Coal Preparation

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Coal preparation involves the upgrading of run-of-mine (ROM) coals using low-cost solid–solid and solid–liquid separation processes. The processes remove undesirable impurities such as waste rock and water from carbonaceous material. For coal-fired power stations, these inorganic impurities reduce coal heating value, leave behind an undesirable ash residue, contribute to particulate and gaseous emissions, and increase transportation costs (Doherty 2006). The presence of impurities can also influence the suitability of coal for high-end uses such as the manufacture of metallurgical coke or the generation of petrochemicals and synthetic fuels. Unwanted surface moisture, which increases transportation costs and decreases heating value, can also create handling and freezing issues for industrial consumers. Consequently, all coal supply agreements impose strict limitations on the purity levels of purchased coal.

The methods and processes used to upgrade coal have changed dramatically during the past century (Carris 2007). In the early 1900s, manual sorting methods such as handpicking were used by the industry for quality improvement. This labor-intensive approach was soon made obsolete by mechanized processes that reduced misplacement and increased productivity. Many of these early facilities cleaned only the coarser particles and either recombined untreated fines back into the coal product or discarded the fines as waste. These historic periods were followed by many decades of technology development that allowed operators to partially or completely upgrade the entire size range of mined coals (Osborne 2012). Today's modern coal processing facilities, some of which handle thousands of short tons per hour (stph) of ROM coal, are as functionally complex as their counterparts in either the mineral beneficiation or chemical processing industries. This level of design sophistication is dictated by increasingly competitive markets that require plants to operate at very high levels of processing efficiency to remain profitable.

PROCESS FLOW SHEET

The push for enhanced efficiency has forced plant designers to develop flow sheets that incorporate several parallel processing circuits, often three or more, which are specifically configured to optimize the upgrading of each size fraction. The multiple circuits are needed to maintain efficiency since the processes employed in coal processing each have a limited range of applicability in terms of particle size. For example, Figure 1 provides a simplified flow sheet for a modern plant that includes four independent circuits for treating coarse (>10 mm), intermediate (10–1 mm), fine (1–0.15 mm) and ultrafine (<0.15 mm) coal. In the United States, the coarse and intermediate size fractions are typically treated using dense medium vessels and dense medium cyclones (DMCs), respectively. In some cases, the dense medium vessel can be eliminated by reducing the top size of the feed coal. The fine coal fraction, which cannot be handled by dense medium processes because of practical constraints associated with recovery of the circulating medium, are typically treated by water-based gravity separation processes that include water-only cyclones (WOCs), spirals, teeter-bed separators, or a combination of these unit operations. The ultrafine size fraction, which is too small for gravity separation processes, is usually upgraded by froth flotation. The entire ultrafine coal slurry may be treated by flotation, deslimed to remove a substantial portion of <45-µm solids prior to flotation, or discarded as waste without treatment (Bethell and Luttrell 2005). Dewatering of the coal products is typically performed using centrifugal basket-type dryers for the coarser factions and screen-bowl centrifuges for the finer fractions. Vacuum filters are used in a few plants but are difficult to justify unless a thermal dryer is available to ensure that moisture specifications can be met. Figure 2 illustrates the range of particle sizes that are typically treated by commercially available processes for the sizing, cleaning, and dewatering of coal feedstocks. Unfortunately, site-specific variations in the liberation characteristics and mineralogical makeup of different ROM coals preclude the adoption of a standard flow sheet that is appropriate for all commercial sites. Differences in circuit layouts are typically justified by designers based on both technical and financial considerations that attempt to balance the necessity to accommodate a

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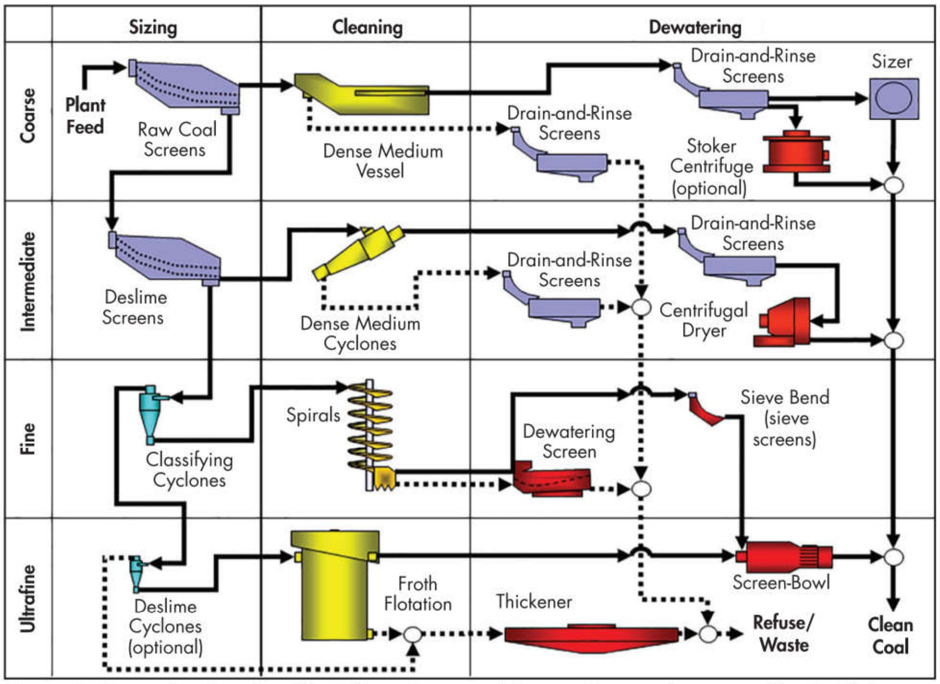


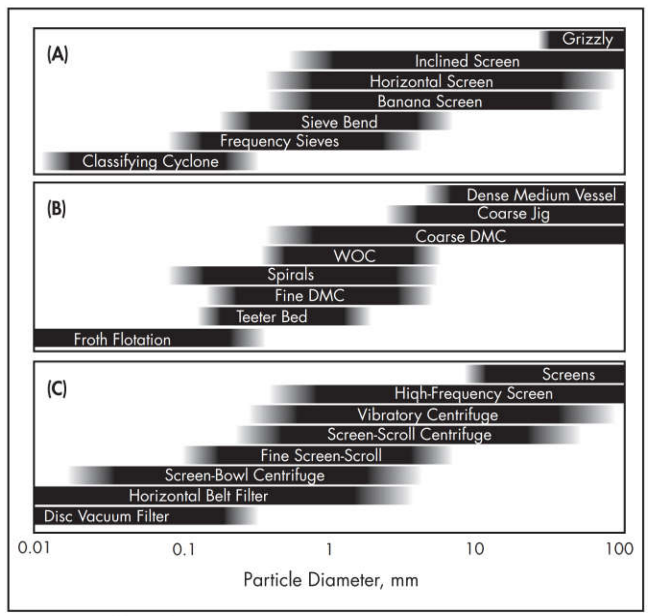
Figure 1 Example of a simplified flow sheet used by modern coal preparation facilities

specific raw coal size distribution or washability against any undesirable increase in capital costs or operating and maintenance costs. Operator preferences and vendor biases also contribute to the large variations in how coal is sized, cleaned, and dewatered. Additional descriptions and details related to modern coal preparation practices and trends can be found in two informative books: *Designing the Coal Preparation Plant of the Future* (Arnold et al. 2007) and *Challenges in Fine Coal Processing, Dewatering, and Disposal* (Klima et al. 2012).

Particle Size Separations

Coarse Particle Screening

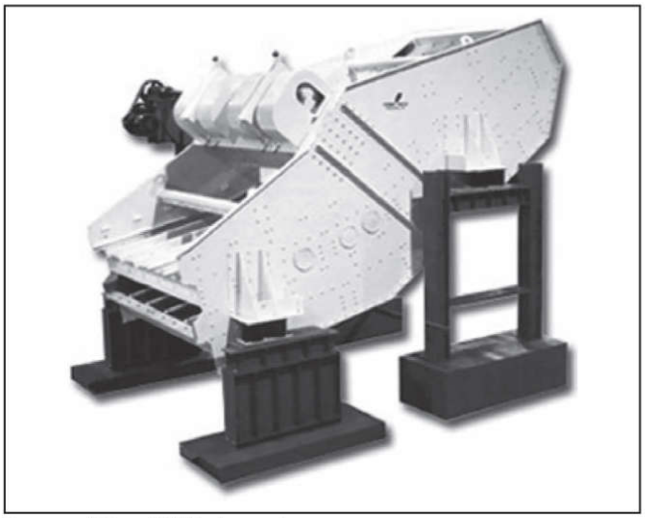
The first step in coal preparation involves crushing and sizing ROM coals into acceptable size classes. The selected top size should balance improvements in liberation with reductions in performance created by the need to treat a higher percentage of the feed tonnage in less-efficient fine coal circuits. The vast majority of equipment used in coal-sizing operations is mature technology and includes well-known unit operations such as screens, sieves, and classifying cyclones. Figure 2A shows the typical sizes of particles that can be produced by these common types of industrial sizing operations. As indicated by this chart, large particles are typically sized using one or more types of screening systems. Screens are simply mechanical sizing devices that use a mesh or perforated plate to sort particles into fine and coarse fractions; that is, particles either pass through the screen openings or are retained on the screen surface. Vibrating screens use an off-balance rotating mechanism to create oscillations/vibrations that segregate particles and move material along the screen surface. Traditional designs of inclined and horizontal vibrating screens have been displaced in recent years by multi-slope (banana) screens, largely because of the high unit capacity and improved sizing efficiency offered by these machines (Figure 3). The steep slope at the feed end creates a thin bed of particles that can more rapidly pass through the screen openings, while the shallow slope at the discharge end provides a lower velocity and repetitive attempts for high-efficiency sorting of near-sized particles.



Source: Arnold et al. 2007

Figure 2 Particle size ranges for (A) sizing, (B) cleaning, and (C) dewatering equipment used in coal preparation

Vibrating screens are used in the coal industry for feed scalping, raw coal sizing, and drain-and-rinse (D&R) applications. Typical operating conditions are given in Table 1. When necessary, heavy-duty scalping screens are used remove large unbroken rock from ROM feeds. After scalping, raw coal screens and deslime screens are used to size feeds



Courtesy of Conn-Weld Industries Inc.
Figure 3 Multi-slope banana screen

Table 1 Typical operating conditions for vibrating screens*

Screen Type	Speed, rpm	Throw, in.	Motion	Water
Scalping	700–800	3/8–1/2	Circular	No
Raw coal	700–900	5/16–7/16	Circular	Yes
Deslime	900–950	3/8–1/2	Straight	Yes
Drain-and-rinse	900–950	3/8–1/2	Straight	Yes

*These are approximate guidelines; actual demands may vary depending on site conditions.

Table 2 General recommendations for water addition to screens*

Screen Type	Recommendation
Raw coal	5–7 gpm of spray water per stph of dry feed
Deslime	4–7 gpm of dilution water per stph of dry feed with 100 gpm of spray water per foot of screen width
Drain-and-rinse	3–5 gpm of spray water per stph of dry feed (or at least 30–50 gpm per foot of screen width)
Fine sieves	Add dilution water to ensure <25% feed solids

*These are approximate guidelines; actual demands may vary depending on site conditions.

for dense medium vessel and DMC circuits, respectively. A double-deck screen with two different deck openings is often used for both sizing steps. These types of screening systems require water sprays/boxes to ensure efficient removal of fines. Recommended water addition rates are listed in Table 2. D&R screens are used to recover and rinse magnetite medium from products generated by dense medium separators. These screens must be sufficiently long to ensure time is available to drain medium, rinse medium, and dewater the solid products. D&R applications often require a combination of flume/banana screens or sieve bend/horizontal screens for efficient medium removal.

Fine Particle Sieving

Screening of very fine particles is typically achieved using static sieve screens (sieve bends), which use a curved panel equipped with slotted bars perpendicular to the flow to slice material from the flowing stream. Sieve screens are often necessary because of the low capacity and increased likelihood of plugging openings in vibrating screens. Sieves use profile wires turned perpendicular to the flow to serve as cutting edges to slice water/fines from the primary flow (Figure 4). Since the sizing is done by water partitioning, it is important to maintain the feed solids content to less than 25%–30% by weight. Recently, operators have begun to install sieves in two sequential stages to further reduce fines misplacement to the oversize product (Bethell and Luttrell 2005). Sieve cut size can be estimated from a rule of thumb that states that the required slot size (in millimeters) can be found by dividing the target cut size (in mesh) into 21. Based on this equation, a target 28 mesh cut size would dictate the use of a 0.75-mm slot opening. Sieve capacity, which is normally reported as flow capacity per unit of width, can be estimated from the values provided in Table 3. A well-designed feed distribution box and properly balanced feed gate are essential to ensure that the full width of the sieve receives feed slurry.

Courtesy of Conn-Weld Industries Inc.
Figure 4 Fine-wire sieve bend

High-frequency screens are also used in the coal industry, but only for the dewatering of fine waste solids prior to disposal. While this type of screen is inefficient for particle sizing, the thick bed of solids created by the high vibrational frequency is ideally suited for solids retention and water rejection. Also, when highly efficient fine sizing is required, coal operators are forced to consider advanced sieving technologies such as the Derrick StackSizer, which can efficiently segregate particles at sizes as fine as 0.075 mm via the use of integrated washing/repulping channels placed at strategic points down the sieve surface (Hollis 2006; Brodzik 2007; Mohanty et al. 2002, 2009).

