CHAPTER 6.2

Gravity Concentration

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Gravity separation is second only to hand sorting in being the oldest of methods used to concentrate minerals. Agricola recorded the use of jig screens and sluices for the separation of heavy metals in the 16th century (Hoover and Hoover 1950). The industrial revolution, which occurred from 1760 to 1840, resulted in a demand for large-scale production of minerals and coal, thereby fueling the ingenuity and energy of innovators to develop density-based separators that were used to recover minerals from relatively rich ore deposits. The modern-day separators such as continuously operating jigs and shaking tables were developed, patented, and commercialized in the late 19th century. Inventions that provided enhanced gravity separations in a mechanically applied centrifugal field were patented in the 1890s and served as the basis of recently commercialized technologies.

The early 20th century brought continued development of high-capacity, density-based separators for coarse particle concentration, such as the Baum jig and the Chance cone. Several dry separation technologies were developed and commercialized from 1920 to 1940, mainly for the treatment of coal. As the ore bodies became lower grade and underground mechanized coal mining became more prevalent, development activities shifted somewhat to technologies capable of treating fine particles (1–0.15 mm). Flowing film separators, such as the spiral concentrator and Reichert cone concentrator, were introduced and realized wide commercial use in the middle part of the 20th century. In addition, hydraulic fluidized-bed classifiers used for particle size separations were proven to provide effective density-based separations for a wide variety of fine materials when operated under certain conditions.

The late 20th century was the era of enhanced gravity separator (EGS) development, because of the desire to provide low-cost, highly efficient recovery of ultrafine heavy metals such as gold and tin. Mechanisms commonly employed in conventional technologies such as jigging, flowing film sluices, fluidized beds, and shaking tables were applied in devices that achieve separation in a centrifugal field created by mechanical action. The EGS units have been commercially applied to the recovery of a wide variety of different metallic minerals present in particle sizes as fine as 10 µm. Burt (1984, 1986, 1999) provides a comprehensive historical overview and description of the various density-based separators that have been used by mineral processors since the early 1900s.

HINDERED SETTLING CONCENTRATION

In the free settling of mineral particles in a liquid, the falling particles are at a distance from each other so that no particle is affected by its neighbor. In hindered settling, the concentration of particles is sufficiently high so that each particle is affected by its proximity to other particles in the suspension. Richards and Locke (1940) described the hindered settling phenomenon as the condition "where particles of mixed sizes, shapes and densities in a crowded mass, yet free to move along themselves, are sorted in a rising current of water, the velocity of which is much less than the free-falling velocity of the particles but yet fast enough so the particles are in motion." This is the condition normally encountered in mineral concentration processes.

For free settling of coarse particles (+2 mm), the Newton equation (Symonds 1986; Wilson et al. 2006) applies, whereas for the settling of fine spheres (~106 µm) in water, the Stokes equation applies (Batchelor 1967). For particles whose size lies between ~2 mm and 106 µm, their settling velocity can be determined from experimental data. These data are available in convenient form in the text by Taggart (1951). Alternatively, a Reynolds number coefficient-of-resistance plot may be used to determine the settling rate of such particles (Gaudin 1939).

Particle shape affects the settling rate of both coarse and fine particles. The general effect is to reduce their settling velocities, and the effect is greater for coarse particles and for those settling under hindered settling conditions than for fine particles or free-settling ones. However, nearly all gravity concentration processes (jigs, tables, flowing film concentrators, heavy media separators) and many sizing devices (sizing classifiers, clarifiers, thickeners, hydroseparators) make use of the hindered settling phenomenon.

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Hindered Settling Separators

