–100	desliming, primary dewatering
hydraulic cone classifier	M,F		0.6–1.6		10–120	closed-circuit grinding
hydrocyclone	N,S		0.01–12		≤ 20[c]	closed-circuit grinding, desliming degritting, dewatering, washing
air separator	N,S		0.5–7.5		≤ 2100	dry classification[d]
solid bowl centrifuge	M,S		0.3–1.4	1.8	0.04–2.5[c]	fine size separations
countercurrent classifier	M,F		0.5–3.3	12	3–600	clean sands, washing, desliming
eutriator, pocket classifier	N,F				4–120	clean sands, washing, desliming

[a] Ref. 7.

[b] M = mechanical transport of sands to discharge;
N = nonmechanical gravity or pressure; S = sedimentation classifier; and F = fluidized-bed classifier.

[c] Units are m^3/min.

[d] For cement.

Table 6. **Types of Gravity Separators**[a]

Equipment	Size[b], m^2	Capacity[c], t/h	Uses
		Jigging	
diaphragm or plunger mineral jig	≤1.2 × 1.1	4[d]	roughing, cleaning, scavenging relatively coarse cassiterite, gold, scheelite
Baum jig	≤17.6[e,f]	20[d]	mainly for coal washing
Batac jig	30[g]	12–24[d]	mainly for coal washing
Wemco-Remer	1.5 × 4.9	7[d]	primarily for aggregate production
circular jig	41.7 (7.5 dia[e])	10[d]	extensively on tin dredges
pneumatic jig	1.8 × 3.8	2–3[d]	for coals, when dry product is an advantage
		Shaking	
shaking table	2.0 × 4.6	0.05–0.25[d]	coal, cassiterite, scheelite, and other heavy minerals
Holmans slimes table	2.0 × 4.6	0.01–0.06[d]	for particles too fine for conventional table; also for cleaning concentrate from Bartles-Mozley
Bartles-Mozley concentrator	1.2 × 1.5	2.5	rougher concentrator for very fine heavy minerals
Bartles cross-belt vanner	2.75 × 2.4	0.5	similar applications to slimes table
		Flowing	
Humphrey's spiral	0.6[e] dia, 2.9[h]	1–5	beach sands, iron ore, and other heavy minerals
pinched sluice	≤1.8 × 0.4	2–4	beach sands, phosphate ore
Reichert cone	2 dia[e]	60–90	ubeach sands, coal, iron ore, trace heavy minerals from tailings

[a] Ref. 7.

[b] Dimensions are width by length unless otherwise noted.

[c] Capacities depend mainly on type of minerals treated and their particle size range.

[d] Value given is per square meter of jig or table.

[e] Value is in meters.

[f] Arrangement of 2 × 6 cells in parallel.

[g] Arrangement of six 5 × 1 cells.

[h] Height of spiral in meters.

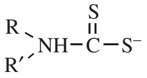
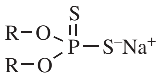
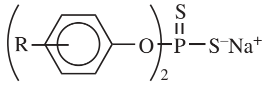
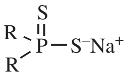
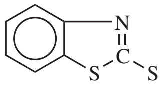
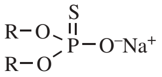
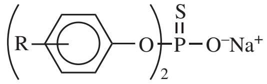
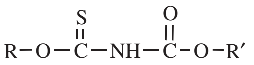
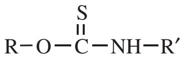
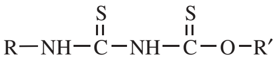
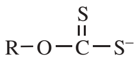
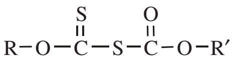
Table 7. **Types of Magnetic Separators**[a]

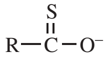
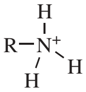
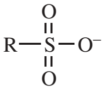
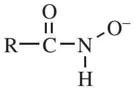
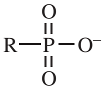
Equipment	Feed rate, m^3/(min·m)	Field strength at 5 cm, T[b]	Size, mm		Application
			Dia	Width	
			Tramp removal and cobbing		
suspended magnets				≤1010	removal of tramp iron in crushers and other process equipment, and iron from foundry sands; recovery of iron from slags
plate and grate magnets			[c]		tramp iron removal
magnetic pulley	≤ 1500[d]		910[e]		similar to suspended magnets
cobbing drum			300–916		coarse magnetic cobbing, tramp iron removal, iron recovery from slags, cleaning of scrap
			Wet low intensity		
concurrent	50–350	0.06–0.7	760–1200	1525–3050	high grade magnetite from coarse ore; taconites; dense medium recovery
counter-rotation	50–250	0.5–0.6 m/s	760–1200	1525–3050	retreating tailings from a concurrent drum such as dense medium recovery; taconites
counter-current	20–250	0.05–0.06	760–1200	1525–3050	finishing separator; good recovery and clean concentrate
			Dry low intensity		
high speed drum	0.05–0.45	0.04–0.05	400–916	300–3000	magnetites; dry grinding circuits
Ball-Norton type	0.13–0.5		≤760	1525	coarse magnetite in dry separation
			Wet high intensity		
carousal type	0.01–1[f]	<2			paramagnetic minerals; hematite and chromite; iron from China clay and other minerals; pyrite from coal
canister type	0.07–0.3[f]	2	2130		primarily iron from China clay; other uses similar to carousal type
			Dry high intensity		
induced roll	0.01–0.1	<2.1	250–1000		dry paramagnetic materials, beach sands, wolframite, monazite, cassiterite, cleaning silica sand and feldspar

cross-belt	0.02	2			450	similar use to induced roll, but limited to high value minerals; extremely selective separation; simultaneously separate many minerals with a range of susceptibilities

[a] Ref. 7.
[b] To convert tesla to gauss, multiply by 1×10^{-4}.
[c] Size of grate = 1 m^2.
[d] Value is in units of m^3/h.
[e] Maximum belt speed is 150 m/min.
[f] Value is in units of t/min.

Table 8. **Collectors Used in Sulfide and Non-Sulfide Mineral Flotation**

Sulfide Collector Families		
1	Dithiocarbamate, Dialkyl	 $\begin{matrix}R\\R'\end{matrix}NH-\overset{\overset{S}{\parallel}}{C}-S^{-}$
2	Dithiphosphate, Dialkyl	 $\begin{matrix}R-O\\R-O\end{matrix}\overset{\overset{S}{\parallel}}{P}-S^{-}Na^{+}$
3	Dithiophosphate, Diaryl	
4	Dithiophosphinate, Dialkyl	 $\begin{matrix}R\\R\end{matrix}\overset{\overset{S}{\parallel}}{P}-S^{-}Na^{+}$
5	Mercaptan, Alkyl	R–SH
6	Mercaptobenzothiazole	
7	Monothiophosphate, Dialkyl	 $\begin{matrix}R-O\\R-O\end{matrix}\overset{\overset{S}{\parallel}}{P}-O^{-}Na^{+}$
8	Monothiophosphate, Diaryl	
9	Thionocarbamate, Alcoxycarbonyl Alkyl	 $R-O-\overset{\overset{S}{\parallel}}{C}-NH-\overset{\overset{O}{\parallel}}{C}-O-R'$
10	Thionocarbamate, Allyl Alkyl	 $R-O-\overset{\overset{S}{\parallel}}{C}-\overset{H}{N}-CH_2-CH=CH_2$
11	Thionocarbamate, Dialkyl	 $R-O-\overset{\overset{S}{\parallel}}{C}-NH-R'$
12	Thiourea, Alcoxycarbonyl Alkyl	 $R-NH-\overset{\overset{S}{\parallel}}{C}-NH-\overset{\overset{S}{\parallel}}{C}-O-R'$
13	Xanthate Ester, Allyl Alkyl	 $R-O-\overset{\overset{S}{\parallel}}{C}-S-CH_2-CH=CH_2$
14	Xanthate, Alkyl	 $R-O-\overset{\overset{S}{\parallel}}{C}-S^{-}$
15	Xanthogen Formate, Dialkyl	 $R-O-\overset{\overset{S}{\parallel}}{C}-S-\overset{\overset{O}{\parallel}}{C}-O-R'$
Non-Sulfide Collector Families		

1	Fatty Acids	 $R-\overset{\overset{\displaystyle S}{\parallel}}{C}-O^-$
2	Primary Amines	 $R-\overset{\overset{\displaystyle H}{\vert}}{\underset{\underset{\displaystyle H}{/}}{N^+}}\diagdown H$
3	Petroleum Sulfonates	 $R-\overset{\overset{\displaystyle O}{\parallel}}{\underset{\underset{\displaystyle O}{\parallel}}{S}}-O^-$
4	Alkyl Hydroxamates	 $R-\overset{\overset{\displaystyle O}{\parallel}}{C}-\underset{\underset{\displaystyle H}{\vert}}{N}\diagup O^-$
5	Alkyl Phosphonates	 $R-\overset{\overset{\displaystyle O}{\parallel}}{\underset{\underset{\displaystyle O}{\parallel}}{P}}-O^-$

Refs. 56,57.

Table 9. **Modifiers Used in Sulfide and Non-Suifide Mineral Flotation**

Ammonium Sulfide
Ca, Mg, Al and Fe Salts
CMC
Copper Sulfate
Dextrine
Guar gum, modified guars
Hydrofluoric Acid
Lignin Sulfonates
Lime
Mercaptoethanol
Nokes reagent, Arsenic Nokes
Organic dyes
Phosphoric Acid
Polyacrylates
Soda Ash
Sodium and Zinc Cyanide
Sodium Ferrocyanide
Sodium Fluoride
Sodium Hypochlorite
Sodium meta and polyphosphates
Sodium metabisulfite
Sodium Silicate
Sodium sulfide and hydrosulfide
Sodium sulfite
Sodium thioglycolate and its trithiocarbonate derivative
Starch
Sulfur dioxide
Sulfuric Acid
Surfactants
Synthetic Functionalized Polyacrylamides
Tannins (Quebracho)
Zinc Sulfate

Ref. 56.