Table 3 Approximate sieve bend capacities*

	gpm/ft for Specified Wire Designation in inches				
Slot, mm	0.03	0.045	1/16	3/32	1/8
1/4	100	60	30	—	—
1/3	115	75	40	—	—
1/2	135	125	90	70	55
3/4	—	—	175	135	110
1	—	—	215	170	135
1 1/4	—	—	235	190	165
1 1/2	—	—	260	215	190
1 3/4	—	—	280	230	210
2	—	—	300	250	225

Courtesy of Conn-Weld Industries Inc.

*Arc angle = 60°, radius = 40 in. Actual capacity varies depending on feed solids, type and amount of liquid, and desired efficiency.

Fine Particle Classification

Conventional screening and sieving of very fine particles is impractical because of capacity limitations (Firth and O'Brian 2003). Therefore, plant designers are forced to employ classifying cyclones (Figure 5) for many fine sizing applications including raw coal (0.15–0.25 mm) sizing and deslime (0.04–0.05 mm) sizing. Classifying cyclones are also used in the coal industry as clean coal cyclones for thickening and desliming ahead of dewatering units and as effluent cyclones for recovering solids from streams for the purpose of clarification. Despite the perceived advantages, classifying cyclones are typically not used in two stages in the coal industry because particle density effects unavoidably concentrate ash-bearing minerals in the underflow product (Honaker et al. 2007). Table 4 lists typical capacity ranges and cut sizes for coal applications. As expected, cyclone diameter is the variable that primarily determines both the particle cut size and capacity. Most commonly, cyclone diameters of 20, 15, and 6 in. are used to produce nominal cut sizes of 0.25, 0.15, and 0.045 mm for raw coal sizing and desliming applications. Feed solids are generally held below 12%–15% solids by weight for 15- and 20-in. raw coal cyclones and below 6%–8% solids by weight for 6-in. deslime cyclones. Inlet pressures are held in the 15–25 psi range for most applications, with the higher-pressure region more common for smaller cyclones. Because of the lack of moving parts, cyclones require little maintenance in coal plants. When problems do occur, potential causes for losses of coarse particles in the overflow include plugged apex, low inlet pressure, misaligned liner, worn vortex finder, and plugged vent pipe. Similarly, primary causes for excess fines and/or water in the underflow product include worn apex and high feed solids.

Solid–Solid Separations

Dense Medium Separation

The most important function of a coal processing facility is the separation of carbonaceous material from waste rock. This separation is typically accomplished using low-cost processes that segregate coal and rock based on differences in physical properties such a size, density, and wettability. As shown in Figure 2B, the effectiveness of different solid–solid separation processes is generally limited to a relatively narrow size range. For coarse particles, the most commonly used property difference for separation is density because

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Figure 5 Classifying cyclone bank

Table 4 Approximate classifying cyclone capacities and cut-size values*

Diameter, in.	Cut Size, mesh	Solids, stph	Pulp, gpm
6	200–270	2–5	80–200
10	150–200	5–10	200–400
15	100–150	11–21	420–800
15†	65–150	16–29	600–1,000
20	65–100	18–34	700–1,300
20†	48–65	29–47	1,100–1,800
26	48–65	39–78	1,500–3,000

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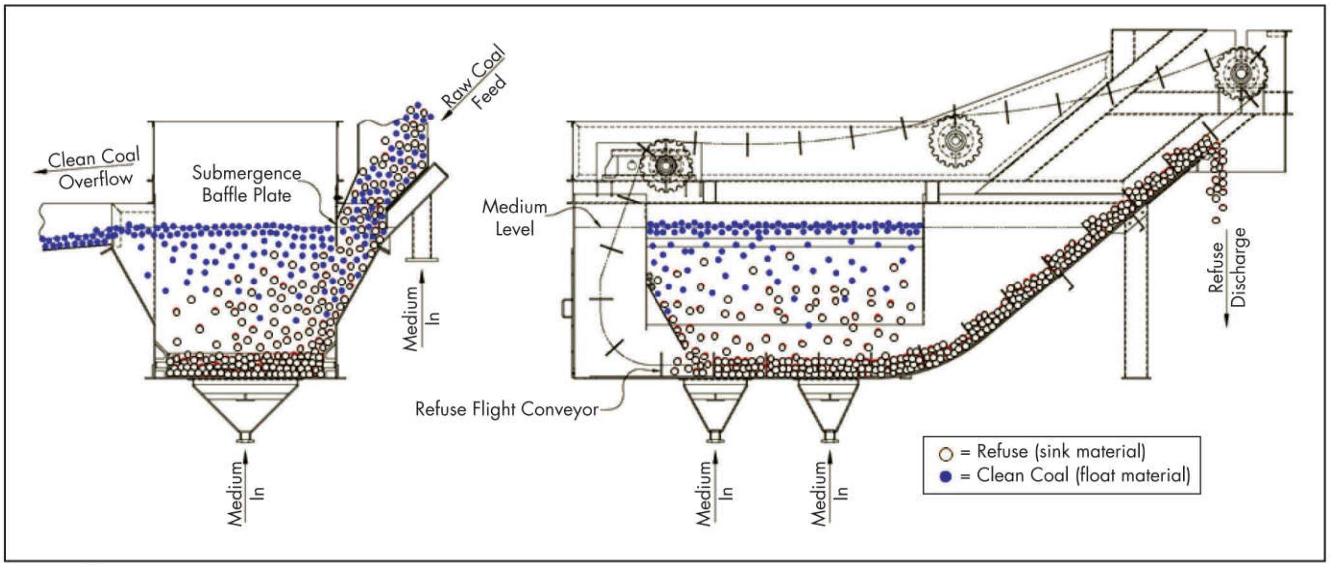
*These are approximate guidelines; actual demands may vary depending on site conditions.

†Extended body cyclone design.

pure organic coal has a density in the range of 1.26–1.32 SG (specific gravity), while waste rock has a much higher density of 2.2–2.7 SG. Therefore, one of the most popular and efficient methods for treating ROM coals coarser than 1/4-in. (6.3–12.7 mm) is the dense medium vessel. This high-capacity separator (Figure 6) consists of a large open tank through which a dense suspension of finely micronized magnetite in water is circulated. Low-density coal particles introduced into the suspension float to the surface of the dense medium where they are transported by the overflow medium onto a collection screen, while high-density waste rock sinks to the bottom of the vessel where it is collected and discarded by a series of mechanical scrapers called *flights*. The washed coal and rock products pass over D&R screens to wash the magnetite medium from the surfaces of the products and to dewater the particles (Figure 7). As shown in Table 5, the clean coal capacity for a dense medium vessel can be estimated from the overflow weir length and medium density. The refuse capacity can be estimated from the flight height and width as shown in Table 6. The medium is circulated through the vessel at a rate of approximately 250 gpm/ft of weir length.



Courtesy of Peters Equipment

Figure 6 Dense medium vessel

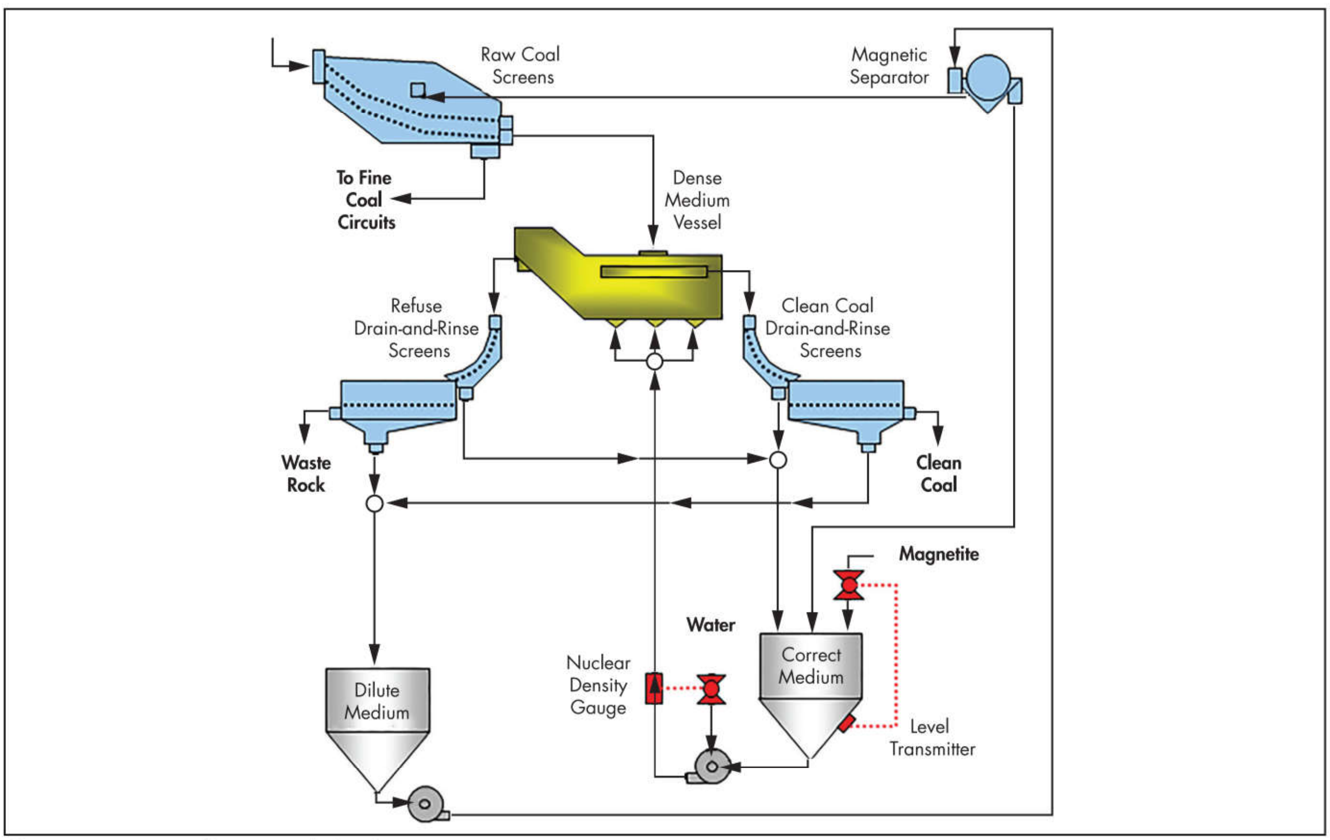


Figure 7 Dense medium vessel circuit

Dilute medium rinsed from the D&R screens is recovered and recycled back through the circuit using a drum-style magnetic separator (Figure 8). Modern magnetic separators incorporate stronger magnets (950 gauss interpole) and a wider magnetic arc (up to 140°) to increase capacity and to reduce losses of magnetic solids down to <0.5 g/gal of treated slurry (Bethell and Barbee 2007). Typical capacities for the standard 36- and 48-in.-diameter drums are 70–120 and 100–170 gpm of dilute medium and 5.5 and 8.0 stph of magnetite solids per foot of drum width, respectively.

Particles too small to be efficiently treated by a static dense medium vessel are upgraded using DMCs (Figure 9).

Table 5 Approximate clean coal capacities for a dense medium vessel*

Medium, SG	Specific Capacity, stph/ft	Clean Coal Capacity, stph, for Specified Weir Length 4 ft	8 ft	12 ft	16 ft
1.4	1.44	100	200	300	400
1.5	1.21	120	240	360	480
1.6	1.17	140	280	420	560
1.7	1.15	160	320	480	640

Courtesy of Peters Equipment
*These are approximate guidelines; actual demands may vary depending on site conditions.

Table 6 Approximate reject capacities for a dense medium vessel*

Flight Height, in.	Specific Capacity, stph/ft	Reject Capacity, stph, for Specified Flight Width 4 ft	4½ ft	5 ft	5½ ft
8	25	261	294	326	359
9	30	316	355	394	433
10	35	370	416	462	508
11	40	424	477	530	583

Courtesy of Peters Equipment
*These are approximate guidelines; actual demands may vary depending on site conditions.

These high-capacity units make use of the same basic principle as dense medium vessels, that is, an artificial magnetite–water medium is used to separate low-density coal from high-density rock. In this case, however, the rate of separation is increased by the centrifugal effect created by passing medium and coal through one or more cyclones. Modern DMCs are capable of treating particle sizes as large as 3–3.5 in. (75–90 mm), which can in many cases eliminate the need for a static vessel (Ziaja and Yannoulis 2007). Nominal capacities and operating limits for DMCs of different diameters are listed in Tables 7 and 8, respectively. Typically, apex and vortex diameters are selected to provide, a two-thirds split of medium to overflow and a medium-to-coal ratio in the feed stream of about 4:1 by volume. The diameters of the inlet, vortex finder, and apex are typically a function of cyclone diameter (Dc) and fall in the ranges of 0.2–0.3 × Dc, 0.43–0.50 × Dc, and 0.3–0.4 × Dc, respectively (De Korte and Engelbrecht 2007).

