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Liquid-Solid Separation

Donald A. Dahlstrom

INTRODUCTION

Water is used in most steps in the beneficiation of minerals and coal, and water is used to process most approximately 80%–90%—of the tonnage of minerals and coal. (A small and decreasing percentage of coal is crushed and screened dry, and industrial minerals such as diatomaceous earth and bentonite can be processed dry.) Beneficiation processes usually use water because it allows greater efficiency, higher recovery, and lower cost per unit of valuable product. In addition, it eliminates air pollution.

Costs of Liquid–Solid Separation

The use of water then necessitates that solids be separated from the liquid. In general, as the particles to be separated decrease in size, cost increases and capacity per unit area decreases. When the solids are colloids (generally considered to be $-10 \mu m$ or less), costs increase even faster. They are difficult to remove by filtration or centrifugation. Usually, a flocculant is added to the mixture to cause the colloids to form larger flocculi or agglomerates; otherwise the colloids remain in suspension because of Brownian movement. Accordingly, liquid–solid separation is a major cost in mineral processing, probably exceeded only by the cost of comminution, flotation, and endothermic reactions. For example, the capital cost of a coal preparation plant increases by about 30%–40% if the –28 mesh coal is processed instead of discarded, and the process water is recovered (by liquid–solid separation) for recycle and reuse. Operating costs per ton of –28 mesh coal also increase substantially as compared with coarser coals. These costs are due primarily to the use of flotation and the liquid–solid separation steps involved.

At the same time, liquid-solid separation by mechanical means (i.e., sedimentation, filtration, and centrifugation) is much less costly than thermal drying, primarily because those means consume less energy. Furthermore, thermal drying usually requires higher-cost fuels, such as gas or oil, whereas mechanical methods can use electrical energy generated by lower-cost fuels.

To illustrate some highly efficient liquid-solid separations, consider this example. A 100-ftdiameter conventional gravitational thickener (at a normal design rate for tailings concentration and water reclamation of 3 ft² per short ton of dry solids per day) will process more than 2,600 short tons of solids per day. With a feed of 15 wt% solids, an underflow concentration of 50 wt% solids or higher can usually be achieved if the solids contain 50%–55% particles that are –200 mesh or coarser. This size consist means that 4.67 lb of water per pound of solids has been eliminated and that more than 82% of the water will report to the thickener overflow for reuse. The thickener drive head will be equipped with a only 5- or $7\frac{1}{2}$ -hp motor. The only other energy-consuming process, pumping, is merely an incremental cost as compared with the alternative—normally a large tailing pond at some distance from the plant.

In the processing of magnetite concentrates derived from the beneficiation of taconite, disk filters are used to dewater magnetite concentrates before the balling step. For an 1,800 cm²/g Blaine concentrate (approximately 85%–90% at 325 mesh), a filtration rate of 230 lb of dry solids/h/ft² is used as a

308 | PRINCIPLES OF MINERAL PROCESSING

design basis (Wolf et al. 1971). The feed concentration would be maintained at 65 wt%, and the vacuum level should be at 24 in. of mercury by using a vacuum pump capacity of about 6 cfm/ft² of filtration area. Power consumption will be very high because of the high vacuum and flow rate. Considering also the filter drive, compressed air requirements, filtrate pump, and agitator, a total of 36 hp would be required for 100 ft² of filtration area. This power requirement is equivalent to 91,620 Btu/100 ft² of filtration area. However, 23,000 lb of dry magnetite solids per hour are dewatered to the 9 wt% moisture required for hauling. At the same time, 10,120 lb of water are extracted. Thus, only 9.05 Btu are required per pound of water eliminated.

Thermal drying, on the other hand, requires around 1,800 Btu per pound of water evaporated. Obviously, mechanical methods of dewatering have relatively low energy consumption per unit of liquid removed. As energy costs continue to increase, other mechanical methods will be developed to further decrease energy consumption.

Liquid-solid separation is also critical in water reclamation and closed water circuits (Wolf et al. 1971). Water that has been used for processing and beneficiating minerals and coals will contain solids that can range in size from a fraction of a micrometer to ¼ in. or more. Some streams will contain the valuable solids and others the refuse or tailings. In both cases, the solids must be separated out if the water is to be reused. Furthermore, the concentration of suspended solids in recycled water must be low enough so that the water does not contaminate the next product. In the case of iron ore processing, the return water must contain 100–150 mg/L or less to minimize the percentage of silica in the final pellet. Coal requires a suspended solids concentration of less than 1 wt%.