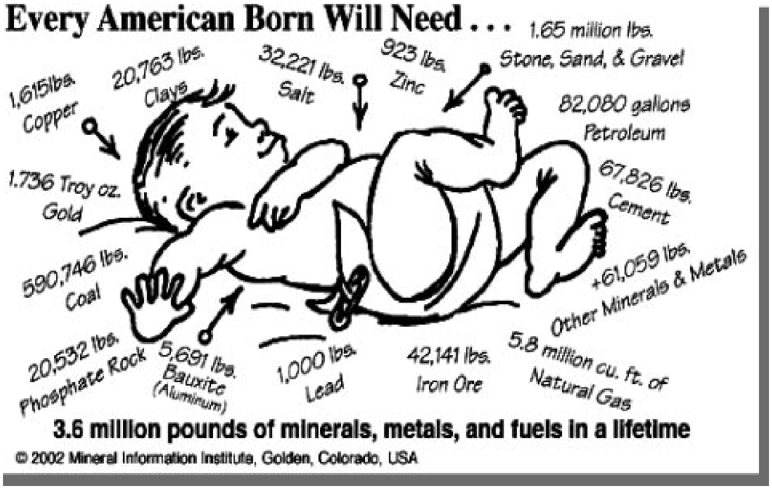


Fig. 1. Mineral Consumption by individuals in the U.S. (Ref. 2).

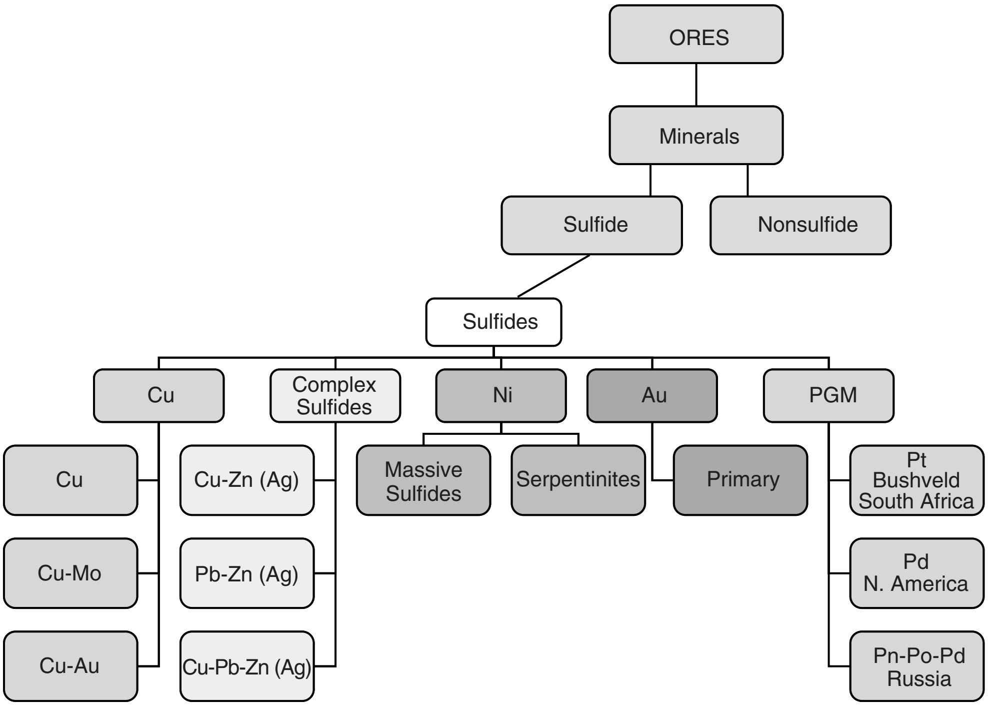


Fig. 2. A broad classification of ores based on separation technology and end use.

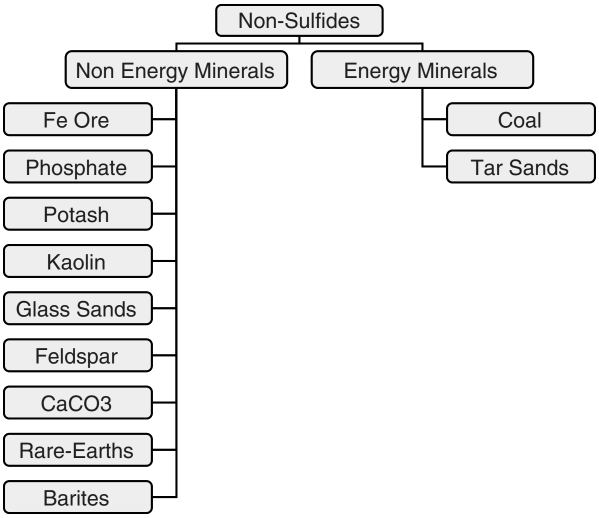


Fig. 2. (*Continued*)

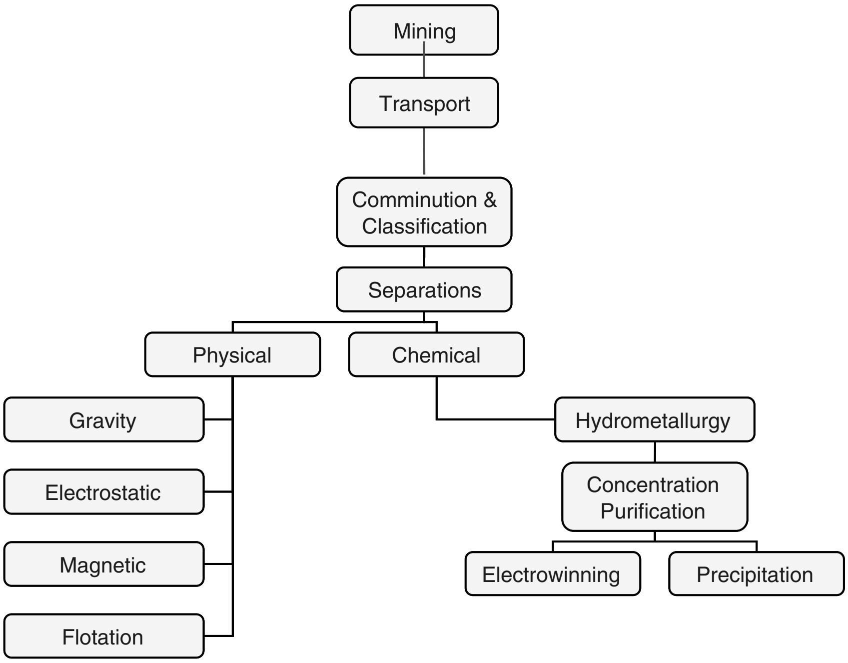


Fig. 3. Generalized Mineral processing Flowsheet.

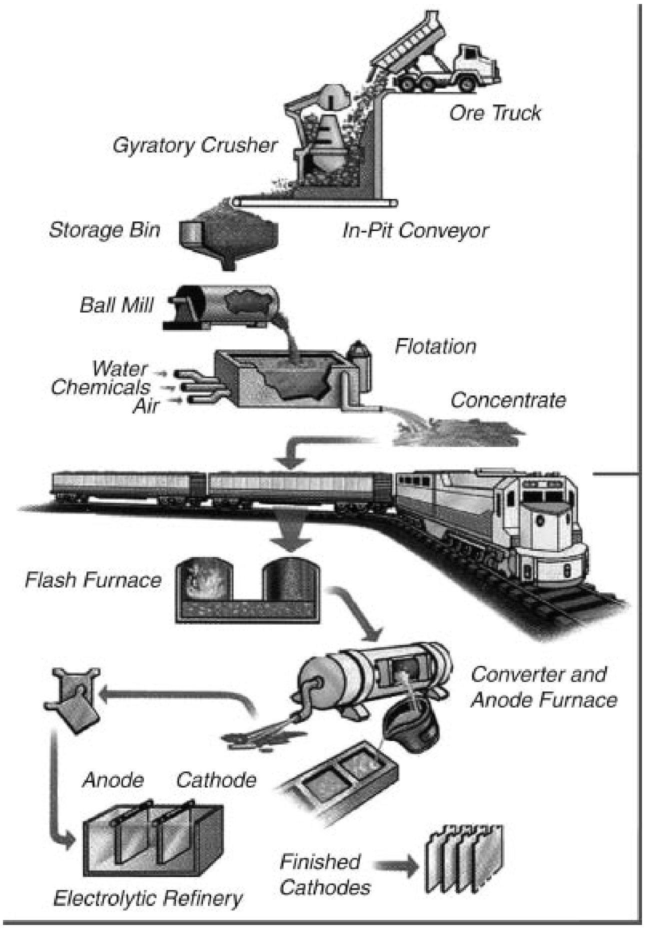
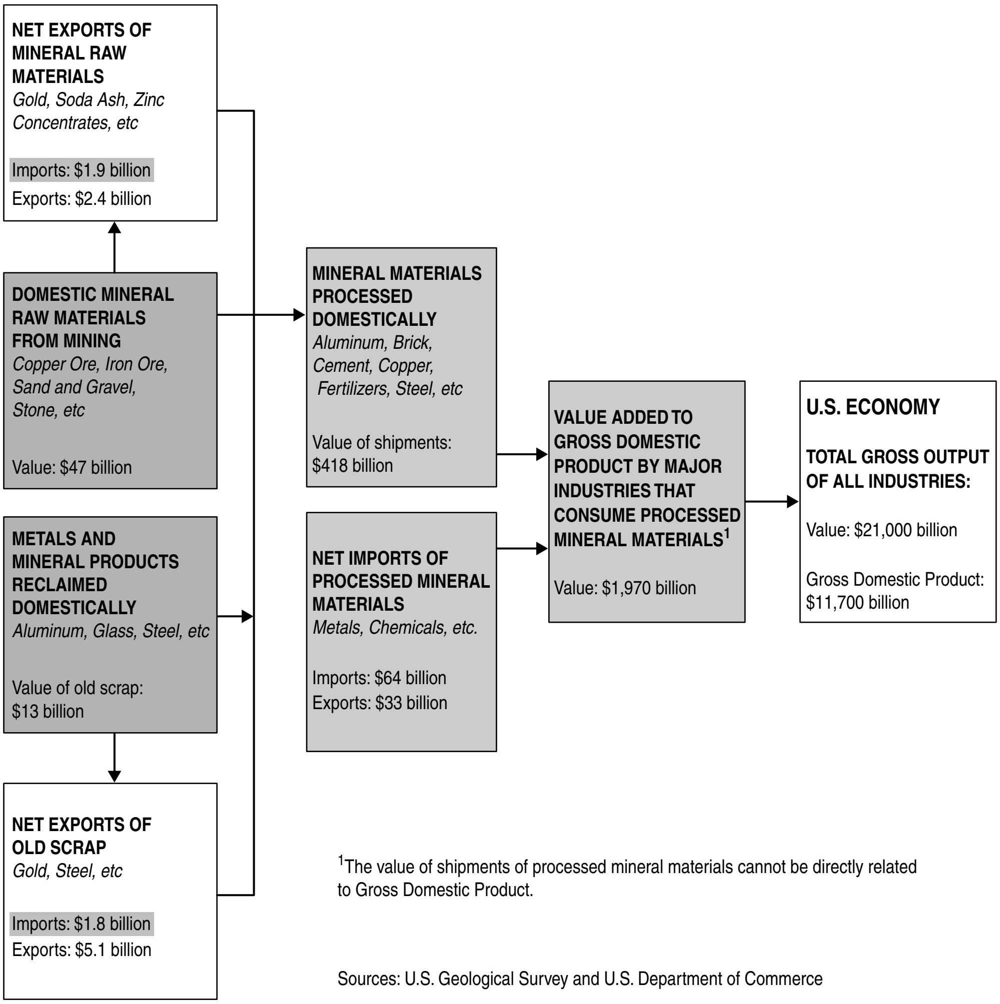