Gravity Separation

Coal particles that are smaller than 1 mm are too small to be effectively handled by the D&R screens required by dense medium processes. Therefore, particles in this size range are commonly processed using water-based gravity separators. As shown in Figure 2B, there are several alternatives for cleaning this size of material, including WOCs, spirals, and teeter-bed separators. WOCs are similar to classifying cyclones but are configured with a truncated wide-angled conical bottom to emphasize density partitioning and suppress classification effects (Figure 10). WOCs of larger diameter are typically required to process coarser solids, as shown in Table 9. The self-generated medium of solids and water within the WOC allows a separation to be made between low-density coal and high-density rock, although the separation is usually not efficient because coarser particles tend to separate at a much lower density than finer particles (Majumder 2011). As a result, WOCs have been largely displaced in the U.S. coal industry by spiral separators. Spiral technology consists of a corkscrew-shaped helical trough supported along a central pole. As slurry passes down the trough, the resultant flowing

Courtesy of Eriez Magnetics
Figure 8 Drum-style magnetic separators

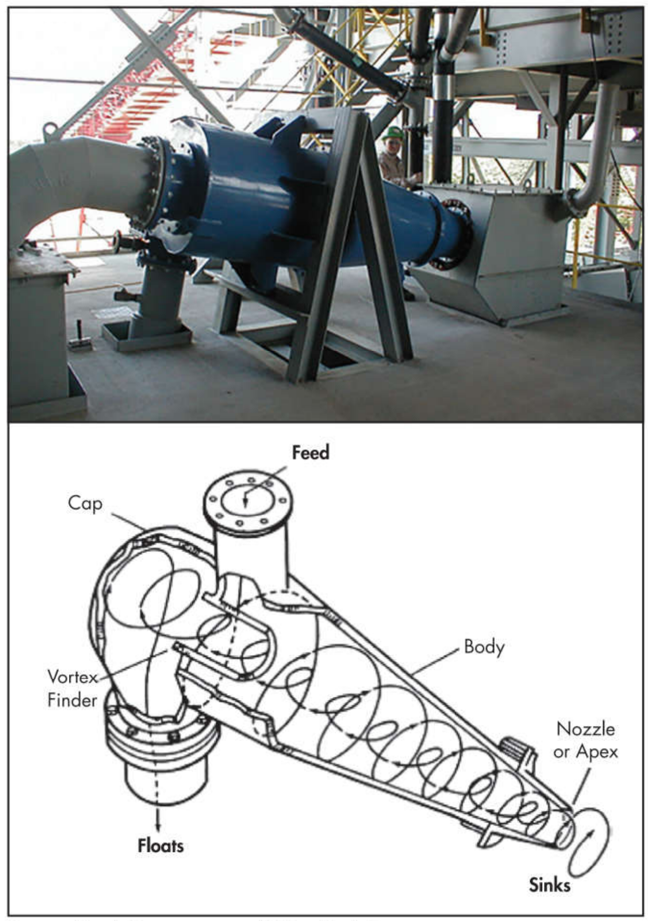


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Figure 9 Dense medium cyclone

Table 7 Approximate capacity ratings for dense medium cyclones*

Diameter, in.	Top Size, in.	Solids, stph	Slurry, gpm
20	¾	85	1,050
26	1½	160	2,045
30	1½	225	2,830
33	2	290	3,650
40	2½	460	5,760
44	3	580	7,280
48	3	725	9,100
55	4	985	12,300

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Table 8 Operating conditions for dense medium cyclones*

	Approximate Inlet Pressure (psi) at Specified Medium Specific Gravity		
Diameter, in.	1.5 SG	1.6 SG	1.7 SG
20	10–12	11–13	12–14
26	13–16	14–17	15–18
30	15–18	16–20	17–21
33	17–20	18–21	19–23
40	20–24	21–26	23–28
44	22–27	23–28	25–30
48	24–29	25–31	27–33
55	27–33	29–35	31–38

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film segregates particles according to density, although variations in particle size and shape also influence the separation. Because of the low unit capacity (<2–4 stph per spiral), spirals are usually arranged in groups that are fed by an overhead distributor (Figure 11). Spirals have the advantage of separating coal at a more consistent density across the size range between 0.2 and 1.0 mm but are generally unable to efficiently separate at densities lower than about 1.65 SG. To maintain acceptable cut points and separation efficiencies, spirals typically have to be run at less than 2–3 stph per spiral and at volumetric slurry rates of about 35–40 gpm per spiral. Also, protection sieves are often used ahead of the spiral circuit to keep oversize solids (>1.5 mm) out of the feed.

Both WOCs and spirals are often employed in two stages or in combination with each other to improve separation efficiencies and reduce misplacement (Luttrell et al. 1998, 2007). In recognition of this limitation, spiral manufacturers now offer compound spirals that incorporate a two-stage rougher–cleaner arrangement along the spiral run. The efficiencies afforded by this configuration rival those of dense medium separation for this size fraction (Zhang et al. 2008). Several plants have also begun to incorporate spiral technology as desulfurization units to remove fine pyrite from fine feed slurries prior to flotation treatment (Chafin et al. 2012). This pretreatment minimizes the recovery of pyritic sulfur, and particularly pyrite containing inclusions of carbon, that often floats in flotation processes that rely on differences in particle wettability for selectivity.

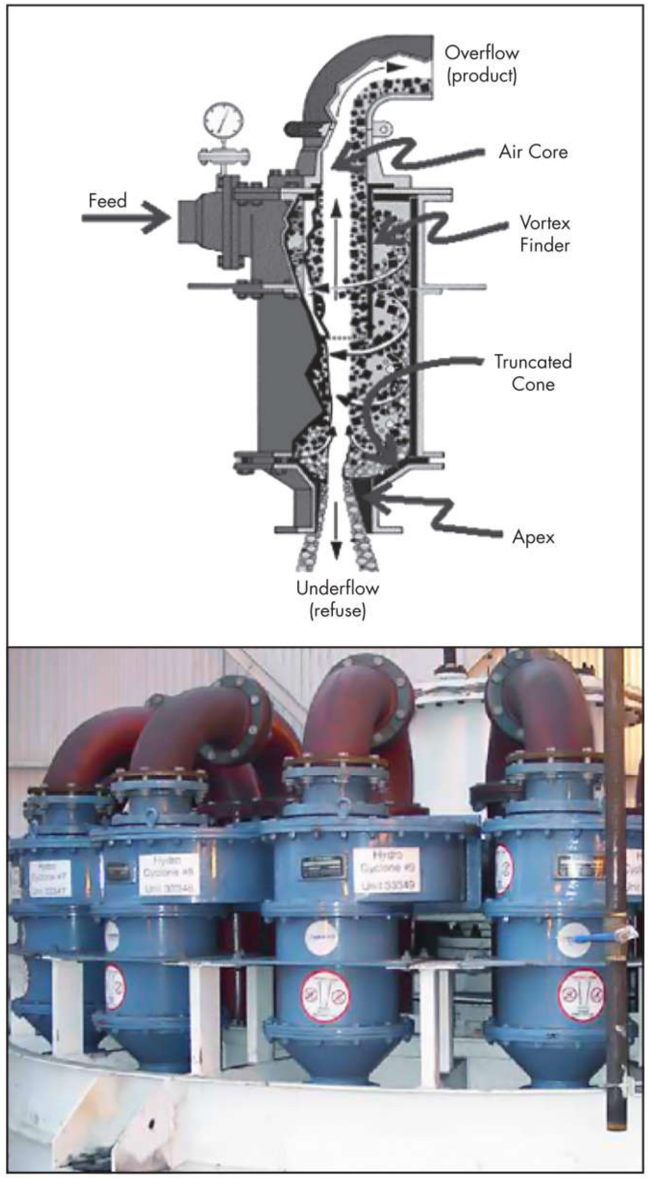

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Figure 10 Water-only cyclones

In recent years, hindered-bed separators have begun to reemerge in the coal industry. New technologies, such as the CrossFlow separator (Figure 12) and the REFLUX™ Classifier, incorporate innovative design features that improve the overall separation efficiency of this historic coal upgrading method (Kohmuench et al. 2002; Galvin 2012; Ghosh et al. 2012). Hindered-bed separators inject fluidization water across the base of the separator through a distribution network. Solids settle against the upward flow of fluidization water and form a dense bed. Consequently, light/small particles report to overflow, while dense/large particles report to underflow. Bed density is automatically controlled using an actuated underflow valve that adjusts in response to signals from pressure transmitters located in the teeter bed. Sizing of the feed solids is critical because fines report to overflow regardless of density. Compared to spirals, this technology has the advantage of being able to operate at substantially lower density cut points (Luttrell et al. 2007). Full-scale industrial units are capable of feed rates >100 stph for feeds as fine as 1 × 0.25 mm (Galvin et al. 2002). Feed solids contents in the range of 30%–40% by weight are typically preferred to maintain separation efficiency. Fluidization water rates vary depending on the application but are typically in the range of 4–9 gpm/ft^2 of cross-sectional area. Also, as with any gravity separator, the presence of excessive oversize solids in the feed should be avoided.

Froth Flotation Separation

Froth flotation is the most common method used to upgrade ultrafine coal particles <0.15 mm. This application of flotation, which has been recently described by Laskowski et al. (2007), exploits inherent differences between hydrophobic carbonaceous matter and hydrophilic mineral matter. Most coal feedstocks can be floated using only a frothing agent, although hydrocarbon collectors, such as diesel fuel or fuel oil, are often added to improve flotation kinetics (Laskowski 2001). The addition rates are very small and typically about

Table 9 Operating conditions for water-only cyclones*

	Diameter, in.			
Condition	**10**	**15**	**20**	**26**
Rate, stph	7–9	20–30	50–60	95–120
Flow, gpm	325–375	775–825	1,000–1,250	1,800–1,250
Top size, mm	½	1	6.4	19
Feed solids, %	8–10	10–15	15–18	18–22
Pressure, psi	8–12	8–15	15–18	18–22

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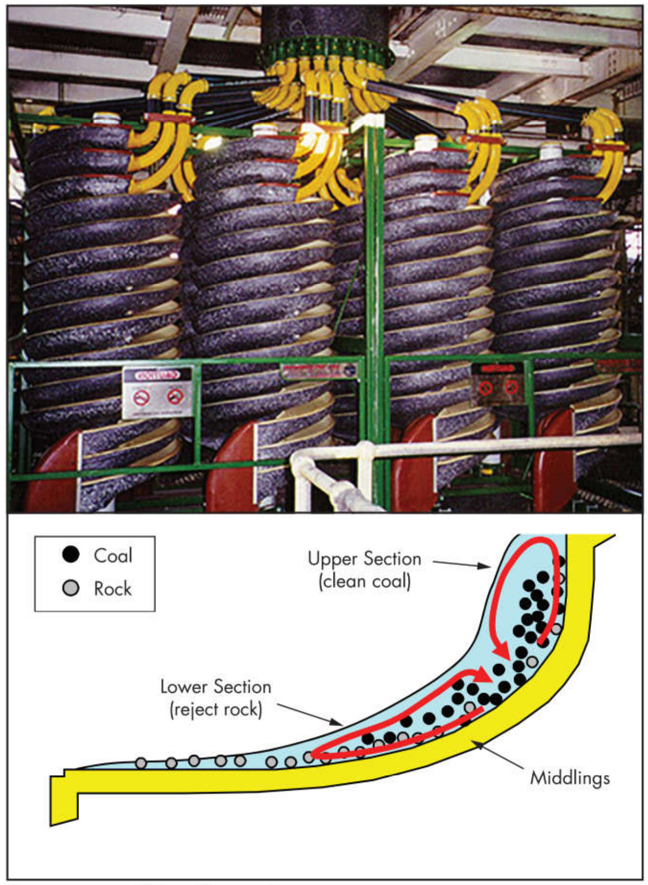