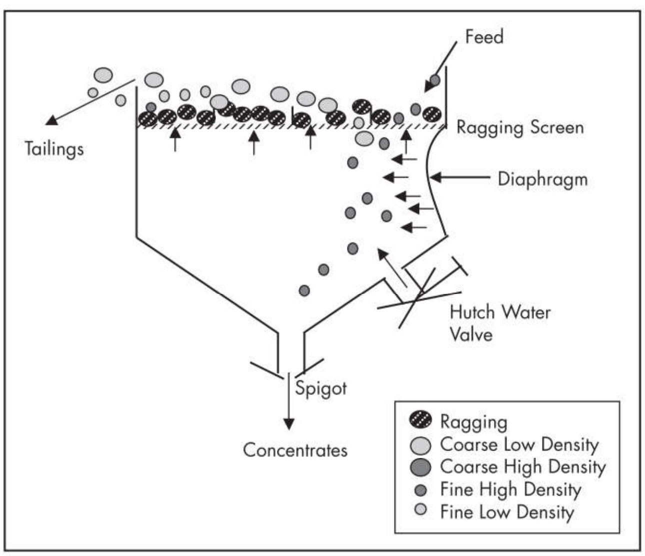
Jigs

Jigging is an ideal preconcentration process, being relatively inexpensive in construction, operation, and maintenance, and relatively unaffected by feed grade. All jigs utilize a screen (the older-type jigs have a fixed screen while the more modern jigs have a moving screen) and a means to provide pulsations through a particle bed (ragging), which results in particle separation on the basis of density by hindered settling and consolidated trickling (Figure 1). Heavy mineral particles pass through the screen while light particles overflow a weir.

The fundamental principles of all jigs are essentially the same. The basic differences between the various types of jigs are a matter of practical engineering to optimize the operating performance, materials handling, maintenance, and control. The basic design features (Taggart 1945, 1951) of a jig are

- Screen to support the mineral bed,
- Hutch or tank containing the liquid beneath the screen,
- Means of creating a jig stroke or relative motion between the liquid and the bed,
- Method of modulating the jig-stroke waveform,
- Method of regulating the upflow of water,
- Method of supplying feed to the bed, and
- Method of removing products from above the screen and from the hutch.

Ragging is a layer of large, heavy particles at the bottom of the bed and on the jig screen, as shown in Figure 1. The ragging controls the rate at which the heavy fine particles penetrate and percolate through the bed to the hutch. For some ores there is enough coarse heavy mineral to provide this layer. However, with many ores it is necessary to provide an added layer of coarse heavy material, which is called ragging. In general, the ragging particles must be heavy enough to remain at the very bottom of the bed, but light enough for dilation on the upstroke. The particle size must be greater than the screen openings and large enough to provide spaces between the particles to allow concentrate particles to percolate through on the downstroke. Harrison (1962) recommends a ragging particle diameter of four times that of the maximum particle size of the hutch product.



Adapted from Falconer 2003

Figure 1 Generic representation of a fixed-screen jig

For mineral jigs, capacity is heavily dependent on the average density of the feed. This determines the volume of material (dry basis) that is fed to the jig. Jig design also influences capacity, as circular jigs have a higher unit capacity than rectangular jigs.

In any jig application, the size distribution and the density of the particles of the ore will result in a unique situation requiring optimization of each of the jig operating parameters. To concentrate fine heavy particles, the suction phase must be augmented, whereas for coarse heavy particles, the pulsion phase is the most important. These opposed considerations give rise to much of the divergence of opinion about optimizing stroke speed, amplitude, and modulation, and with jig design in general.

The following are some of the major adjustment factors for jig operation:

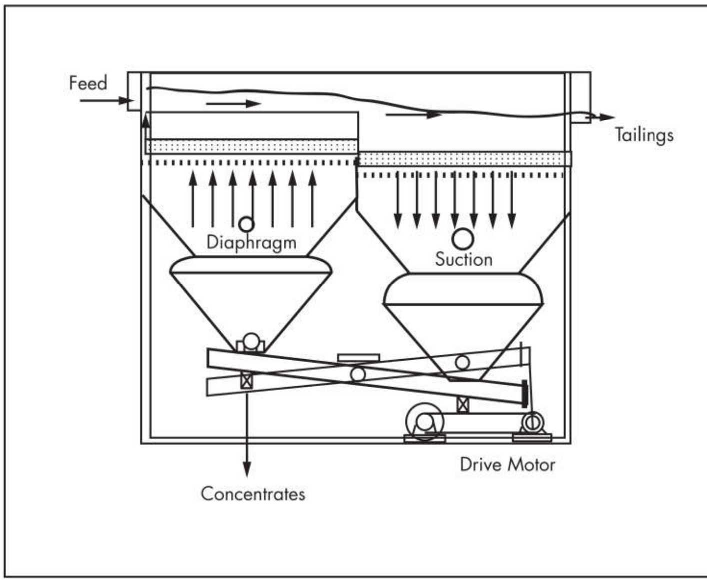
- **Stroke speed and length influence the water waveform across the screen.** Balancing the stroke speed and length is an important aspect leading to good or poor performance.
- **Water flow.** The jig stroke is modulated by the water upflow. Too much hutch water will result in fine particle values to the tailings. Too little hutch water will dilute the hutch concentrate with non-valuable particles.
- **Ragging.** Choosing the correct type and size of ragging is important to get the desired cut point for separation. Ragging size also limits the size of the coarse material that will pass through the jig screen.
- **Jig slope.** The bed must be sloped from feed to tailings discharge to facilitate transport of solids in the bed. Jig design must allow for adjustment of this slope during installation.
- **Jig area.** The area determines the capacity of the jig.