Fig. 4. A schematic flow sheet for a copper ore operation, from mined ore to metal.



[1] The value of shipments of processed mineral materials cannot be directly related to Gross Domestic Product.

Sources: U.S. Geological Survey and U.S. Department of Commerce

Fig. 5. The Role of Nonfuel Minerals in the U.S. Economy in 2003 (From U.S. Geological Survey and U.S. Department of Commerce, Ref. 23).

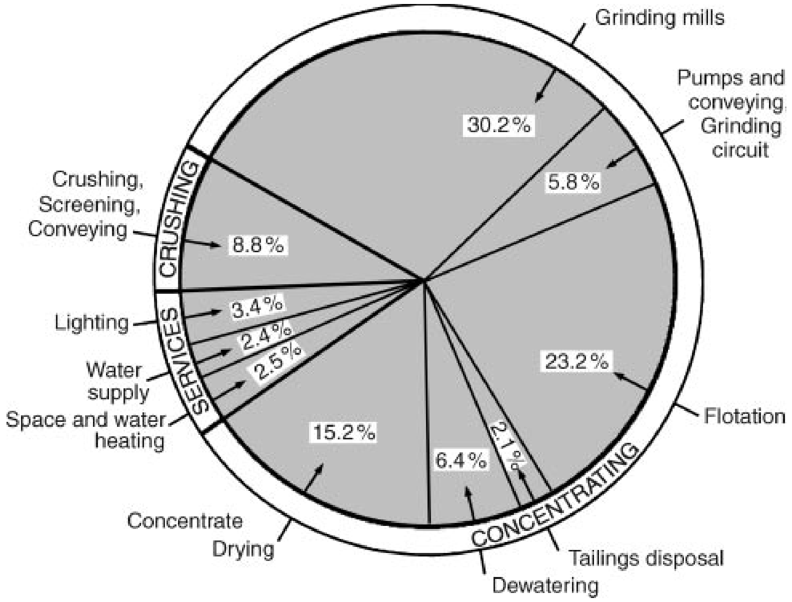


Fig. 6. An example of the distribution of power consumption in a separation plant, such as flotation (Ref. 26).

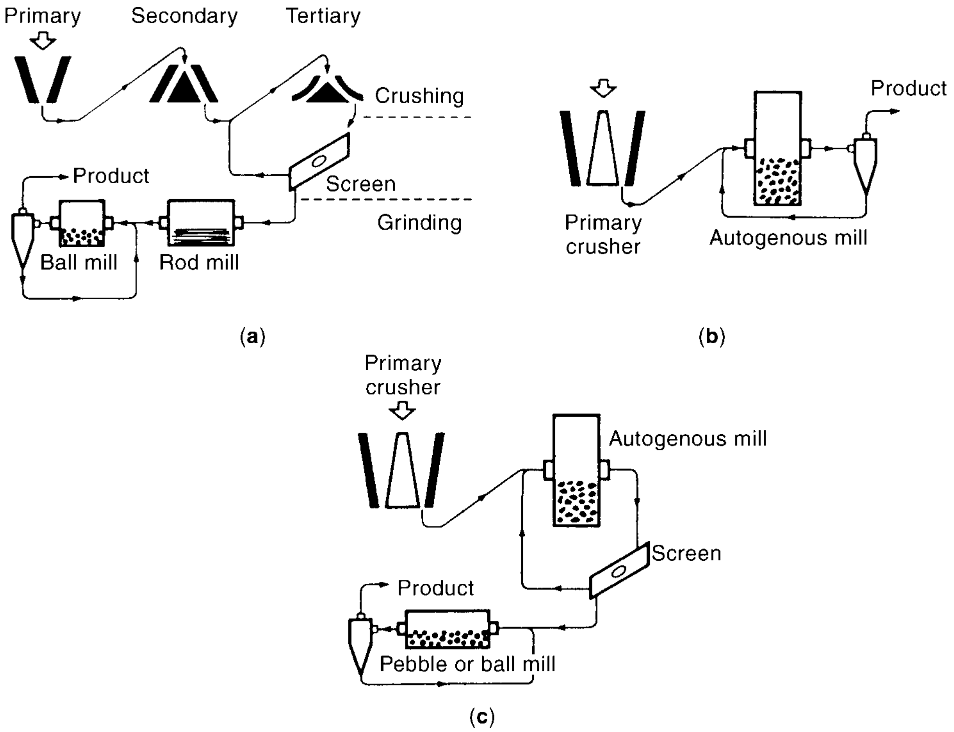


Fig. 7. Three basic types of size reduction circuits: (**a**) conventional, (**b**) autogenous, and (**c**) autogenous plus separate fine grinding (7).

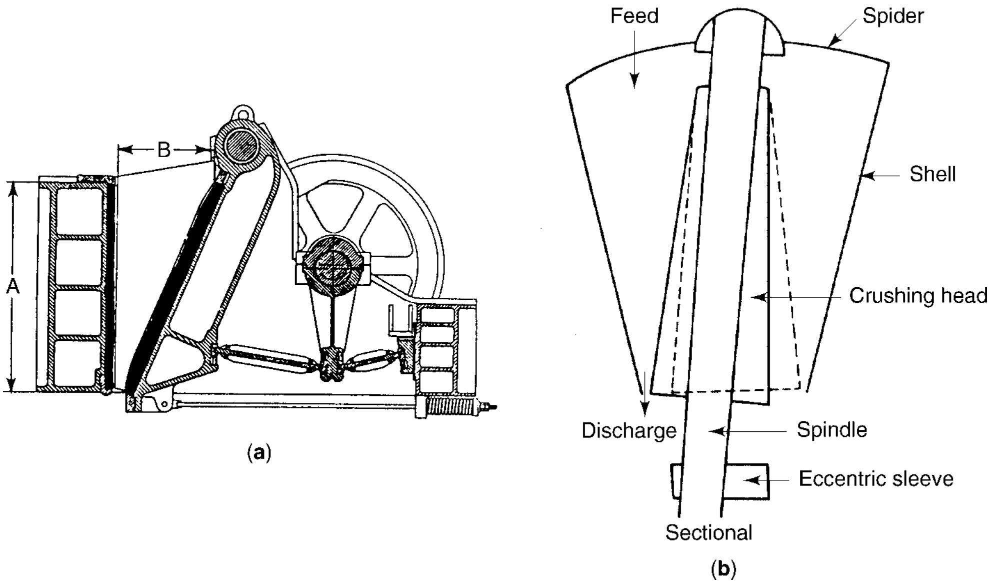


Fig. 8. (**a**) Cross section of a double-toggle jaw crusher where A and B represent dimensions (13). (**b**) Functional diagram of a gyratory crusher (4).

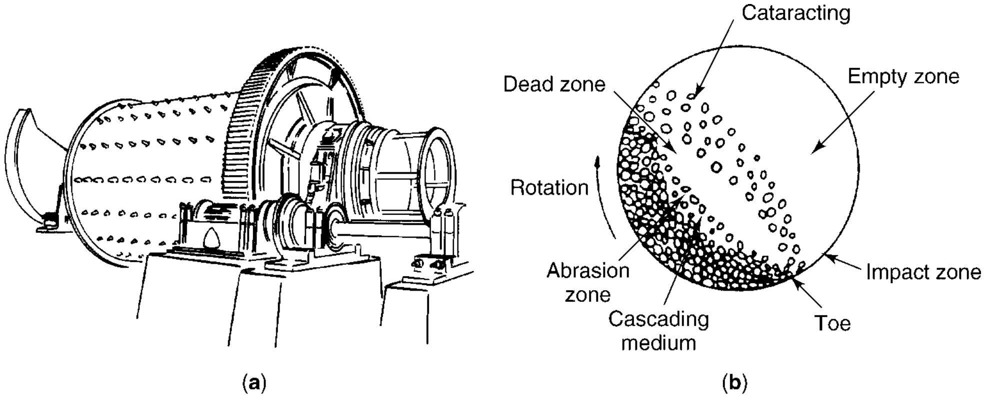


Fig. 9. (**a**) Tumbling mill; (**b**) motion of charge within it (4).