Photo courtesy of Eriez Flotation Division

Figure 11 Spiral separator

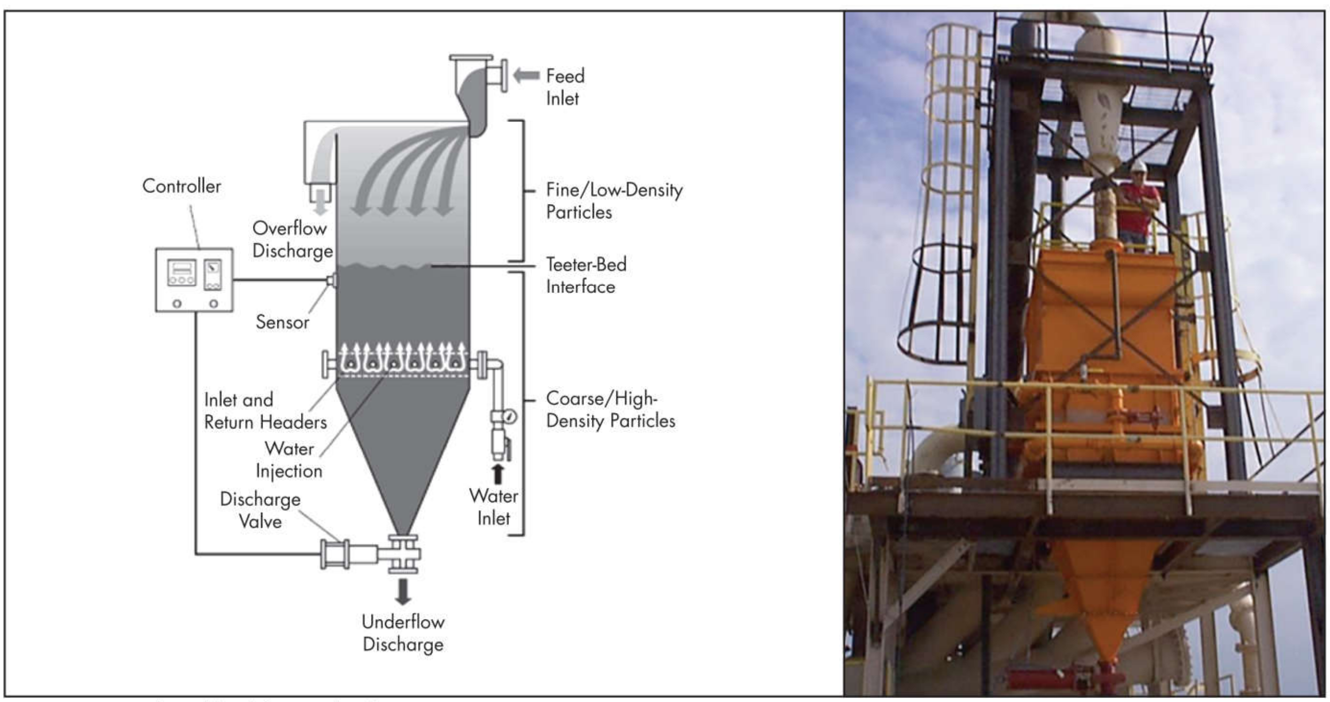
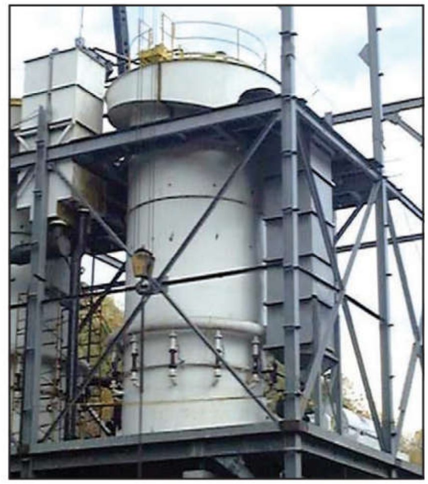


Figure 12 Hindered-bed (teeter-bed) separator



Courtesy of Eriez Flotation Division
Figure 13 Installation of column flotation cells

0.1–0.5 lb of reagent per ton of coal feed. A variety of environmentally friendly collectors, such as blends of canola, vegetable, and soybean oils, have become more common in the coal industry for applications involving underground injection of fine wastes (Skiles 2003). Near-neutral pH levels (6.5–8) are generally acceptable for the flotation of most coals.

In addition to reagent type and dosage, a wide variety of other parameters can impact the flotation of coal. For example, while particles as large as 0.5 mm can be treated in coal flotation plants, the industry typically uses this process only for particles with top sizes smaller than about 0.1–0.2 mm to attain maximum separation efficiencies. Deslime flotation circuits, in which <45-µm particles are largely removed and discarded prior to flotation, have become popular because of the low value, high moisture, low capacity, and high processing cost of ultrafine slimes (Bethell 2012). Both mechanical stirred-tank machines and flotation columns are used in the U.S. coal industry. Stirred (conventional) machines are commonly used in banks of four to five cells in series. Pulp residence times of 4–5 minutes are often sufficient to achieve good recoveries of combustible material for these types of circuits. Since the late 1990s, column-type flotation cells (Figure 13) have become favored over conventional machines largely because of the ability of columns to remove high-ash clays from the froth product via the addition of wash water to a relatively deep froth (Davis et al. 1995). The main downside is that columns typically require longer pulp residence times, with 10–15 minutes or more being used in industrial installations. These installations require water flow rates of 3–4 gpm/ft^2 and gas flow rates of 3–5 scfm/ft^2. The majority of column flotation installations incorporate either static spargers such as SlamJet gas injectors or dynamic spargers such as CavTube (Kohmuench et al. 2012) or Microcel (Yoon et al. 1992) bubble generators. More recently, a compact column known as the StackCell technology has made inroads in the coal industry by offering column-like performance in a low-profile design that eliminates the high foundation loads and larger gas compressors typically associated with column cells (Kohmuench et al. 2010).

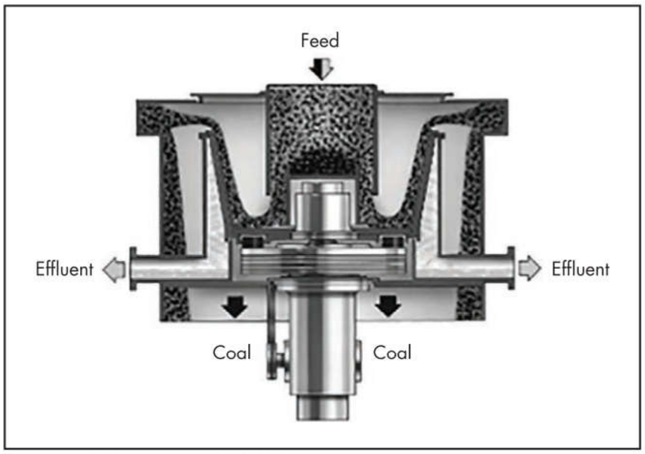
Solid–Liquid Separations

The final step in coal preparation is the removal of water from the plant products. Solid–liquid separations include dewatering units for the removal of unwanted moisture from plant products and thickening/clarification units for the removal of solids from process water. For clean coal products, unwanted surface moisture must be removed to avoid lower heating values, handling/freezing problems, and higher transportation costs. Excess moisture must also be removed from coal wastes to avoid downstream handling and disposal issues associated with coarse and fine refuse streams. Figure 2C provides an overview of the range of particle sizes that are commonly treated by the various solid–liquid separation processes used in the coal preparation industry. As should be expected, several different types of mechanical dewatering methods are required to attain optimal performance for each size class (Arnold 1999). The removal of water from coarser (>5 mm) coal is typically carried out using simple screens, while finer particles typically require the use of more intense solid–liquid separation processes such as centrifuges, filters, and thermal dryers (Meenan 2005).

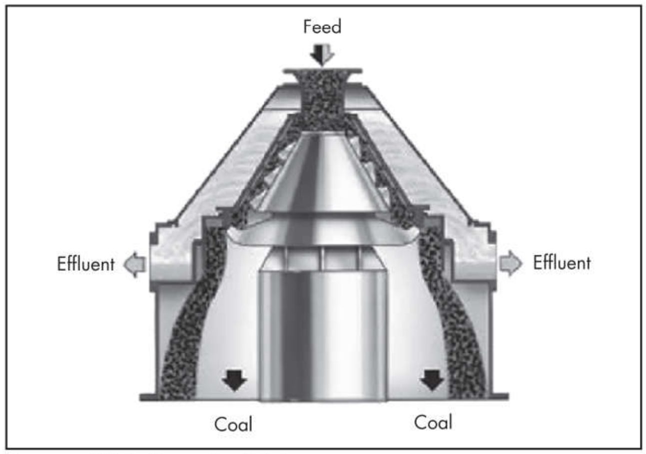
Centrifugal Dewatering

Popular centrifuge-style dewatering machines used in the coal industry include centrifugal dryers and screen-bowl centrifuges. Two designs of centrifugal dryers are the vibratory centrifuge (Figure 14) and screen-scroll centrifuge (Figure 15). The vibratory centrifuge uses a reciprocating motion to induce the flow of solids across the surface of the dewatering basket. The screen-scroll centrifuge uses a screw-shaped scroll that rotates at a slightly different speed than the basket to positively transport the solids along the basket surface. The vibratory centrifuge, which is typically is the least costly to operate, is commonly used to dewater coarser particles in the size range of 3 × ¼ in. The largest unit (56-in. diameter) can reach a capacity of more than 300 stph for a single machine. Both the screen-scroll and the vibratory centrifuges are appropriate for intermediate sizes, although the screen-scroll is more commonly used for ½ in. × 28 mesh solids. Screen-scroll machines also typically provide lower product moistures than vibratory machines for the same size of feed. The largest screen-scroll centrifuges (48-in. diameter) can provide capacities of up to 160 stph.

For finer particles <1 mm, the screen-scroll centrifuge has been displaced in modern facilities by the screen-bowl centrifuge (Figure 16). A screen bowl is a horizontal centrifuge that consists of a solid bowl section and a screen section. Slurry is introduced into the bowl section where most of the solids settle out under the influence of the centrifugal field. A rotating scroll transports the settled solids up a beach and across the screen section where additional dewatering takes place prior to product discharge. Solids that pass through the screen section are circulated back to the machine feed. This unit is capable



Courtesy of Elgin Separation Solutions
Figure 14 Vibratory centrifuge



Courtesy of Elgin Separation Solutions
Figure 15 Screen-scroll centrifuge

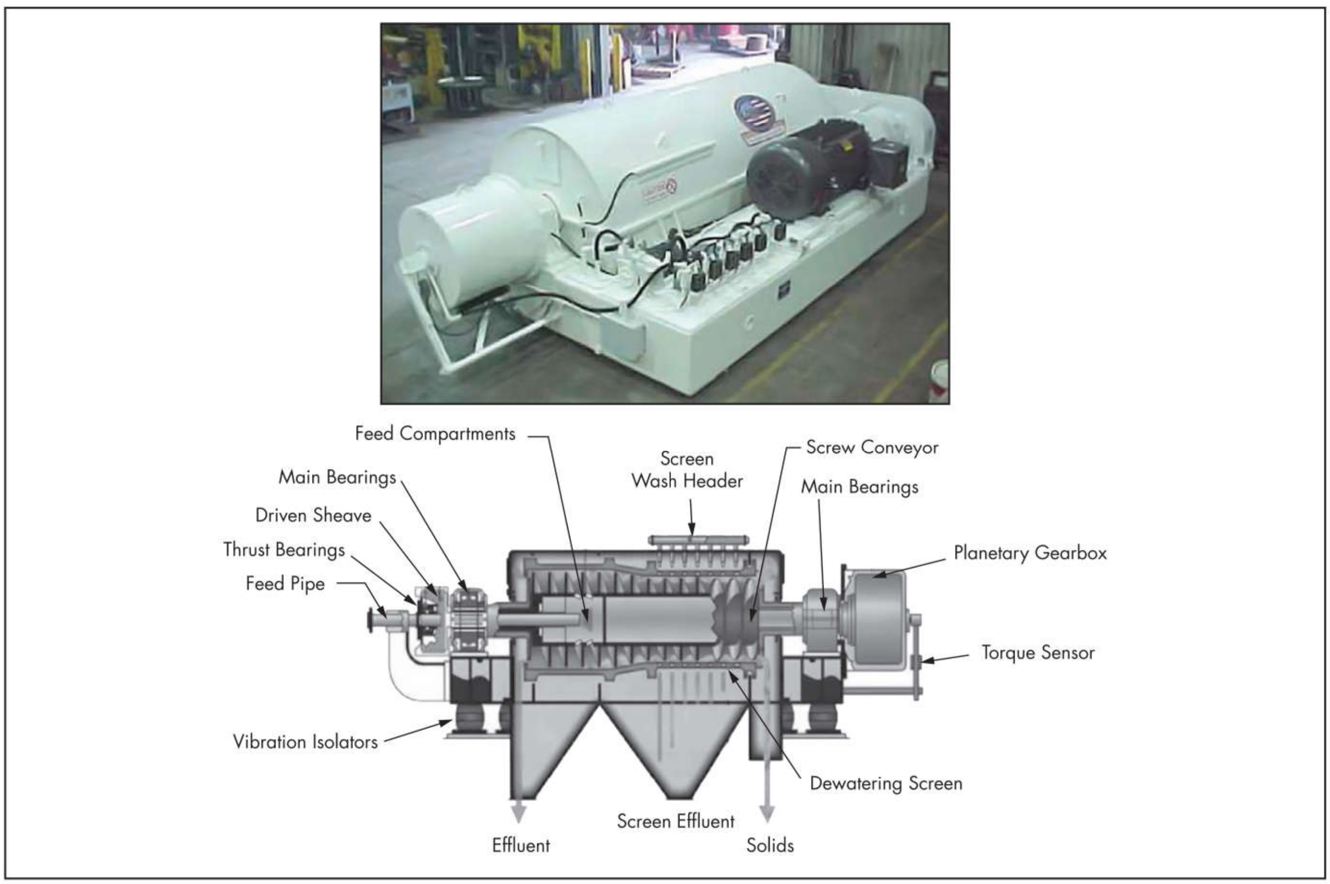


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Figure 16 Screen-bowl centrifuge

Table 10 Capacity ratings for screen-bowl centrifuges*

	Throughput Capacity		
Machine Type	Coarse, stph†	Fine, stph‡	Slurry, gpm
36 × 72	35–40	20–25	400
36 × 72H§	40–50	25–30	400
36 × 96	35–50	25–28	400
40 × 72	45–65	25–30	500
44 × 132	75–80	50–60	800
44 × 132H§	80–100	60–65	800