This brief discussion shows why liquid–solid separation is a prerequisite for a closed water circuit. It will also be required before any effluent is disposed of in lakes, streams, or other public water sources. State and federal regulations generally require that such effluents contain no more than 10–50 mg/L of suspended solids.

Steps in Liquid–Solid Separation

Separating liquid and solids requires many steps (Hassett 1969; Kynch 1952; Hitzrot and Meisel 1985). In a coal-washing plant, clean coal coarser than $\frac{1}{2}-1$ in. will normally be dewatered by screening, probably the simplest and lowest-cost method of liquid-solid separation. Dewatering increasingly finer particles may require centrifugation, filtration, expression (mechanically squeezing water from feed slurries), or batch filters operating at pressures of 250 psig (Sandy and Matoney 1979).

Medium-size clean coal ($\frac{1}{2}$ in. to +28 mesh) is dewatered by centrifuges and sometimes screens (Anon. 1963). The -28 mesh clean coal is dewatered either by continuous filters (normally disk type) or solid bowl scroll discharge centrifuges. Gravitational thickeners are used to concentrate the refuse tailings and reclaim water for reuse. Refuse tailings are dewatered by disk or drum filters, belt presses, or pressure filters.

In minerals beneficiation, ore is usually ground much finer to achieve liberation and produce both desired grade and recovery (Henderson et al. 1957; Kobler and Dahlstrom 1979). Thus, base metals and iron ore concentrates, large-tonnage minerals, will usually be rated according to the percentage of -325 mesh or even -400 mesh solids. Because of their abrasive character, these and other minerals will be dewatered on filters. Both continuous filters and centrifuges are used with crystalline solids such as potash or sodium sulphate. Gravity thickeners are used for both concentrate and tailings (Coe and Clevenger 1916; Roberts 1949; Terchick et al. 1975). The latter are generally sent to tailings ponds as their tonnage and volume can be very large. However, more stringent regulation of the construction of tailings ponds ensures that mechanical dewatering methods will be increasingly used in the future (Chironis 1976).

Hydrometallurgical processing always creates abundant colloidal solids during the leach step. Proper liquid–solid separation enables recovering the maximum amount of pregnant liquor while minimizing its dilution. Thus, multistage countercurrent sedimentation, countercurrent washing filtration on a single filter, or two- or three-stage filtration with single washes per stage are practiced. The mineral or metal is commonly precipitated from solution and washed to maximize purity. Thus, both continuous vacuum filters and pressure filters are involved (Kobler and Dahlstrom 1979; Nelson and Dahlstrom 1957; Osborne 1975).

Because of today's emphasis on water reuse, by-products of beneficiation—dissolved salts and metals, pH, suspended solids, and temperature—must be controlled. If effluent is disposed of in a tailings pond, that effluent must be satisfactory for reuse or disposal to a watercourse, and it must not contaminate ground water or aquifers if it migrates out of the pond. If solids are disposed of in a landfill, the discharged solids must be low in moisture and must not rewet with rain. In either case, tailings ponds and landfills must be carefully monitored to ensure their stability and thus to avoid ground water pollution.

These paragraphs very briefly discuss only the major liquid-solid separation steps. However, it should be apparent that liquid-solid separation is critical to efficient and low-cost processing and probably substantially affects capital and operating costs.

MAJOR INFLUENCES ON LIQUID-SOLID SEPARATION

Many factors external to the liquid–solid separation equipment itself influence its performance and productivity. The most common of these follow (Cross 1963; Dahlstrom 1978; Weber 1977; Hsia and Reinmiller 1977; Robins 1964; Wetzel 1974; Sakiadis 1984; Rushton 1978; Bosley 1974; Scott 1970; Silverblatt et al. 1974):

- 1. Particle size and shape
- 2. Weight and volume percentage of solids
- 3. Fluid viscosity and temperature
- 4. pH and chemical composition of the feed
- 5. Variation and range in feed quality (items 1-4)
- 6. Specific gravity of solids and liquid
- **7.** Quality requirements of discharge streams from liquid-solid separation steps, particularly as they influence results upstream and downstream