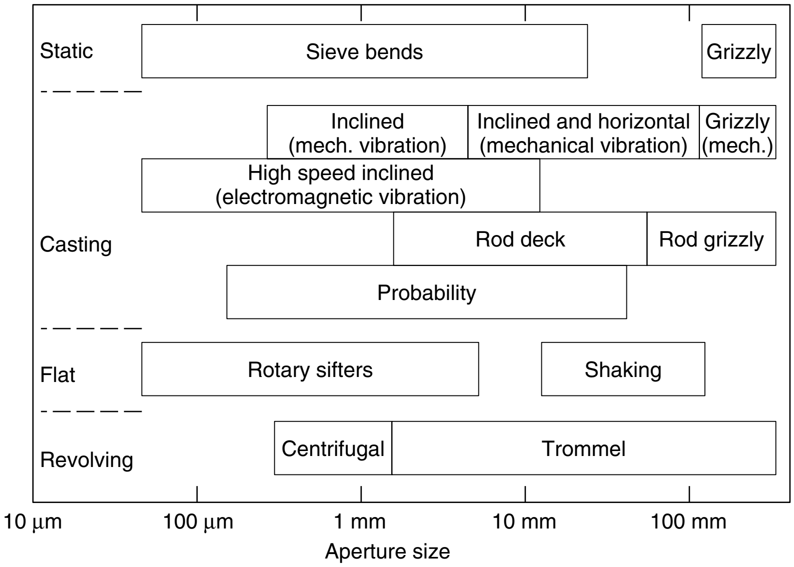


Fig. 10. Typical separation sizes of the basic screen types (7).

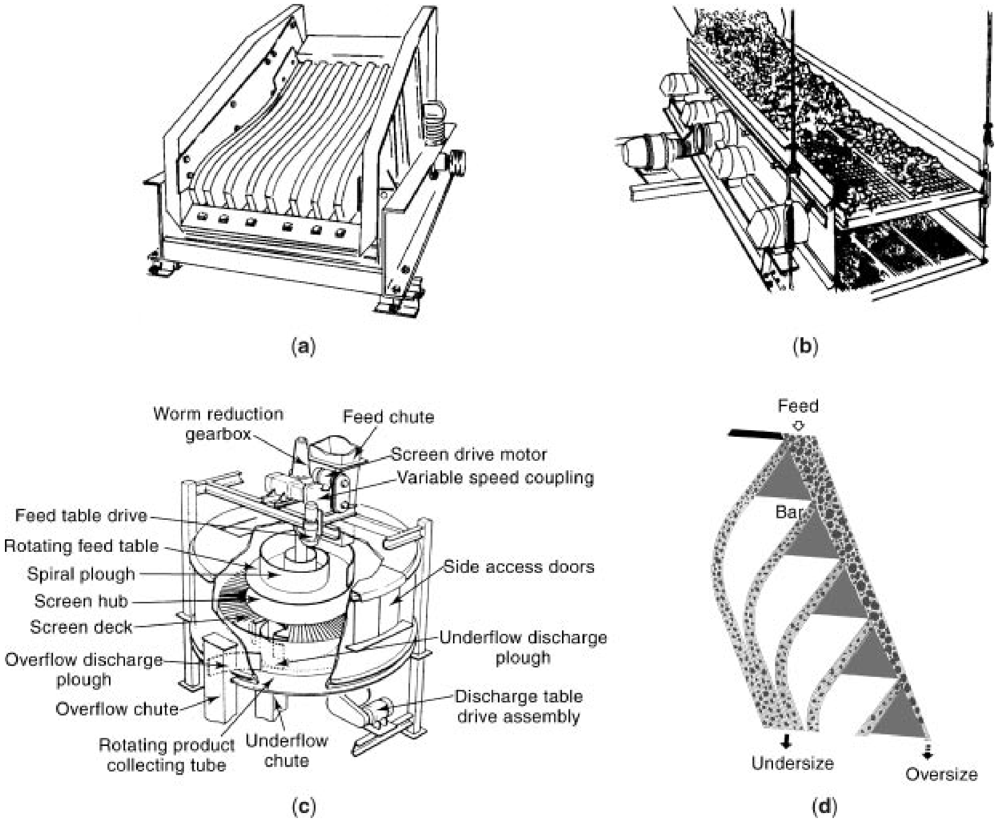


Fig. 11. Schematic diagrams of screens: (**a**) vibrating grizzly, (**b**) double-deck vibrating screen, and (**c**) a rotating probability screen (4). A schematic illustration of the operating principle of a sieve bend is shown in (**d**) (7).

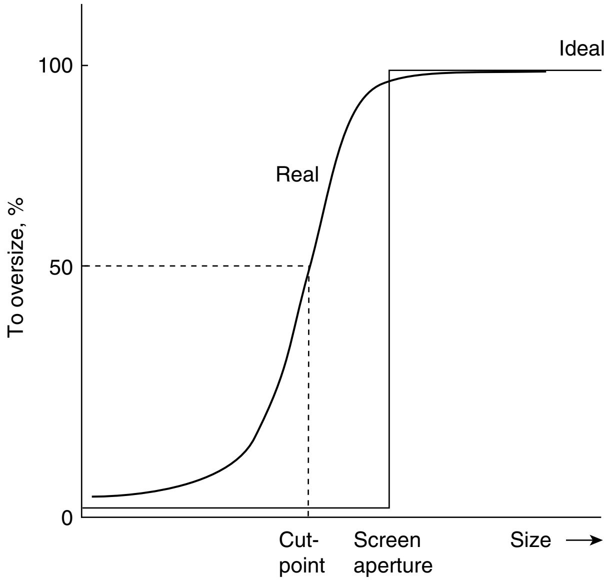


Fig. 12. Partition or Performance Curve for Screens and Classifiers (Ref. 4).

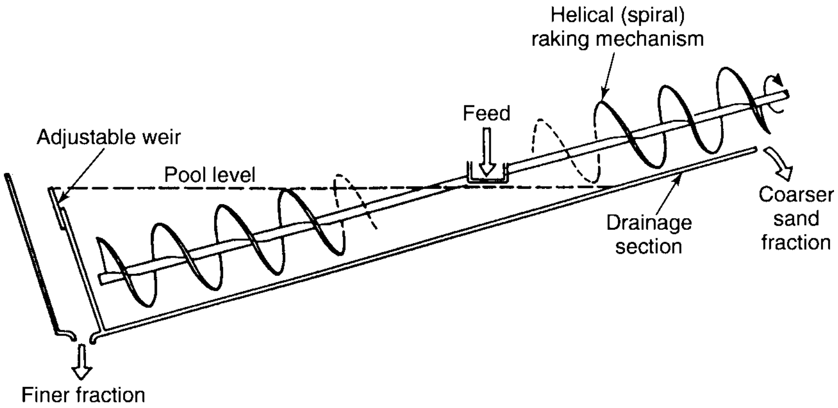


Fig. 13. Schematic of a mechanical classifier with submerged spiral rake where (–) is the pool level and slope = 1:4 to 1:3 (37).

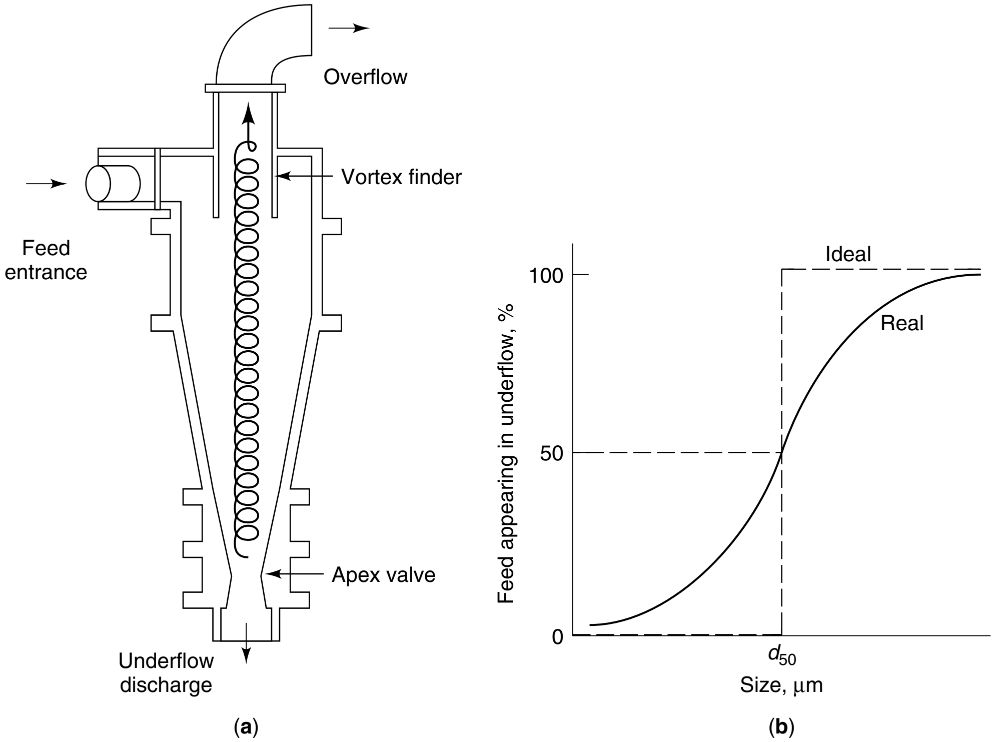


Fig. 14. (**a**) Schematic of a hydrocyclone; (**b**) its partition curve (4).