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*These are approximate guidelines; actual demands may vary depending on site conditions.
†Coarse = 1 × 0.1 mm and 35%–50% feed solids
‡Fine = Minus 0.1 mm and 20%–25% feed solids
§H = High capacity

of providing low moistures, although some ultrafine solids are lost with the main effluent discarded from the solid bowl section (Mohanty et al. 2009). An industry rule of thumb is that screen-bowl centrifuges recover +98% of the +0.1 mm solids and about 50% of particles finer than 45 μm (Schultz et al. 2008). An estimate of product moisture from a screen-bowl centrifuge can be obtained from the following equation:

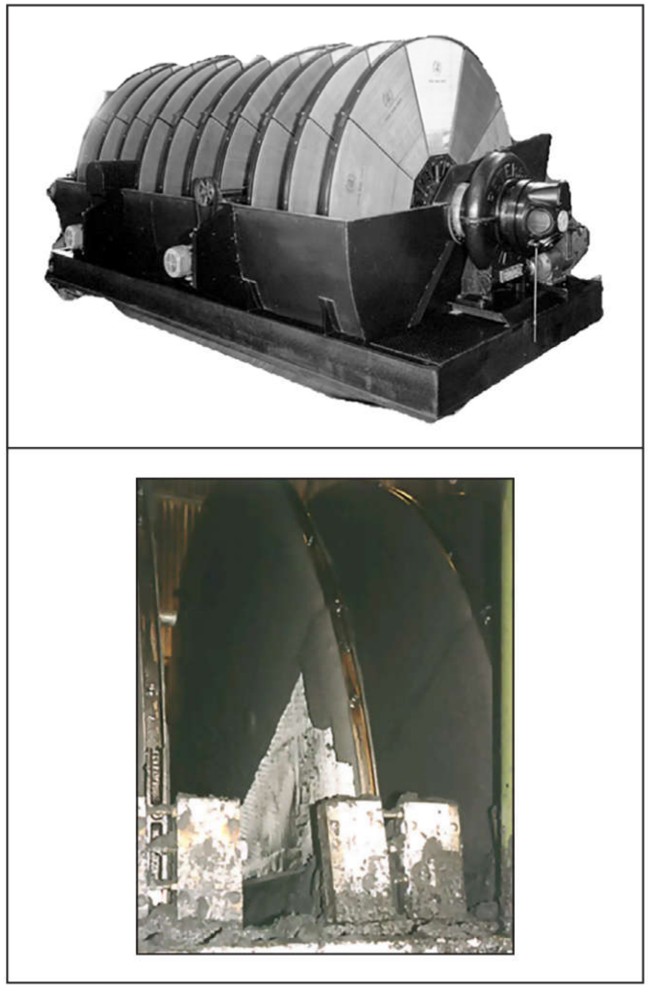
$$\text{moisture, \%} = 9.2 + 0.01(S)^2 \quad \textbf{(EQ 1)}$$

where S is the percentage of –45 μm solids in the feed. For maximum coal dryness, screen-bowl centrifuges should be run using a effluent weir setting that provides a low pool depth of 50 mm (2 in.) or less. For maximum recovery of solids, screen-bowl centrifuges should be run at a weir setting that provides a pool depth of 76–114 mm (3–4.5 in).

Approximate production capacities for screen-bowl centrifuges are listed in Table 10. The load placed on a screen-bowl centrifuge is monitored online using torque sensors and amp meters. The torque sensor is used to monitor the solids loading on the scrolling mechanism. An increase in torque indicates that more solid material is being fed to the unit. An amp meter installed on the drive is used to monitor the overall load generated by the rotating assembly. Higher amps typically indicate that more feed is going into the machine, although blinding of one of the discharge hoppers can also increase the amp reading.

Filtration Dewatering

If high coal recovery is desirable, higher-cost filtration processes may be used to dewater fine coal (Meenan 2005). Historically, flotation concentrates were dewatered in the coal industry using disc vacuum filters. Disc filters consist of a series of circular filtration discs that are constructed from independent wedge-shaped segments (Figure 17). At the beginning of a cycle, a segment rotates down into the filter tub filled with fine coal slurry. Vacuum is applied to the segment so that water is drawn through the filter media and a particle cake is formed. The resultant cake continues to dry as the segment rotates out of the slurry and over the top of the machine. The drying cycle ends when the vacuum is removed and the cake is discharged using scrapers and a reverse flow of compressed air. Filters typically produce products that are higher in moisture (25%–35%) than screen-bowl centrifuges but also achieve higher solids recoveries (95%–99%) than screen-bowl centrifuges (80%–90%). Cake capacities for disc filters typically fall in the range of 40–50 lb/h of dry solids per square foot of filter disc area for clean coal. These units, which typically operate at vacuums of 13–15 in. Hg, require vacuum pumps that deliver 5–7 cfm of gas flow per square foot of disc area. Belt filters are commonly used in countries such as Australia but are not typically employed in the mountainous U.S. coalfields because of footprint constraints. In extreme applications, plate-and-frame pressure filters have been placed into service in the coal industry for dewatering ultrafine particles. These batch-type units are effective, but their use is difficult to justify across the coal industry because of high capital/operating costs and high-pressure gas safety concerns.



Courtesy of Peterson Filters Corp.
Figure 17 Coal disc filter

Thickening/Clarification

Solid–liquid separation is a key issue for water conservation and waste disposal and typically involves clarification technologies such as thickeners, belt filter presses, and plate-and-frame pressure filters (National Research Council 2002). Thickening is performed in all coal plants to clarify dilute slurry rejected from fine coal circuits. A thickener is a large settling tank that produces a clarified overflow that can be reused in the plant as process water and a thickened underflow containing 20%–35% solids by weight that is typically discarded into a waste impoundment (Figure 18). Chemical additives are introduced

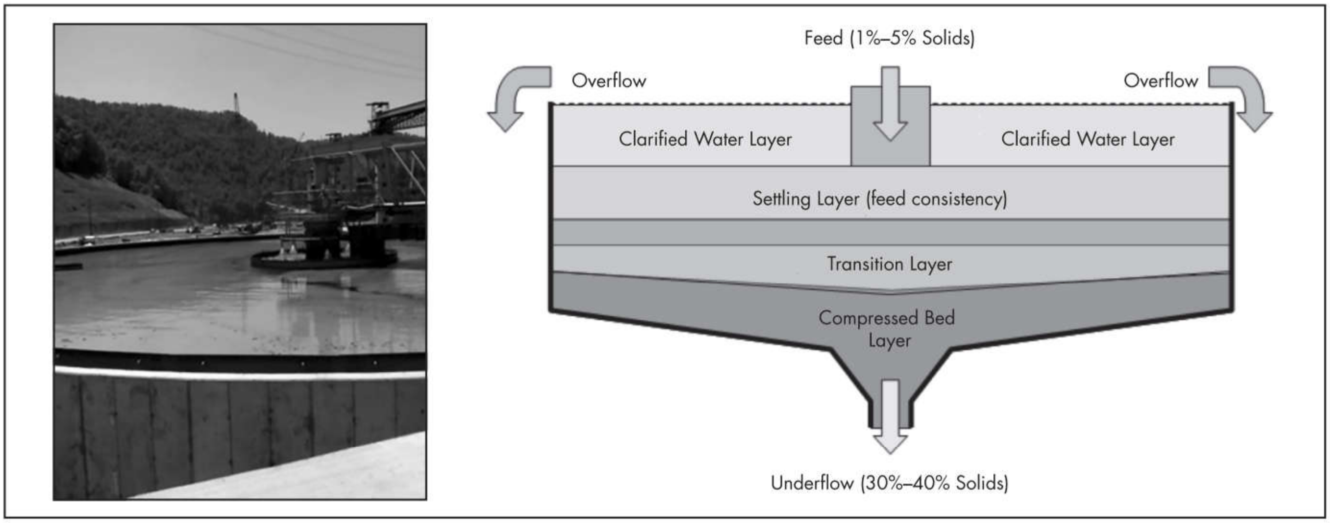


Figure 18 Conventional coal waste thickener

to the thickener feed to promote aggregation of ultrafine particles. These additives include coagulants, which neutralize particle charges so that self-aggregation occurs, and flocculants, which consist of long-chain high-molecular-weight molecules that bridge particles together. Proper chemical dosing is needed to ensure that good water clarity and proper settling rates are maintained. Control of slurry pH is also important, with an acceptable range being 6.5–8 for most sites. An estimate of 1 ft^2 of surface area per gallon per minute of feed slurry is often used for first-pass designs of coal thickeners. Coal thickeners require constant monitoring to ensure that overflow clarity and underflow density are maintained and the rake mechanism is not overloaded. In some cases, the underflow may be further dewatered using belt filter presses or plate-and-frame pressure filters to meet strict disposal requirements imposed by some waste disposal permits.

CHARACTERIZATION

Particle Size Analysis

The most important characterization procedure used in coal preparation is particle size analysis. Sizing is typically performed using standard square-grid screens/sieves with shaking/rapping times of <10 minutes to avoid degradation of friable coals. The mass of dry solids retained on each screen is then cumulated as shown in Table 11. Columns 1 and 2 are the passing and retained screen sizes, respectively. Column 3 is the individual weight percentage retained on each screen or sieve. The cumulative values for the weight retained for each screen size is tabulated in column 4. For coal, the cumulative mass retained can often be plotted as a linear relationship using Rosin–Rammler graph paper based on the following equation:

$$Y = 100\exp(-D/K)^m \quad \textbf{(EQ 2)}$$

where Y is the cumulative weight percent retained on the screen or sieve, D is the screen or sieve opening (particle size), K is the size constant, and m is the distribution constant. The Rosin–Rammler plot is obtained by plotting the cumulative percent retained (column 4) versus the retained opening size (column 2). An example is provided in Figure 19. The solid line corresponding to K = 14.6 and m = 0.75 represents the

Table 11 Example of particle size data for a ROM coal

[1] Passing, mm	[2] Retained, mm	[3] Mass %	[4] Cumulative % Retained
102	50.8	6.76	6.76
50.8	25.4	15.38	22.14
25.4	12.7	18.58	40.72
12.7	9.53	8.44	49.16
9.53	1.5	34.58	83.74
1.5	1.0	1.90	85.64
1.0	0.25	8.96	94.60
0.25	0.15	2.29	96.89
0.15	0.075	1.60	98.49
0.075	0.044	0.65	99.14
0.044	0.00	0.86	100.00

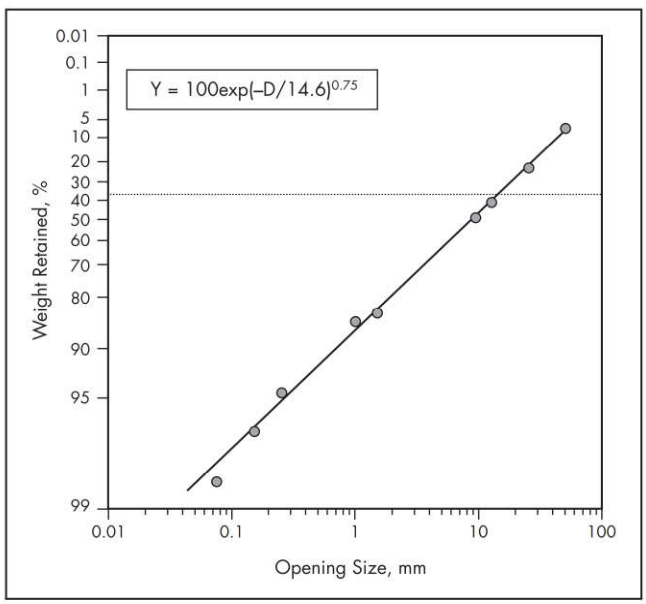


Figure 19 Rosin–Rammler graph for a ROM coal

Table 12 Washability data for three size fractions of ROM coal

Specific Gravity		Individual			Cumulative Float			Cumulative Sink		
[1]	[2]	[3]	[4]	[5]	[6]	[7]	[8]	[9]	[10]	[11]
Sink	Float	Weight %	Ash %	Sulfur %	Weight %	Ash %	Sulfur %	Weight %	Ash %	Sulfur %
A. 50 × 10 mm (56.82% of feed)										
1.26	1.30	10.16	5.67	0.56	10.16	5.67	0.56	100.00	62.01	0.60
1.30	1.40	16.24	11.37	0.71	26.40	9.18	0.65	89.84	68.38	0.61
1.40	1.50	3.20	21.68	1.27	29.60	10.53	0.72	73.60	80.96	0.58
1.50	1.60	2.56	31.53	1.49	32.16	12.20	0.78	70.40	83.66	0.55
1.60	1.80	2.83	40.74	2.39	34.99	14.51	0.91	67.84	85.63	0.52
1.80	2.00	2.60	55.90	2.04	37.59	17.37	0.99	65.01	87.58	0.44
2.00	2.40	62.41	88.90	0.37	100.00	62.01	0.60	62.41	88.90	0.37
B. 10 × 1 mm (24.21% of feed)										
1.26	1.30	39.20	5.53	0.57	39.20	5.53	0.57	100.00	30.71	0.83
1.30	1.40	23.06	11.33	0.72	62.26	7.68	0.63	60.80	46.94	1.00
1.40	1.50	5.47	19.42	1.15	67.73	8.63	0.67	37.74	68.70	1.17
1.50	1.60	2.46	28.93	1.65	70.19	9.34	0.70	32.27	77.06	1.17
1.60	1.80	1.79	39.07	2.28	71.98	10.08	0.74	29.81	81.03	1.13
1.80	2.00	2.40	51.22	2.43	74.38	11.41	0.80	28.02	83.71	1.06
2.00	2.40	25.62	86.75	0.93	100.00	30.71	0.83	25.62	86.75	0.93
C. 1 × 0.1 mm (18.97% of feed)										
1.26	1.30	43.05	3.51	0.64	43.05	3.51	0.64	100.00	22.38	0.78
1.30	1.40	26.92	9.95	0.74	69.97	5.99	0.68	56.95	36.65	0.88
1.40	1.50	6.06	19.14	1.03	76.03	7.04	0.71	30.03	60.59	1.01
1.50	1.60	2.45	28.14	1.52	78.48	7.69	0.73	23.97	71.06	1.00
1.60	1.80	2.50	34.76	1.78	80.98	8.53	0.76	21.52	75.95	0.94
1.80	2.00	1.55	52.46	2.52	82.53	9.36	0.80	19.02	81.37	0.83
2.00	2.40	17.47	83.93	0.68	100.00	22.38	0.78	17.47	83.93	0.68

best fit through the coarse end of the size distribution. More than one line segment may be necessary to represent sizing data from upstream material handling operations that degrade the feed coal from its natural size distribution or from feed streams that contain particles from multiple sources with different inherent breakage properties.