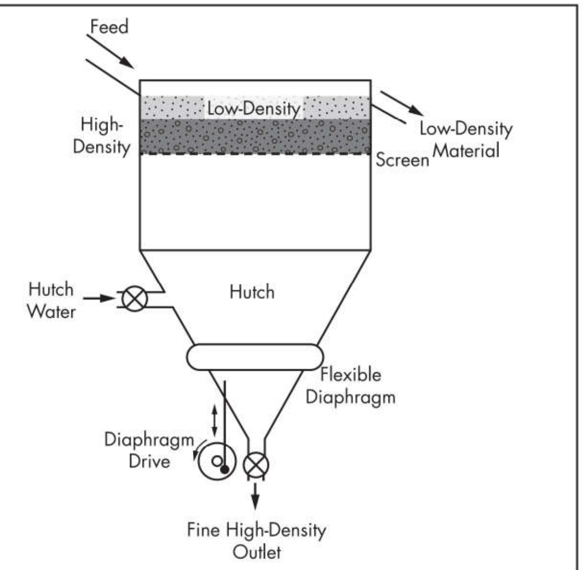
Pan-American jig. The most common mineral jig in use today is the Pan-American style, although some Yuba and Denver jigs can still be found. The Pan-American (often called just Pan-Am) jig uses a flexible diaphragm in the cone section of the jig, directly below the fixed screen, which is manipulated by a mechanical arm mounted eccentrically to provide an up-and-down movement of the cone and thus a pulsation source to the particle bed (Figure 2). Hutch water is added to reduce the speed of the suction stroke and allow additional time for particle stratification based on density.

The typical Pan-Am jig is normally built as a duplex jig (a pair of balanced jig cells). Each cell consists of an upper hutch (usually rectangular) and a lower hutch (usually conical), and it is joined by an annular diaphragm of flexible rubber to allow up-and down movement of the lower hutch. The standard Pan-Am jig consists of two cells (each 1 m × 1 m) with a common drive for energy savings and the addition of 193 kilos of 4.75-mm steel shot per cell as ragging. It has a stroke length of 19–38 mm and frequency of 20–200 cycles per minute.

Pan-Am duplex jigs are popular in many regions and are operated in Alaska (United States), Yukon (Canada), South America, Africa, and Asia. Much of this popularity is due to their inherent simplicity and relative ease of maintenance.

IHC jig. The IHC circular jig is shown in Figure 3. A trapezoidal-bed diaphragm mechanism provides a sawtooth pulse pattern. The jig can handle a broad particle range from





Source: Coggin 1995

Source: Honaker et al. 2014

Figure 2 Pan-American jig

Source: IHC Technical Services, n.d.

Figure 3 IHC circular jig

~50 µm to 12 mm. Capacities of the different sized jigs vary from 260 m^3 to 1,250 m^3 of slurry per hour. Freshwater requirements are ~14 to 336 m^3/h. The power requirements for the jig range from 7.5 kW to 49 kW (IHC Technical Services, n.d.).