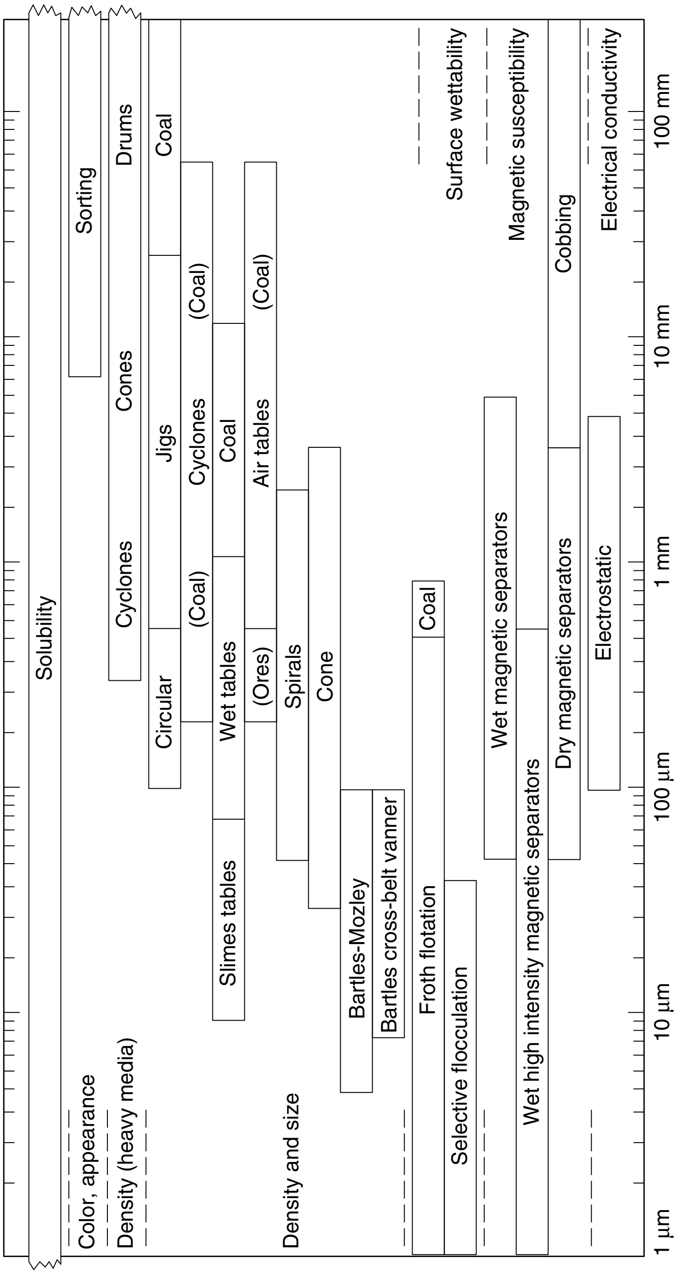


Fig. 15. Particle size ranges for concentrating equipment based on mineral properties (7).

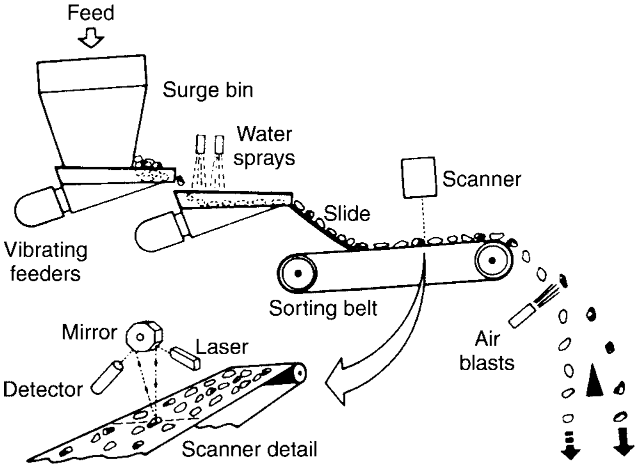


Fig. 16. Primary features of a sorting machine (7).

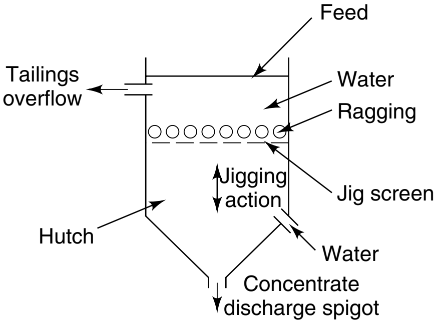


Fig. 17. Basic jig construction (4).

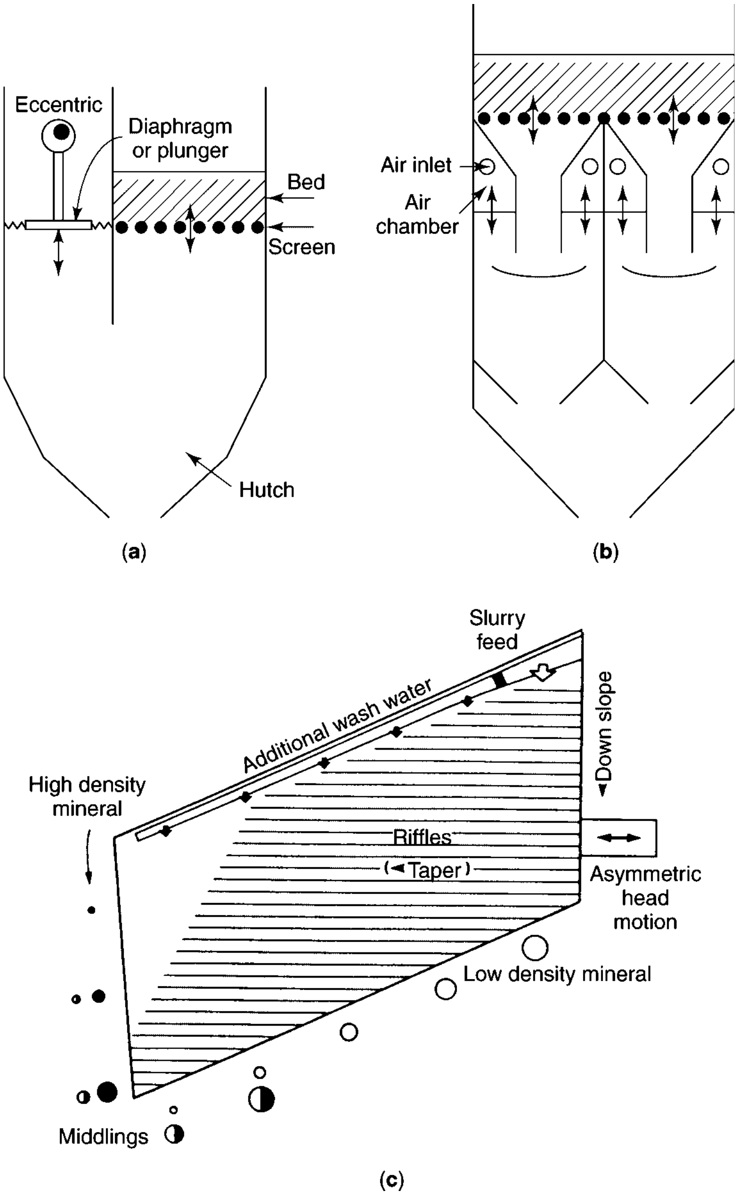
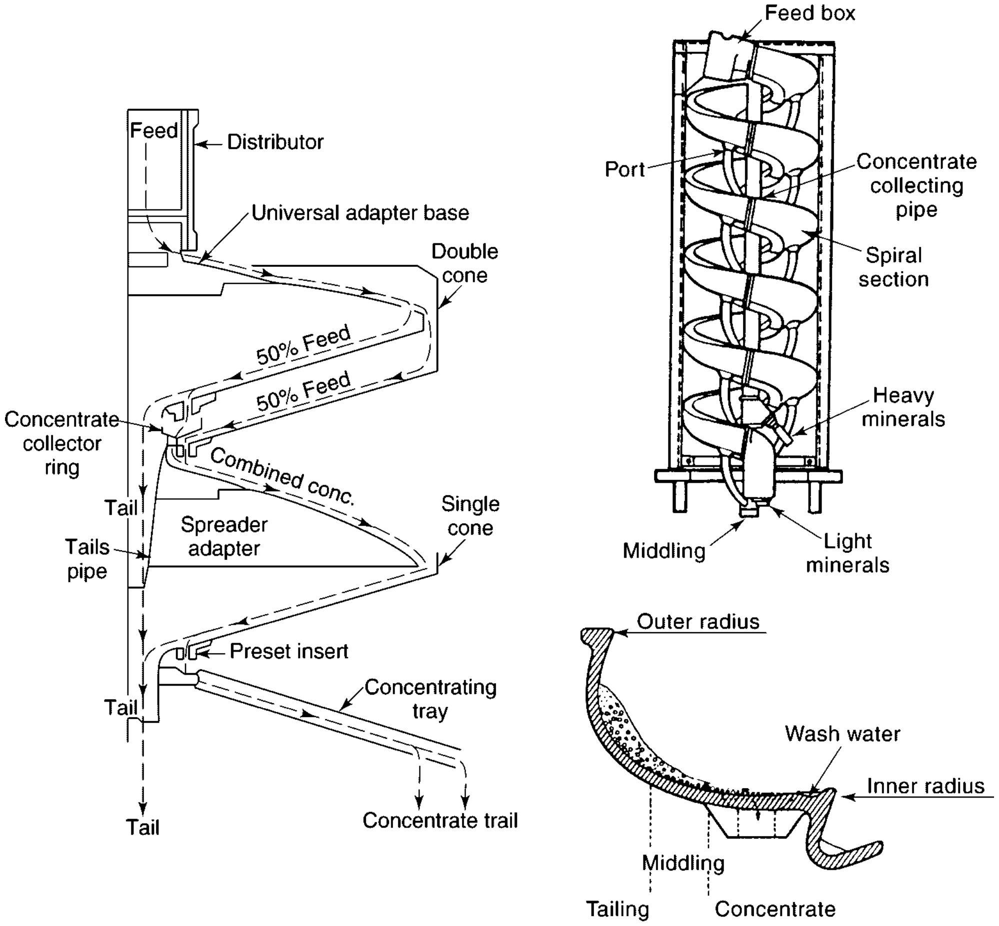


Fig. 18. Schematic diagrams of the basic types of jig where ↔ represents the jigging action: (**a**) Denver/Harz and (**b**) Batac. (**c**) Schmatic of a shaking table, showing the distribution of products (7).