Washability Analysis

The ability of coal preparation to improve coal quality varies widely from site to site because of variations in the liberation characteristics of ROM coals. The degree of liberation, and ultimately the coal cleanability, can be quantified in the laboratory using washability (float–sink) analysis. Washability tests are performed by sequentially passing a sized coal sample through flasks/tanks of dense liquids of increasingly higher densities, as described by ASTM D4371-06 (ASTM International 1994). Table 12 shows size-by-size washability data collected for a ROM coal feed crushed to a 2-in. (50-mm) top size. Columns 3–5 are the experimental mass, ash, and sulfur values for each density class listed in columns 1–2. These individual class values are cumulated in columns 6–8 and 9–11 for the cumulative float and cumulative sink products, respectively. Based on this data, the coarsest 50 × 10 mm fraction floated at a density of 1.6 SG would produce a clean coal product containing 12.20% ash and 0.78% sulfur at a clean coal yield of 32.16%. The reject product, which consists of 1.6 SG sink, would consist of the remaining 67.84% reject yield and would contain 85.63% ash and 0.52% sulfur.

The size-by-size washability analyses are typically cumulated across all size fractions to obtain total yield and quality data expected for a given feed coal. For example, Table 13 shows the cumulated data for the size-by-size washabilities listed in Table 12 for a plant processing 1,200 stph of dry ROM feed. Because of the near-linear relationship between mean specific gravity and ash, the maximum total clean coal yield is usually obtained when all size classes are separated at the same density (Abbott 1982). Therefore, as shown in Table 13, section A, separating all three size fractions at identical 1.60 SG produces 549 stph of clean coal at a 9.76% ash content. Separation at different SG values in an attempt to independently produce a 9.76% ash for all three size fractions results in a suboptimal operating point that provides only 540 stph of clean product at 9.76% ash, as indicated in Table 13, section B. The net difference of 9 stph represents a potential revenue gain of more than $2.5 million annually (i.e., 9 stph × $50/st × 5,700 h/yr = $2.57 million/yr). The selection of optimal SG values for cases involving quality specifications other than ash can be handled using an optimization method known as micropricing, which is discussed elsewhere in the literature (Luttrell et al. 2014).

Flotation Release/Kinetics Analysis

Froth flotation separates particles based on differences in surface wettability and not on differences in density. Therefore, the recommended method for determining the washability of fine coal is a technique known as release analysis. Although several versions of this laboratory method exist, the short-form

Table 13 Combined yields and qualities for three size factions of ROM coal separated at (A) constant density and (B) constant cumulative clean coal ash

[1] Size, mm	[2] Size Split %	[3] Feed, stph	[4] Cut Point, SG	[5] Clean Yield %	[6] Clean, stph	[7] Clean Ash %	[8] Clean Sulfur %
A. Operation at identical densities of 1.60 SG for all size classes							
50 × 10	52.09	625.1	1.600	31.90	199.4	11.99	0.79
10 × 1	22.15	265.8	1.600	69.93	185.9	9.22	0.70
1 × 0.15	17.40	208.8	1.600	78.48	163.9	7.66	0.73
Total	**91.64**	**1,200.0**	**—**	**45.76**	**549.1**	**9.76**	**0.74**
B. Select SG to provide the same cumulative ash content from all size classes							
50 × 10	52.09	625.1	1.439	28.13	175.8	9.76	0.67
10 × 1	22.15	265.8	1.744	71.62	190.4	9.76	0.73
1 × 0.15	17.40	208.8	2.106	83.15	173.6	9.76	0.81
Total	**91.64**	**1,200.0**	**—**	**44.99**	**539.8**	**9.76**	**0.74**

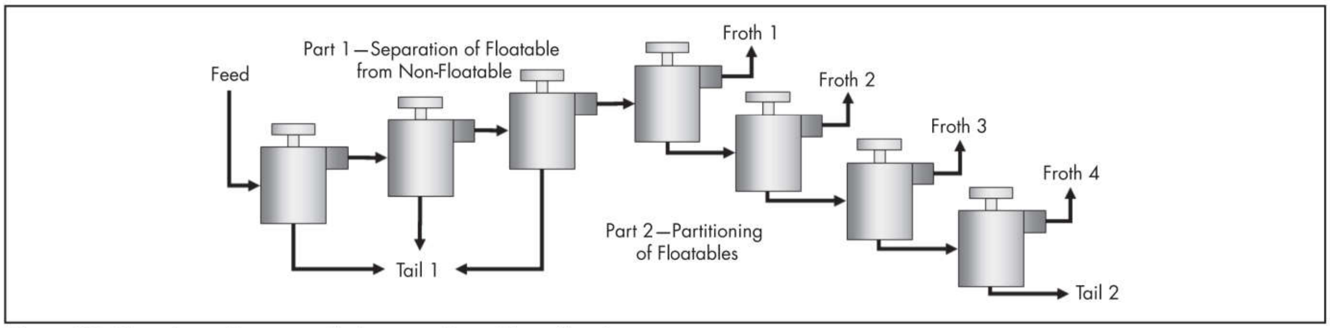


Figure 20 Flotation release analysis procedure (short form)

method is the most popular in the United States. As shown in Figure 20, the short-form method involves two parts. In part 1, feed coal is placed in a laboratory froth flotation cell and floated to exhaustion. The froth concentrate is diluted with fresh water and refloated. This procedure is repeated several times (usually three or four) until essentially no non-floatable solids remain in the flotation cell. All of the tailings from part 1 are combined as the primary tailings (Tail 1). In part 2, the cumulative froth concentrate is floated again using low air rates and impeller speeds. Froth products are taken at intervals that provide roughly the same amount of froth solids in each froth product. This generates several samples of clean coal (Froth 1–4) and a secondary tailing (Tail 2). The data from the release tests is then cumulated as shown in Table 14 under Release Analysis Test. An evaluation of the traditional procedures used for flotation release analysis has been discussed extensively in the literature (Forrest et al. 1994; Mohanty et al. 1998).

In many cases, flotation release tests are performed in conjunction with a timed kinetics test in which froth products are taken at timed intervals, usually ¼, ½, 1, 2, 4, and 8 minutes (see Table 14 under Kinetics Flotation Test). A comparison of flotation kinetics and release data allows designers to ascertain whether column cells equipped with a froth washing system can be justified and to estimate residence time requirements for a given flotation feed slurry.

PERFORMANCE ASSESSMENT AND PREDICTION

Partition Analysis for Density Separations

Data obtained from washability analyses represent ideal separations that cannot be duplicated in industrial practice because of the unavoidable misplacement of particles in real-world separation processes. In the coal industry, this misplacement is commonly quantified by means of a partition curve analysis (Wills 1997). For density separators, partition analyses are often carried out by first collecting representative samples of the feed, clean, and refuse streams around a dense medium or gravity separator. After they are collected, the samples are dried and representative splits are subjected to ash analysis. From the ash values, the ratio (Z) of clean tonnage to reject tonnage is calculated using this equation:

$$Z = (A_r - A_f)/(A_f - A_c) \quad \textbf{(EQ 3)}$$

where A_r, A_f, and A_c, and are the refuse ash, feed, and clean contents, respectively. Next, float–sink tests are performed on representative samples of the clean coal and refuse products over an appropriate range of densities. Columns 1–3 in Table 15 provide an example of the types of information obtained from such an exercise. From these data, the partition factors (P) for the separation can be calculated from the following equation:

$$P = 1/[1 + Z(c/r)] \quad \textbf{(EQ 4)}$$

where c and r are the respective weight percentages of clean coal and reject in each density class. The partition factor represents the probability that a feed particle in a given density class will report to the refuse stream. As shown in Figure 21, the partition curve can now be constructed by plotting the partition factors (column 4) as a function of the mean SG of each density class (column 1). The *cut point* (SG_{50}) is defined as the separating SG that corresponds to P = 0.5. The *steepness* of

Table 14 Comparison of flotation release analysis and timed-flotation kinetic data

	Release Analysis Test						
	Individual		Cumulative Froth		Cumulative Tail		Combined
Float Product	Weight %	Ash %	Weight %	Ash %	Weight %	Ash %	Recovery
Froth 1	18.23	3.09	18.23	3.09	81.77	51.99	31.04
Froth 2	18.23	3.21	36.46	3.15	63.54	65.99	62.04
Froth 3	9.32	5.22	45.78	3.57	54.22	76.44	77.56
Froth 4	7.23	9.07	53.01	4.32	46.99	86.80	89.11
Tail 2	4.23	31.23	57.24	6.31	42.76	92.30	94.22
Tail 1	42.76	92.30	100.00	43.08	0.00	—	100.00
	Kinetics Flotation Test						
	Individual		Cumulative Froth		Cumulative Tail		Combined
Time, minutes	Weight %	Ash %	Weight %	Ash %	Weight %	Ash %	Recovery
0.50	31.30	6.20	31.30	6.20	68.70	58.16	50.53
1.00	15.40	9.23	46.70	7.20	53.30	72.30	74.59
2.00	7.70	11.23	54.40	7.77	45.60	82.61	86.35
4.00	3.80	17.32	58.20	8.39	41.80	88.55	91.76
8.00	1.90	22.30	60.10	8.83	39.90	91.70	94.30
Tail	39.90	91.70	100.00	41.90	0.00	—	100.00

Table 15 Example of partition calculations for a dense medium separator*

[1] Specific Gravity	[2] Clean, wt %	[3] Refuse, wt %	[4] Partition Factor
1.28	13.13	0.00	0.00
1.35	61.48	0.00	0.00
1.43	16.16	0.00	0.00
1.48	5.58	0.01	0.00
1.53	2.44	0.08	0.05
1.58	1.17	0.59	0.43
1.65	0.04	1.17	0.98
1.75	0.00	1.99	1.00
1.90	0.00	3.96	1.00
2.10	0.00	9.03	1.00
2.20	0.00	83.17	1.00
—	100.00	100.00	—

*Given that A_f = 51.69%, A_c = 10.62%, and A_r = 78.89.
Z = (78.89 – 51.69/(51.69 – 10.62) = 0.66

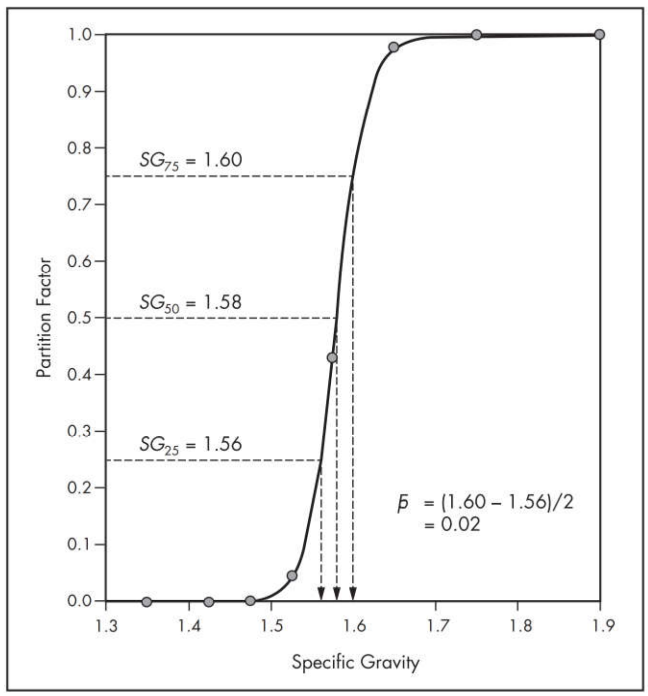


Figure 21 Density separation partition curve

the partition curve, which reflects the sharpness of the separation, is typically defined by an *Ep* value calculated from the following equation:

$$Ep = (SG_{75} - SG_{25})/2 \qquad \textbf{(EQ 5)}$$

where SG_{75} and SG_{25} are the densities corresponding to P values of 0.75 and 0.25, respectively. A lower *Ep* is desirable since it means that fewer particles are misplaced during the separation. For the example shown, SG_{50} = 1.58 and Ep = 0.02. The *Ep* value can be compared with historic values to determine whether the level of separation performance is acceptable. The overall shape of a partition curve also provides operators with a useful diagnostic tool. Ideally, the head and tail of the partition curve should close out at P values of 0 and 1, respectively. Values that do not close out at these upper and lower limits represent gross misplacement of particles that adversely deteriorate the performance of the separation process. This undesirable bypass can often be eliminated through better operating practices, improved maintenance, or upgraded equipment and circuit designs. The extent of coal misplacement is also commonly reported in the coal industry as an *organic efficiency*, which is defined as the actual yield of clean coal divided by the theoretical maximum yield of clean coal that could be achieved at the same ash content according to a washability analysis. Organic efficiencies may be in the high-90 percentiles for well-designed and well-run operations, although lower values are not uncommon for problematic plants.