Particle Size and Shape

Size distribution greatly affects liquid-solid separation rates. Stokes' law can be used to illustrate this fact. This law permits the terminal settling velocity (maximum velocity achieved during free fall) to be determined as follows:

$$v_t = \frac{(\rho_s - \rho)gD^2}{18\mu}$$
 (Eq. 9.1)

where

 v_t = terminal velocity, ft/s

- ρ_s = particle density, lb/ft³
- ρ = liquid density, lb/ft³
- g = gravitational acceleration, (ft/s²) × (lb mass/lb force) = 32.17, at sea level at 45° latitude. This value is used for g, the standard gravitational acceleration on this planet, and is usually symbolized by gc.
- D = particle diameter, ft
- μ = fluid viscosity, lb/ft × s

Viscosity is normally measured in centipoises. One centipoise = 6.72×10^{-4} lb/ft-s. (All nomenclature has been given in English units but metric [Système International] units can be used as long as they are used consistently.) The equation has some limitations. It assumes laminar flow (i.e., a Reynolds number of less than 0.1) and nonhindered settling. In nonhindered settling, a particle is not influenced by the presence of other particles or by the slurry's specific gravity, conditions that require very dilute slurries.

If we assume a 20- μ m particle and an 80- μ m particle of the same density and in the same fluid, from Eq. 9.1 the terminal velocity is found to be directly proportional to the square of the diameter. In this case, the 80- μ m particle will fall 16 times as fast as the 20- μ m particle.

Fine particles, particularly under laminar flow, achieve terminal settling velocity almost immediately. This rapidity is caused by the drag force, which resists settling and quickly equals the buoyancy force; the drag force acts on particles through the difference in specific gravity between liquid and solid. This behavior is generally true of most particles that are settled or otherwise classified in the mineral industry. The drag force is given by Eq. 9.2:

$$F_d = C A_p \rho v^2 / 2g_c$$
 (Eq. 9.2)

where

 F_d = drag force, lb force

C = drag coefficient, dimensionless

- A_p = projected area of particle in direction of motion, ft²
- ρ = density of liquid, lb/ft³
- v = velocity of particle, ft/s
- g_c = conversion factor = 32.17

The buoyancy force is given by

$$F_p = \frac{m_s}{\rho_s} \left(\rho_s - \rho \frac{g}{g_c} \right)$$
 (Eq. 9.3)

where

 F_p = buoyancy force, lb force

- m_s = mass of particle, lb mass
- ρ_s = density of solid, lb/ft³
- ρ = density of liquid, lb/ft³
- g = acceleration resulting from gravity ft/s²

At terminal velocity, drag force equals the buoyancy force or

$$CA_p \rho v^2/2g_c$$
 (Eq. 9.4)

Solving for v_t and for a spherical particle,

$$v_t = \left[\frac{4gd(\rho_s - \rho)}{3\rho c}\right]^{1/2}$$
(Eq. 9.5)

Figure 9.1 is a plot of $C = (F_d/A_p)/(\rho v^2/2g_c)$ (from Eq. 9.2) as a function of the Reynolds number. The Reynolds number is dimensionless and for spheres, disks, and cylinders, it equals $\rho v d/\mu$. Laminar flow exists for the constant slope value up to Re = 0.1. The actual value of the coefficient *C* is 24/Re for laminar flow. Substituting this value of *C* in Eq. 9.5 will yield Stokes' law. Thus, Eq. 9.5 and Figure 9.1 can be used to solve for v_t , knowing d, ρ_s , ρ , and μ , although it becomes a trial-and-error solution.

The trial-and-error solution can be eliminated by using of the terms CN_{Re}^2 and C/N_{Re} . This substitution works because v_t is not in CN_{Re}^2 and d is not in C/N_{Re} . Thus, the following equations are written:

$$CN_{\rm Re}^2 = 4g\rho d^3(\rho_s - \rho)/3\mu^2$$
 (Eq. 9.6)

$$C/N_{\rm Re} = \frac{4g\mu(\rho_s - \rho)/3m^2}{3\rho^2 v_t^3}$$
(Eq. 9.7)