(a) (b)

Fig. 19. Gravity flow concentrators: (**a**) Reichert cone (2), and (**b**) spiral, where (○) represent particles of low density and (●), particles of high density (37).

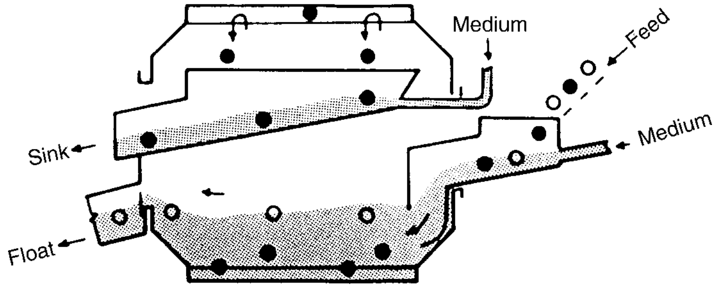


Fig. 20. Gravity dense medium drum separator where (○) is the float and (●), the sink (4).

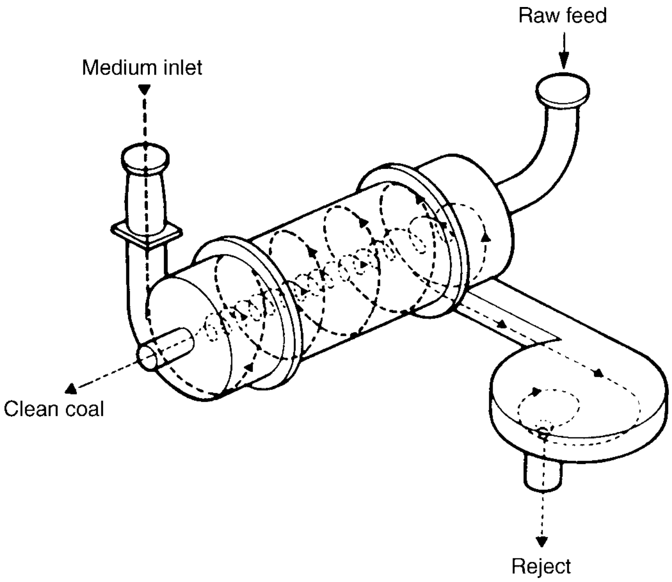


Fig. 21. LARCODEMS separator (4).

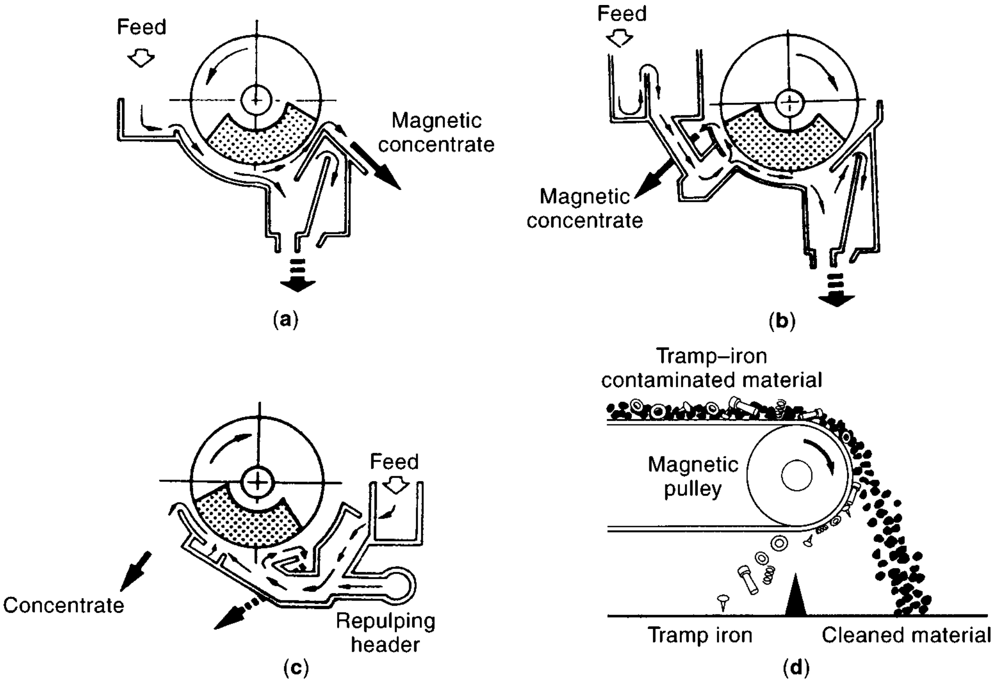


Fig. 22. Wet drum low intensity magnetic separator tank designs: (**a**) concurrent, (**b**) counter-rotation, and (**c**) countercurrent (7); (**d**) shows the operating principle of a magnetic pulley. Courtesy of Eriez Magnetics.

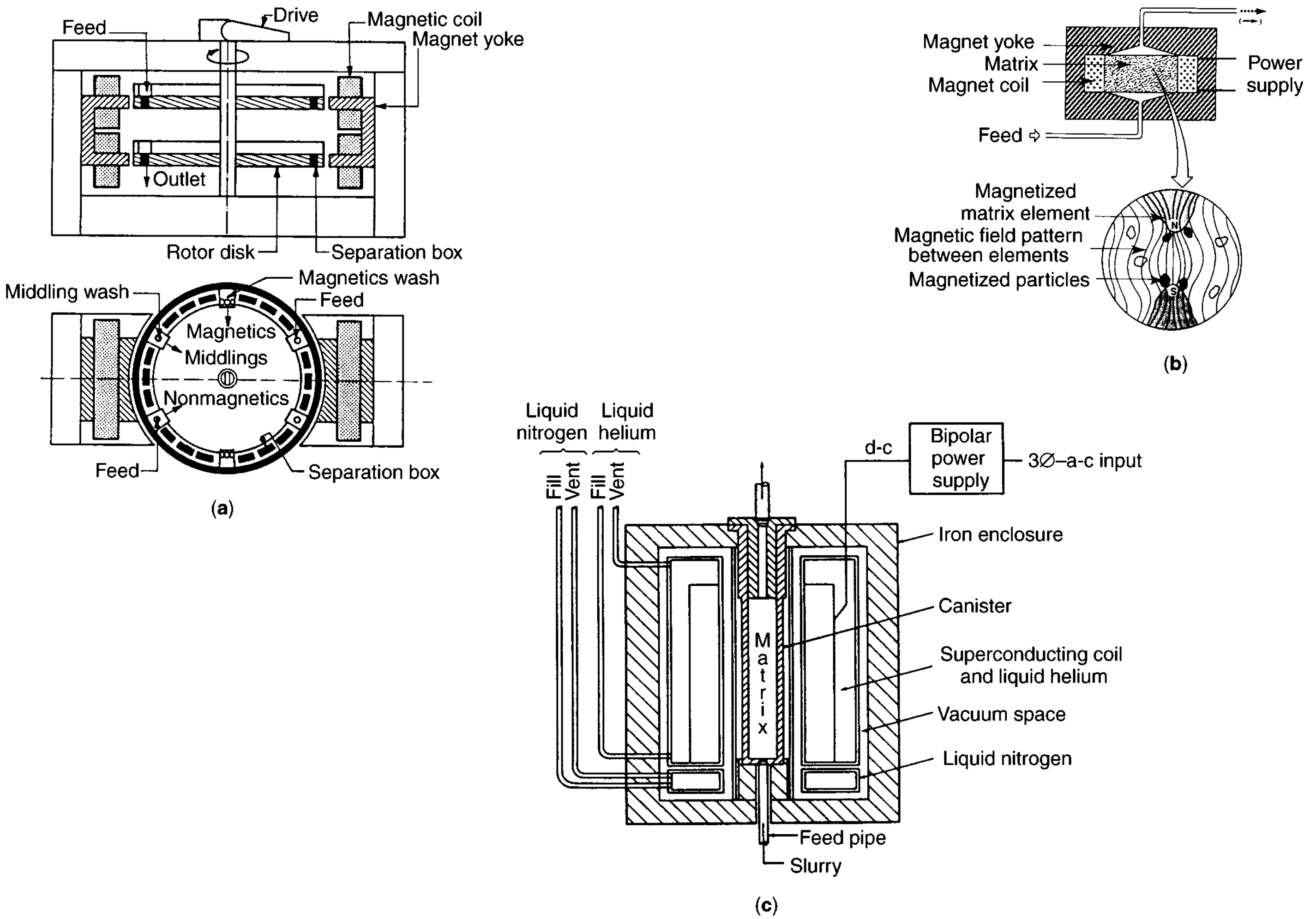


Fig. 23. Operating components of various separators: (a) Jones continuous high intensity wet magnetic separator (13), (b) canister-type high intensity magnetic (7), and (c) superconducting magnetic separator (4).