Table 16 Example of partition analysis for a classifying cyclone bank

[1] Pass, mm	[2] Retain, mm	[3] Mean, mm	[4] Feed, wt %	[5] Oversize, wt %	[6] Undersize, wt %	[7] (u – o)	[8] (u – f)	[9] Partition Factor
8	4	6	14.6	26.5	0.0	–26.5	–14.6	1.00
4	2	3	17.9	32.1	0.4	–31.7	–17.5	0.99
2	1	1.5	10.7	17.2	2.8	–14.4	–7.9	0.88
1	0.5	0.75	16.7	14.4	20.5	6.1	3.8	0.47
0.5	0.25	0.375	14.1	5.1	25.1	20.0	11.0	0.20
0.25	0.15	0.2	8.5	1.8	15.9	14.1	7.4	0.12
0.15	0.045	0.0975	10.4	1.7	21.2	19.5	10.8	0.09
0.045	0	0.0225	7.1	1.2	14.1	12.9	7.0	0.09
—	—	**Totals**	**100.0**	**100.0**	**100.0**	—	—	—

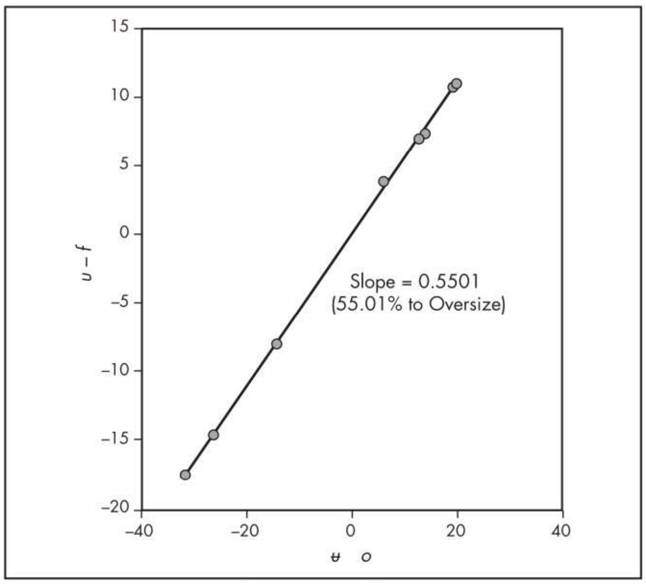


Figure 22 Estimation of oversize split

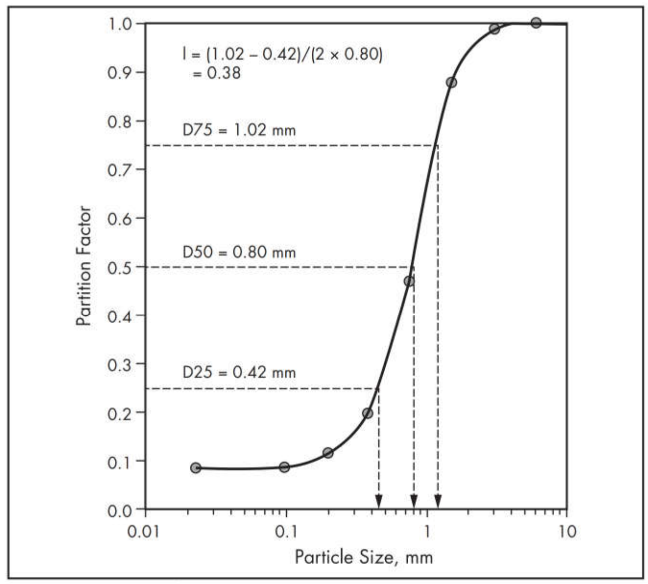


Figure 23 Particle sizing partition curve

Partition Analysis for Size Separations

A different procedure is required for constructing partition curves for screening and classification units since ash values cannot be relied upon for the determination of the Z ratio. For this type of separation, the partition analysis is performed by collecting samples of the feed, oversize, and undersize products from sizing devices such as screens, sieves, or classifying cyclones. Each sample is then sized in the laboratory to determine the weight percentage in each size class. An example of such a data set is provided in Table 16. Columns 7 and 8 are calculated by plotting the differences in undersize minus oversize ($u - o$) versus undersize minus feed ($u - f$). As shown in Figure 22, the slope of the resultant line is the mass yield of product reporting to the oversize, which is 55.01% for the current example. Data for size fractions that produce points that do not fall along the line generally represent data that are not reliable and, as outliers, they should be eliminated from the slope calculation. After the oversize yield has been determined, the oversize partition factor (P) for each size class can be calculated using the following equation:

$$P = Y(o/f) \quad \text{(EQ 6)}$$

where Y is the yield to oversize, and o and f are the respective weight percentages in the oversize and feed. Using this expression, the partition factors for each size class are determined as listed in column 9 of Table 16. Plotting of the mean size (column 3) versus the partition factor (column 9) provides a partition curve for the sizing device (Figure 23). This plot indicates a cut size (D_{50}) of 0.80 mm for this example. The sharpness of the sizing separation is also often determined from the partition curve using the *imperfection* (I) defined as

$$I = (D_{75} - D_{25})/(2 \times D_{50}) \quad \text{(EQ 7)}$$

where D_{75} and D_{25} are the particle sizes corresponding to P values of 0.75 and 0.25, respectively. A lower imperfection is desirable. Typically, imperfection values for classifying cyclones range from 0.20 to 0.80 (Heiskanen 1993), with values of 0.35–0.45 common for raw coal classifying cyclone applications. It is also normal for classifying cyclone partition curves to display a low-tail bypass (B_{10}) because of short-circuiting of finer particles with water to the oversize product. Low-tail bypass values of 5%–15% are not uncommon for raw coal classifying cyclones and may exceed 35%–40% for coal deslime cyclone applications. For the plot provided in

Figure 23, I = 0.38 and B_{lo} = 9%. After this is determined, databases of partition curves such as these can be used to estimate size distributions for oversize and undersize products for different feed coals.

Misplacement in Froth Flotation

Because standard float–sink tests cannot be performed on flotation products, coal operators are forced to use the release analysis procedure to determine the misplacement in froth flotation circuits. While this laboratory technique is normally used to characterize the ultimate separation of feed coals, the procedure can also be used to quantify misplacement in clean coal and refuse samples. For example, Table 17 shows the results of a release analysis test performed on a sample of clean coal from a bank of conventional flotation machines. Although the clean coal contained 9.44% ash, 93.58% of the solids in the froth product contained less than 3.89% ash. The remaining 6.42% of the mass had a very high ash content of 90.37%. The high-ash fraction is typically associated with the hydraulic entrainment of high-ash clay slimes with the water reporting to the froth concentrate. These data suggest that multistage recleaning or column cells equipped with froth washing systems would be a better technology choice for this flotation feed. Typically, well-run column cells will allow less than 1% of the water present in the feed to report to the clean coal product. Release analysis can also be used as a yardstick against which the separation efficiency for flotation can be estimated. The *flotation efficiency* is defined as the actual yield of clean coal in the froth product divided by the theoretical maximum yield of clean coal that could be achieved at the same ash content according to release analysis. Flotation efficiencies are typically lower (80%–90%) than the organic efficiencies observed for well-run dense medium separators.

Table 17 Release analysis on a clean froth product

	Individual, %		Cumulative, %		
Product	Mass	Ash	Mass	Ash	Recovery
Clean 1	26.89	2.01	26.89	2.01	29.10
Clean 2	25.25	2.30	52.14	2.15	56.34
Clean 3	26.73	3.60	78.87	2.64	84.79
Clean 4	13.58	8.15	92.45	3.45	98.56
Tail 2	1.13	39.72	93.58	3.89	99.32
Tail 1	6.42	90.37	100.00	9.44	100.00
Feed	100.00	9.44	—	—	—

Process Simulation

Historic data compiled from partition analyses can also be used to simulate the performance of coal processing circuits (King 1999). This approach is commonly used to evaluate the cleanability of new feed coals in existing plants or to design new

Table 18 Simulation of circuit performance using partition modeling

Input Data

[1]	[2]	[3]	[4]	[5]	[6]	[7]	[8]	[9]	[10]	[11]	[12]	[13]
Specific Gravity Class	Feed, % of Class			Ash, % of Class			Partition to Oversize			Partition to Refuse		
	50 × 10	10 × 1	1 × 0.1	50 × 10	10 × 1	1 × 0.1	50 × 10	10 × 1	1 × 0.1	50 × 10	10 × 1	1 × 0.1
1.26 × 1.30	10.16	39.20	43.05	5.67	5.53	3.51	0.99	0.92	0.07	0.00	0.00	0.00
1.30 × 1.40	16.24	23.06	26.92	11.37	11.33	9.95	0.99	0.92	0.07	0.00	0.00	0.00
1.40 × 1.50	3.20	5.47	6.06	21.68	19.42	19.14	0.99	0.92	0.07	0.05	0.05	0.05
1.50 × 1.60	2.56	2.46	2.45	31.53	28.93	28.14	0.99	0.92	0.07	0.50	0.50	0.50
1.60 × 1.80	2.83	1.79	2.50	40.74	39.07	34.76	0.99	0.92	0.07	0.95	1.00	1.00
1.80 × 2.00	2.60	2.40	1.55	55.90	51.22	52.46	0.99	0.92	0.07	1.00	1.00	1.00
2.00 × 2.40	62.41	25.62	17.47	88.90	86.75	83.93	0.99	0.92	0.07	1.00	1.00	1.00
Total	**100.00**	**100.00**	**100.00**	**62.01**	**30.71**	**22.38**	—	—	—	—	—	—

Output Data

[14]	[15]	[16]	[17]	[18]	[19]	[20]	[21]	[22]	[23]	[24]	[25]	[26]
Specific Gravity Class	Feed, % of Total			Oversize, % of Total			Reject Product			Clean Product		
	50 × 10	10 × 1	1 × 0.1	50 × 10	10 × 1	1 × 0.1	50 × 10	10 × 1	1 × 0.1	50 × 10	10 × 1	1 × 0.1
1.26 × 1.30	5.77	9.49	8.17	5.71	8.73	0.57	0.00	0.00	0.00	5.71	8.73	0.00
1.30 × 1.40	9.23	5.58	5.11	9.13	5.14	0.36	0.00	0.00	0.00	9.13	5.14	0.00
1.40 × 1.50	1.82	1.32	1.15	1.80	1.22	0.08	0.09	0.06	0.00	1.71	1.16	0.00
1.50 × 1.60	1.45	0.60	0.46	1.44	0.55	0.03	0.72	0.27	0.02	0.72	0.27	0.02
1.60 × 1.80	1.61	0.43	0.47	1.59	0.40	0.03	1.51	0.40	0.03	0.08	0.00	0.03
1.80 × 2.00	1.48	0.58	0.29	1.46	0.53	0.02	1.46	0.53	0.02	0.00	0.00	0.02
2.00 × 2.40	35.46	6.20	3.31	35.10	5.71	0.23	35.10	5.71	0.23	0.00	0.00	0.23
Total	**56.82**	**24.21**	**18.97**	**56.25**	**22.27**	**1.33**	**38.89**	**6.97**	**0.31**	**17.36**	**15.30**	**0.31**
		Weight, %	100.00		Weight, %	79.85		Weight, %	46.17		Weight, %	32.96
		Ash, %	46.91		Ash, %	52.62		Ash, %	83.56		Ash, %	10.87

Calculation Legend:

Col[15] = 56.82 × Col[2] Col[16] = 24.21 × Col[3] Col[17] = 18.97 × Col[4] Col[18] = Col[15] × Col[8]
Col[19] = Col[16] × Col[9] Col[20] = Col[17] × Col[10] Col[21] = Col[18] × Col[11] Col[22] = Col[19] × Col[12]
Col[23] = Col[20] × Col[13] Col[24] = Col[18] – Col[21] Col[25] = Col[19] – Col[22] Col[26] = Col[20] – Col[23]

processing facilities based on compiled coal characterization data. The simulation calculations involve applying projected partition factors for particle sizing and density separation to the size-by-size washability for a feed coal sample. The resultant values for the individual size and density classes can then be cumulated to obtain the overall performance in terms of mass yield and product quality. An example of such a series of calculations is provided in Table 18. These calculations assume that the partition curve is independent of the feed coal washability. This assumption is often not valid for water-based density separators in which particle–particle interactions influence the partitioning response (Tavares and King 1995). For these unit operations, the overall shape of the partition curve can also be greatly impacted by water splits and other operational parameters (Rao et al. 2003). The approach can also introduce significant discretization errors if the upper and lower SG ranges for any density class are too large (Mohanta and Mishra 2009). Therefore, considerable care should be exercised when applying partitioning data from one operation to another application. Several software programs are commercially available that can be used for this type of simulation. Commercial programs include Limn, MetSim, and ModSim, as well as custom programs created by a wide variety of individuals and companies (Rong and Lyman 1985; Rong 1992). In most cases, the simulation routines use empirical partition functions that belong to a mathematical family of transition functions (Scott and Napier-Munn 1992; King 2001).