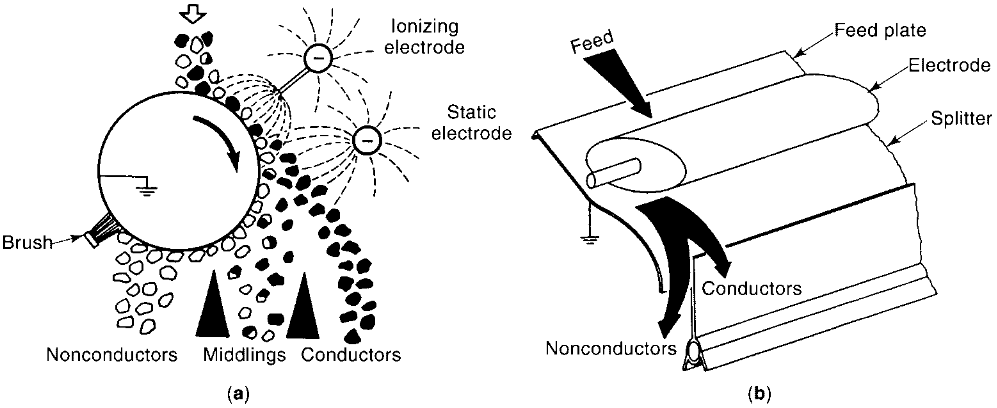


Fig. 24. Electroseparators: (**a**) high tension separator, and (**b**) plate-type electrostatic (7).

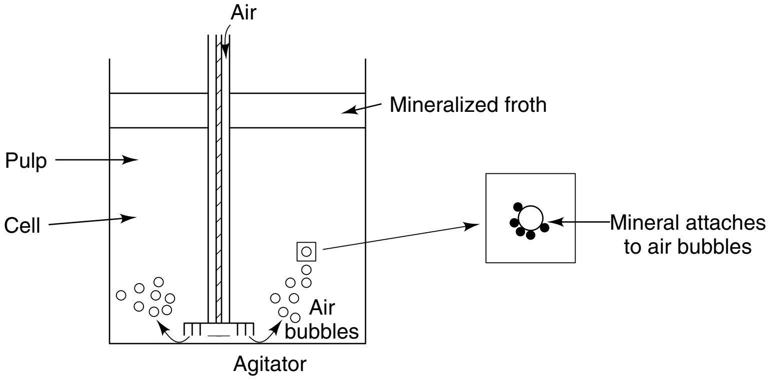


Fig. 25. Principles of froth flotation (4).

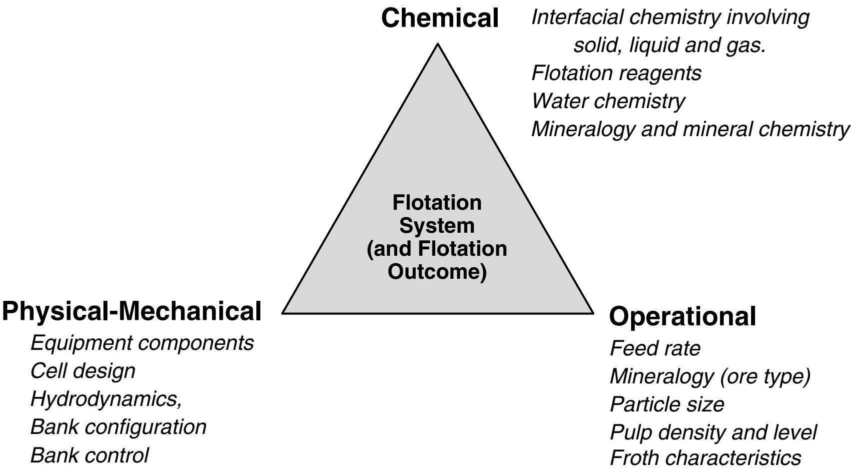


Fig. 26. A schematic of the Complex Flotation System. Flotation outcome is determined by complex interactions between the chemical, physical-mechanical and operational factors. The triangle implies a trade-off between the three factors, and the optimum in performance can be anywhere in the triangle depending upon the conditions in the plant. These factors (especially the chemical and operational) change routinely in the plant, and the process demands continuous control to achieve optimum outcome (58).

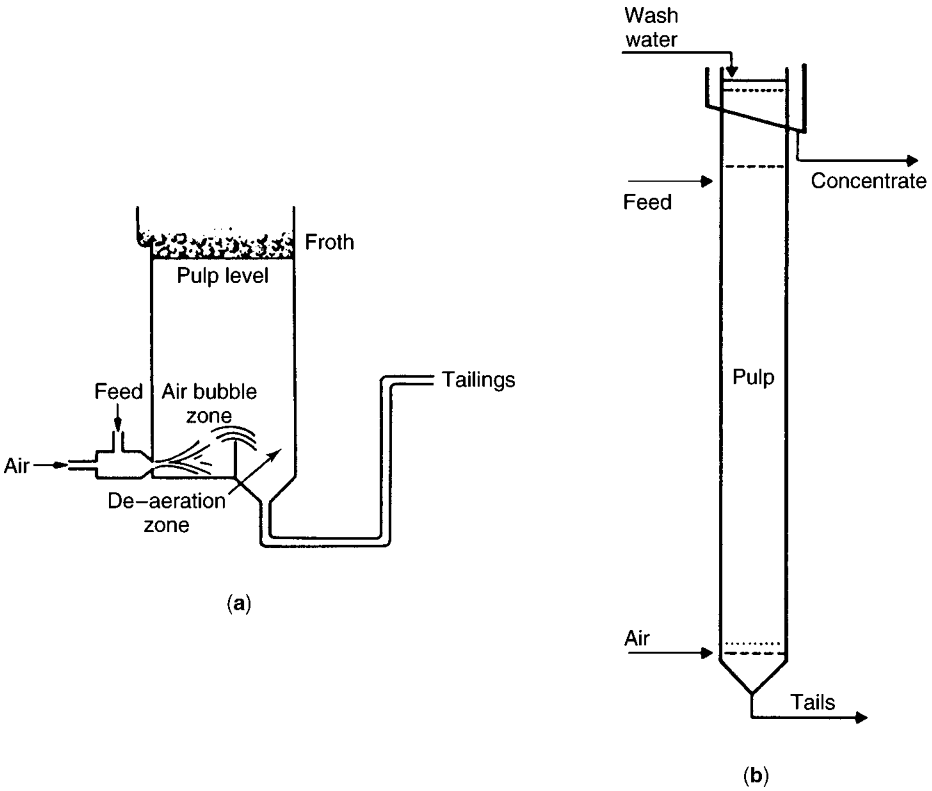


Fig. 27. (**a**) Davcra pneumatic flotation cell, and (**b**) schematic diagram of a flotation column cell (4).

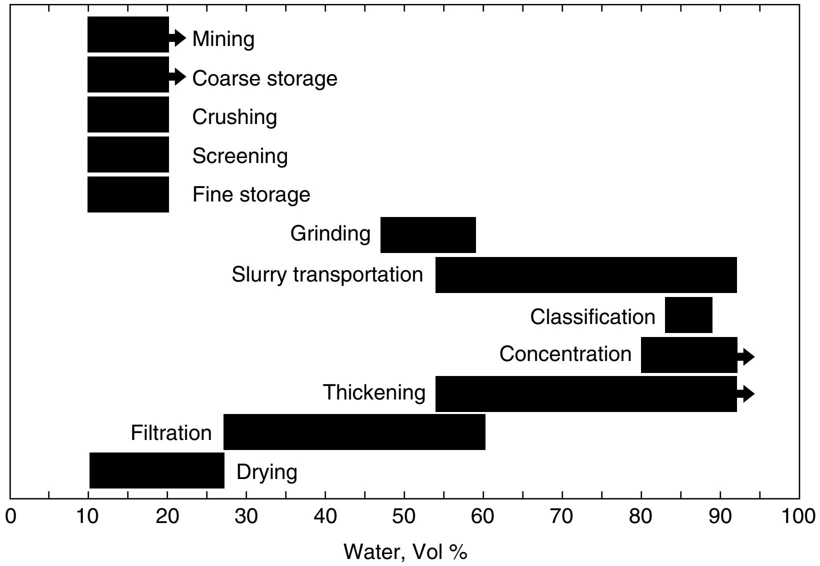


Fig. 28. Volume % water in various unit operations (Ref. 26).

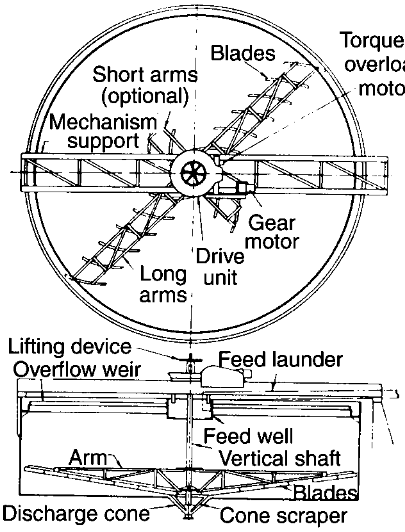


Fig. 29. Cylindrical thickener with mechanism supported by superstructure (4).

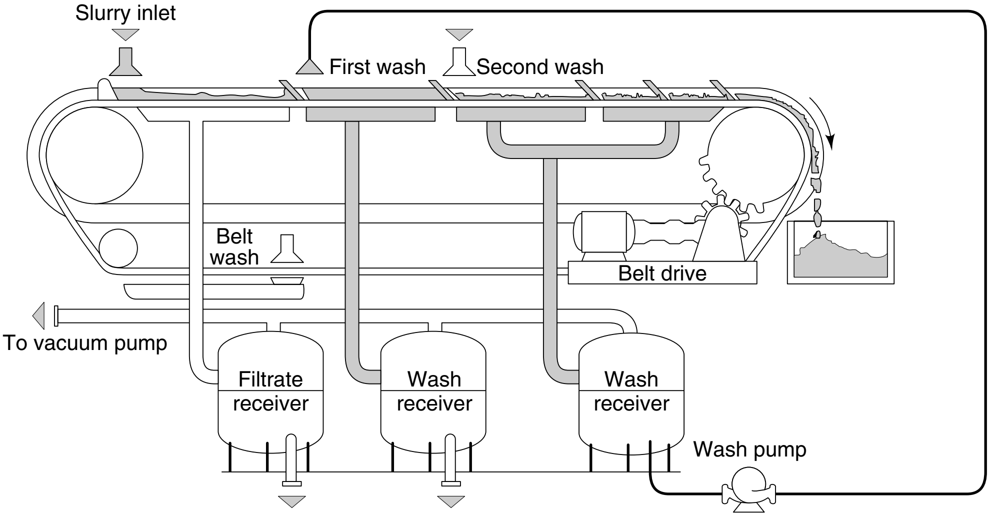


Fig. 30. Schematic of horizontal belt filter (4).

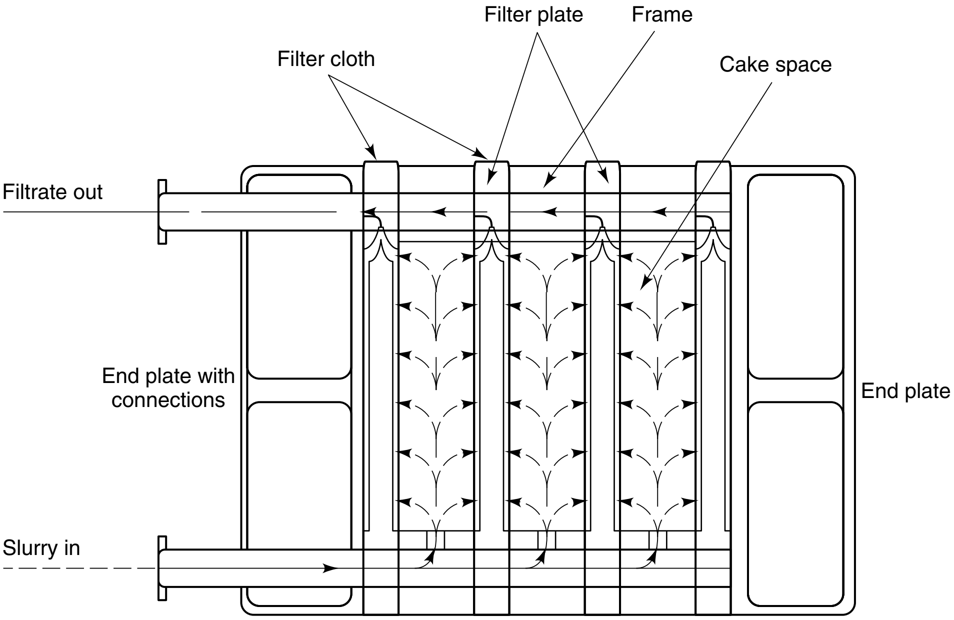


Fig. 31. Plate and frame filter press (4).

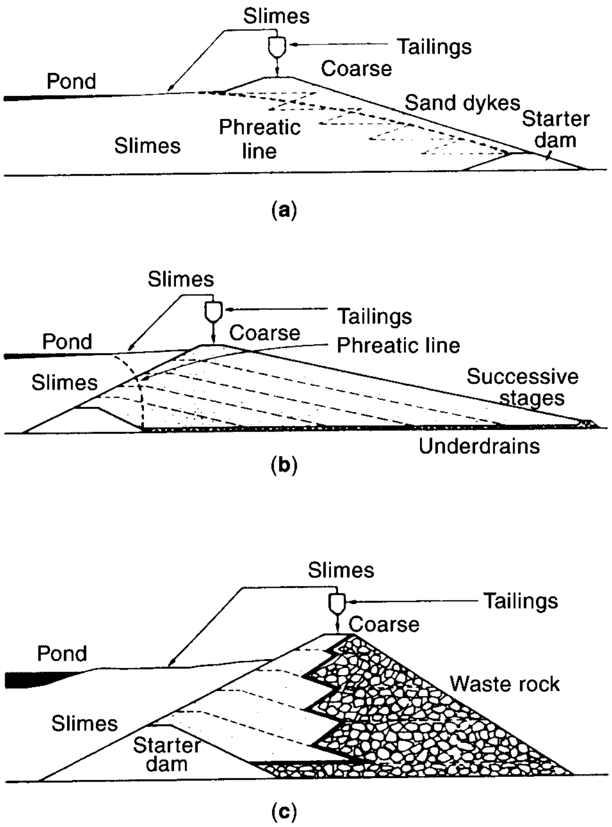


Fig. 32. Methods of tailings dam construction: (**a**) upstream method, (**b**) downstream method, and (**c**) mine waste rock dam construction (7).

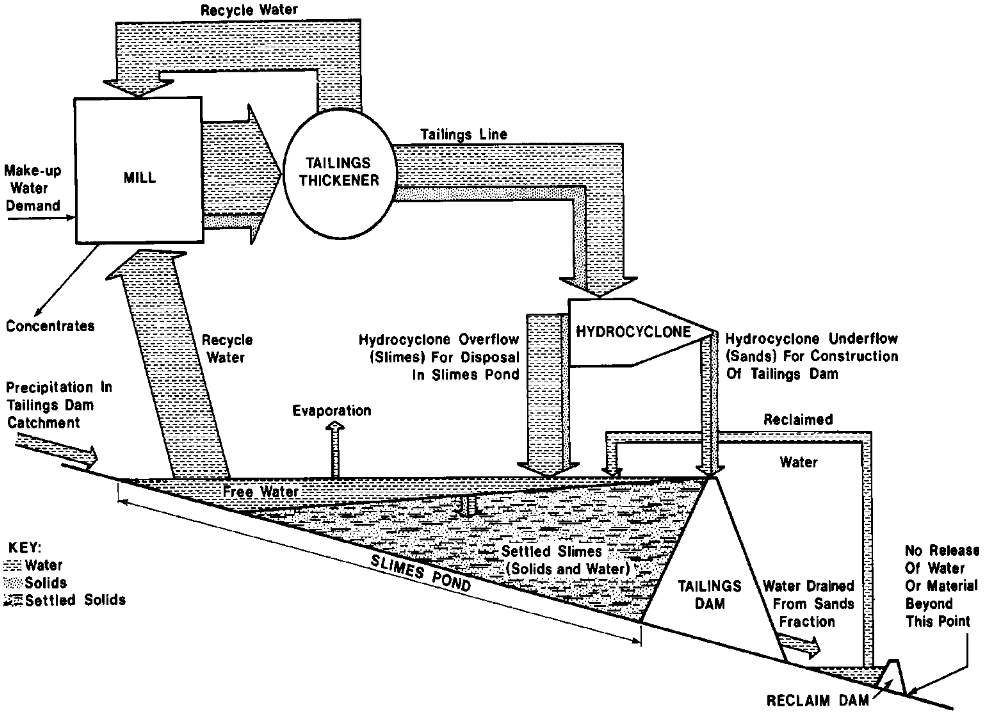


Fig. 33. A Schematic diagram showing water balance for a typical mineral processing plant (Ref. 26).

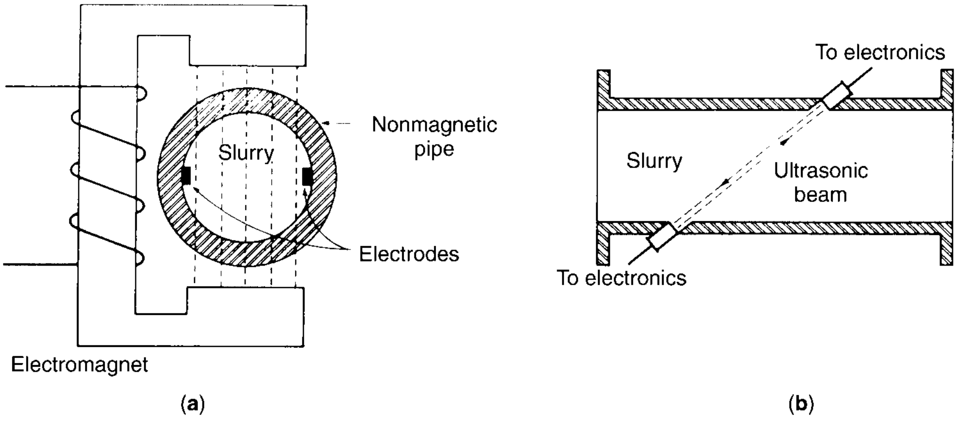


Fig. 34. Flow meters for on-line measurement of slurry flow rates: (**a**) magnetic and (**b**) ultrasonic (7).

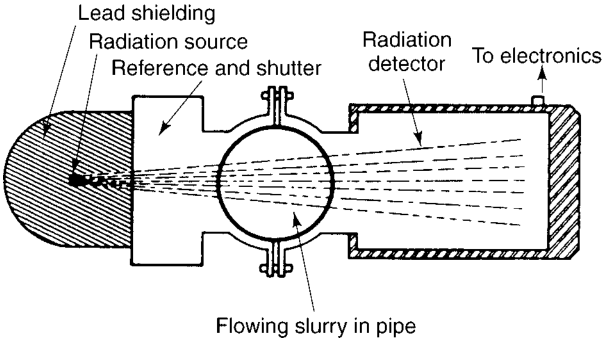


Fig. 35. Nuclear density meter for on-line measurement of slurry density (7).