SUMMARY

Coal preparation has played, and will continue to play, an essential role in supplying high-quality feedstocks for coal-fired power stations, metallurgical coking operations, and industrial boiler systems. Modern coal preparation plants incorporate a complex array of solid–solid and solid–liquid separation processes to remove unwanted impurities such as ash, sulfur, and moisture from ROM coals. Separation processes used by these facilities include screening, classification, dense medium separation, gravity concentration, froth flotation, centrifugation, filtration, and thickening. These operations require constant monitoring to ensure that optimal separations are maintained. Monitoring typically involves the routine collection of representative samples from the various plant circuits to ensure that product specifications have been met and that the misplacement of coal and/or rock is maintained within strict limits. Troubleshooting of performance problems often requires additional laboratory analyses that may include particle sizing, float–sink testing, and flotation release analysis. These analytical procedures often make it possible for plant operators to identify inefficiencies that can be corrected through better operating procedures, improved maintenance programs, or redesign of plant equipment or circuitry. In many cases, these improvements generate significant revenue from the recovery of usable coal from waste streams, which provides a strong financial incentive for management to develop and support efficiency monitoring programs.

REFERENCES

Abbott, J. 1982. The optimisation of process parameters to maximise the profitability from a three-component blend. In *Optimisation of Coal Recovery: Proceedings of the First Australian Coal Preparation Conference*. Newcastle, New South Wales: Coal Preparation Societies of New South Wales and Queensland. pp. 87–105.

Arnold, B.J. 1999. Simuation of dewatering devices for predicting the moisture content of coals. *Coal Prep.* 20(1-2):35–54.

Arnold, B.J., Klima, M.S., and Bethell, P.J. 2007. *Designing the Coal Preparation Plant of the Future*. Littleton, CO: SME.

ASTM International. 1994. Standard test method for determining the washability characteristics of coal. In *Annual Book of ASTM Standards*, Vol. 5.05, Standard No. D4371. West Conshohocken, PA: ASTM International.

Bethell, P.J. 2012. Dealing with the challenges facing global fine coal processing. In *Challenges in Fine Coal Processing, Dewatering, and Disposal*. Edited by M.S. Klima, B.J. Arnold, and P.J. Bethell. Englewood, CO: SME. pp. 33–45.

Bethell, P.J., and Barbee, C.J. 2007. Today's coal preparation plant—A global perspective. In *Designing the Coal Preparation Plant of the Future*. Edited by B.J. Arnold, M.S. Klima, and P.J. Bethell. Littleton, CO: SME. pp. 9–20.

Bethell, P.J., and Luttrell, G.H. 2005. Effects of ultrafine desliming on coal flotation circuits. Paper No. 38. In *Proceedings of the Centenary of Flotation Symposium*. Melbourne, Victoria: Australasian Institute of Mining and Metallurgy. pp. 719–728.

Brodzik, P. 2007. Application of Derrick Corporation's StackSizer technology in clean coal spiral product circuits. In *Designing the Coal Preparation Plant of the Future*. Edited by B.J. Arnold, M.S. Klima, and P.J. Bethell. Littleton, CO: SME. pp. 89–96.

Carris, D.M. 2007. A historic perspective. In *Designing the Coal Preparation Plant of the Future*. Edited by B.J. Arnold, M.S. Klima, and P.J. Bethell. Littleton, CO: SME. pp. 3–8.

Chafin, C., Bethell, P.J. DeHart, G., and Corder, R. 2012. Maximizing fine pyrite rejection at the Arch Coal Leer Plant. In *Challenges in Fine Coal Processing, Dewatering, and Disposal*. Edited by M.S. Klima, B.J. Arnold, and P.J. Bethell. Englewood, CO: SME. pp. 67–78.

Davis, V.L., Jr., Bethell, P.J., Stanley, F.L., and Luttrell, G.H. 1995. Plant practices in fine coal column flotation. In *High-Efficiency Coal Preparation*. Edited by S.K. Kawatra. Littleton, CO: SME. pp. 237–246.

De Korte, G.J., and Engelbrecht, J. 2007. Dense medium cyclones. In *Designing the Coal Preparation Plant of the Future*. Edited by B.J. Arnold, M.S. Klima, and P.J. Bethell. Littleton, CO: SME. pp. 61–71.

Doherty, M. 2006. Coal quality impact on unit availability and emissions. Presented at the American Electric Power Site Visit, Asia Pacific Partnership on Clean Development and Climate, Columbus, OH, October 30–November 4.

Firth, B., and O'Brian, M. 2003. Hydrocyclone circuits. *Coal Prep.* 23:167–183.

Forrest, W.R., Adel, G.T., and Yoon, R.-H. 1994. Characterizing coal flotation performance using release analysis. *Int. J. Coal Prep. Util.* 14(1-2):13–27.

Galvin, K.P. 2012. Development of the REFLUX™ Classifier. In *Challenges in Fine Coal Processing, Dewatering, and Disposal*. Edited by M.S. Klima, B.J. Arnold, and P.J. Bethell. Englewood, CO: SME. pp. 159–185.

Galvin, K.P., Doroodchi, E., Callen, A.M., Lambert, N., and Pratten, S.J. 2002. Pilot plant trial of the REFLUX Classifier. *Miner. Eng.* 15(1-2):19–25.

Ghosh, T., Patil, D., Honaker, R.Q., Damous, M., Boaten, F., Davis, V.L., and Stanley, F.S. 2012. Performance evaluation and optimization of a full-scale REFLUX Classifier. In *Proceedings of Coal Prep 2012*. New York: Penton Publishing. pp. 67–82.

Heiskanen, K. 1993. *Particle Classification*. Amsterdam, The Netherlands: Springer.

Hollis, R. 2006. Blue Diamond Coal Company's Leatherwood preparation plant undergoes major circuit modification. In *Proceedings of the 24th International Coal Preparation Conference*. New York: Penton Publishing. pp. 95–112.

Honaker, R.Q., Boaten, F., and Luttrell, G.H. 2007. Ultrafine coal classification using 150 mm gMax cyclone circuits. *Miner. Eng.* 20(13):1218–1226.

King, R.P. 1999. Practical optimization strategies for coal-washing plants. *Coal Prep.* 20:13–34.

King, R.P. 2001. *Modeling and Simulation of Mineral Processing Systems*. Woburn, MA: Butterworth-Heinemann.

Klima, M.S., Arnold, B.J., and Bethell, P.J. 2012. *Challenges in Fine Coal Processing, Dewatering, and Disposal*. Englewood, CO: SME.

Kohmuench, J.N., Mankosa, M.J., Luttrell, G.H., and Adel, G.T. 2002. Process engineering evaluation of the CrossFlow separator. *Miner. Metall. Process.* 19(1):43–49.

Kohmuench, J.N., Mankosa, M.J., and Yan, E.S. 2010. Evaluation of the StackCell technology for coal applications. *Int. J. Coal Prep. Util.* 30(2-5):189–203.

Kohmuench, J.N., Yan, E.S., and Christodoulou, L. 2012. Column and nonconventional flotation for coal recovery—Circuitry methods and considerations. In *Challenges in Fine Coal Processing, Dewatering, and Disposal.* Edited by B.J., Arnold, M.S. Klima, and P.J. Bethell. Englewood, CO: SME. pp. 187–209.

Laskowski, J.S. 2001. Coal flotation and fine coal utilization. In *Developments in Mineral Processing*. Amsterdam, The Netherlands: Elsevier Science Publishing.

Laskowski, J.S., Luttrell, G.H., and Arnold, B.J. 2007. Coal flotation. In *Froth Flotation: A Century of Innovation*. Edited by M.C. Furstenau, G. Jameson, and R.-H. Yoon. Littleton, CO: SME.

Luttrell, G.H., Kohmuench, J.N., Stanley, F.L., and Trump, G.D. 1998. Improving spiral performance using circuit analysis. SME Preprint No. 98-208. Littleton, CO: SME.

Luttrell, G.H., Honaker, R.Q., Bethell, P.J., and Stanley, F.L. 2007. Design of high-efficiency spiral circuits for coal preparation plants. In *Designing the Coal Preparation Plant of the Future*. Edited by B.J. Arnold, M.S. Klima, and P.J. Bethell. Littleton, CO: SME. pp. 73–87.

Luttrell, G.H., Noble, A., and Stanley, F. 2014. Micro-price optimization of coal processing operations. SME Preprint No. 14-127. Englewood, CO: SME.

Majumder, A.K. 2011. Processing of coal fines in a water-only cyclone. *Fuel* 90(2):834–837.

Meenan, G. 2005. Implications of new dewatering technologies for the coal industry. Invited lecture at the Center for Advanced Separation Technologies (CAST) Workshop, Virginia Tech, Blacksburg, VA, July 26–28.

Mohanta, S., and Mishra, B.K. 2009. On the adequacy of distribution curves used in coal cleaning: A statistical analysis. *Fuel* 88:2262–2268.

Mohanty, M.K., Honaker, R.Q., Patwardhan, A., and Ho, K. 1998. Coal flotation washability: An evaluation of traditional procedures. *Int. J. Coal Prep. Util.* 19(1-2):24–32.

Mohanty, M.K., Palit, A. and Dube, B. 2002. A comparative evaluation of new fine particle size separation technologies. *Miner. Eng.* 15:727–736.

Mohanty, M.K., Zhang, B., and Geilhausen, R. 2009. *On-Site Evaluation of the Stack Sizer in Conjunction with the Falcon Concentrator.* Final technical report to Department of Commerce and Economic Opportunities/Illinois Clean Coal Institute, Project No. 07-1/9.1B-2, April.

National Research Council. 2002. *Coal Waste Impoundments: Risks, Responses, and Alternatives.* Washington, DC: National Academies Press.

Osborne, D. 2012. Milestones in fine coal cleaning development. In *Challenges in Fine Coal Processing, Dewatering, and Disposal*. Edited by M.S. Klima, B.J. Arnold, and P.J. Bethell. Englewood, CO: SME. pp. 3–32.

Rao, B.V., Kapur, P.C., and Rahul, K. 2003. Modeling the size-density partition surface of dense-medium separators. *Int. J. Miner. Process.* 72(1-4):443–453.

Rong, R.X. 1992. Optimization of complex coal preparation flowsheets. *Int. J. Miner. Process.* 34(1-2):53–69.

Rong, R.X., and Lyman, G.J. 1985. Computational techniques for coal washery optimization—Parallel gravity and flotation separation. *Coal Prep.* 2:51–67.

Schultz, W., Jahnig, R., Bratton, R.C., and Luttrell, G.H. 2008. Operating and maintenance guidelines for screenbowl centrifuges. *Proceedings of the 25th Annual International Coal Preparation Exhibition and Conference*. New York: Penton Publishing. pp. 169–177.

Scott, I.A., and Napier-Munn, T.J. 1992. Dense-medium cyclone model based on the pivot phenomenon. *Trans. Inst. Min. Metall.* 101(January-April):C61–C76.

Skiles, K.D. 2003. Search for the next generation of coal flotation collectors. *Proceedings of Coal Prep 2003, 20th Annual International Coal Preparation Exhibition and Conference*. New York: Penton Publishing. pp. 187–204.

Tavares, L.M., and King, R.P. 1995. A useful model for the calculation of the performance of batch and continuous jigs. *Coal Prep.* 15:99–128.

Wills, B.A. 1997. *Mineral Processing Technology*. 6th ed. Oxford: Butterworth-Heinemann.

Yoon, R.-H., Luttrell, G.H., Adel, G.T., and Mankosa, M.J. 1992. The application of Microcel column flotation to fine coal cleaning. *Coal Prep.* 10:177–188.

Zhang, B., Yang F., Sahoo, P.K., Mohanty M.K., and Zhang, X. 2008. Optimization of a compound spiral for cleaning high sulfur fine and ultrafine coal. *Proceedings of the XXIV International Mineral Processing Congress*. Vol. 2. Beijing: Science Press. pp. 653–662.

Ziaja, D., and Yannoulis, G.F. 2007. Is there anything new in coarse of intermediate coal cleaning? In *Designing the Coal Preparation Plant of the Future*. Edited by B.J. Arnold, M.S. Klima, and P.J. Bethell. Littleton, CO: SME. pp. 43–59.