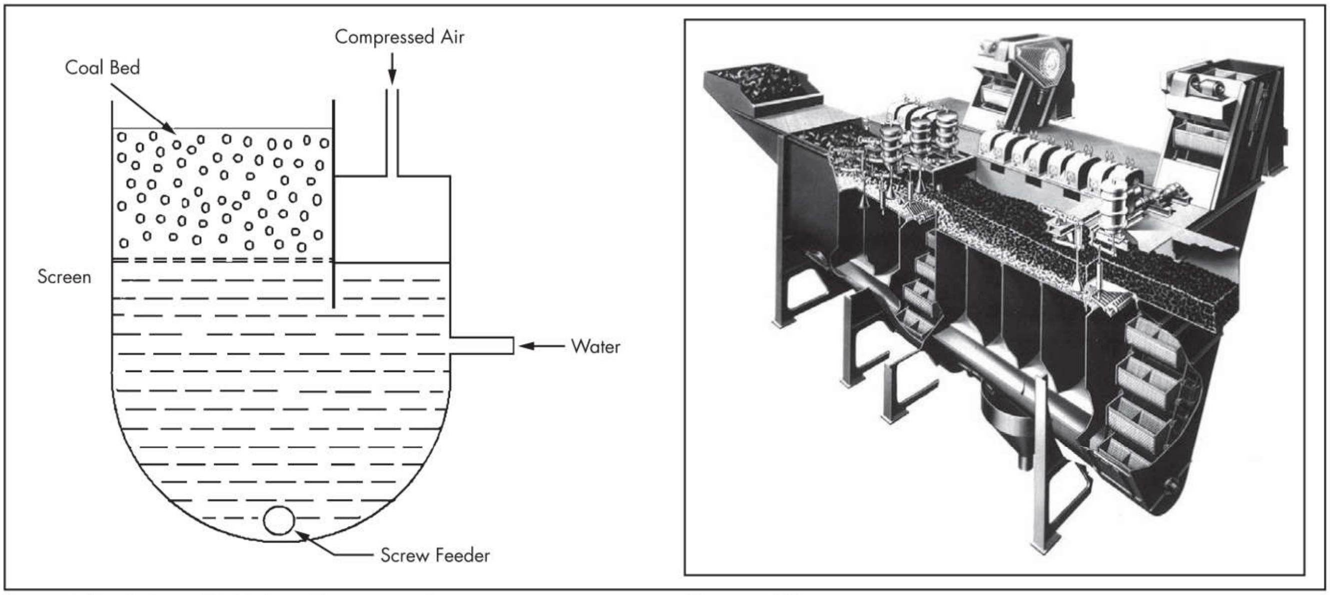
The IHC jig employs a sawtooth pulsation pattern that reduces the hutch water requirement. The sawtooth movement consists of a fast upward and a slow downward stroke. The aim of the fast upward stroke of short duration is to prevent the loss of fine heavy material. Each system has its own features; deciding which system to use depends on the number of modules used in an installation. The downward stroke, also called the suction stroke, is much longer. During this phase, the fine particles are drawn into the bed. IHC Merwede has developed two types of jig drives, a mechanical drive and a mechanical-hydraulic drive. Both drives are designed to give reliable performance under the conditions that prevail on mineral dredgers and on mines. Two jig plant options are available:

1. **Standard design (SD) jig series.** The SD jig installation has a modular construction, which makes it a dismountable processing plant and easier to transport. Depending on the required capacity, modules can be combined up to a maximum of twelve. For smaller capacities a mini, micro, and super micro module are available.
2. **Skid frame (SF) jig series.** The SF series of jigs are mounted on skid frames, used for onshore installations. These jig installations are standardized to various capacities. Similar to the SD jig installation, the SF jig is also of modular construction.

Baum jig. The Baum jig, developed in the late 1890s, was a significant development with the replacement of conventional plungers in jigs at that time by compressed air and exhaust through the use of valves (Figure 4). The later improved control of the pulse cycle in larger compartments allowed for more-efficient separation over a wider range of particle sizes at higher throughput capacities (Sanders et al. 2002). The Baum jig became the most popular unit for coal beneficiation in the late 20th century with bed widths of ~2.5 m. The jig pulse cycle is typically sinusoidal, and substantial research has been undertaken to optimize the waveform to improve jig performance (Tanaka et al. 1990; Iijima et al. 1998).

Baum jig capacity is usually 29–59 t/m^2/h of active screen area, with machine capacities ranging from 23 t/h to 635 t/h. The water requirement is usually 3.8–9.5 m^3/min but can be kept at a minimum by operating controls and recirculation. Water is split 30% to the feed, and the remainder to the cells. Because of density and viscosity effects, it is not advisable to operate with recirculated water with densities over 1.04 or if the feed contains more than 15% clay or suspended solids by weight (Leonard and Mitchell 1968; Price and Bertholf 1959). The power requirement is approximately 0.8 kW/m^2 of screen area, although this may be exceeded if the jig is not operated at the correct frequency. Compressed air supply is only 21 kPa.

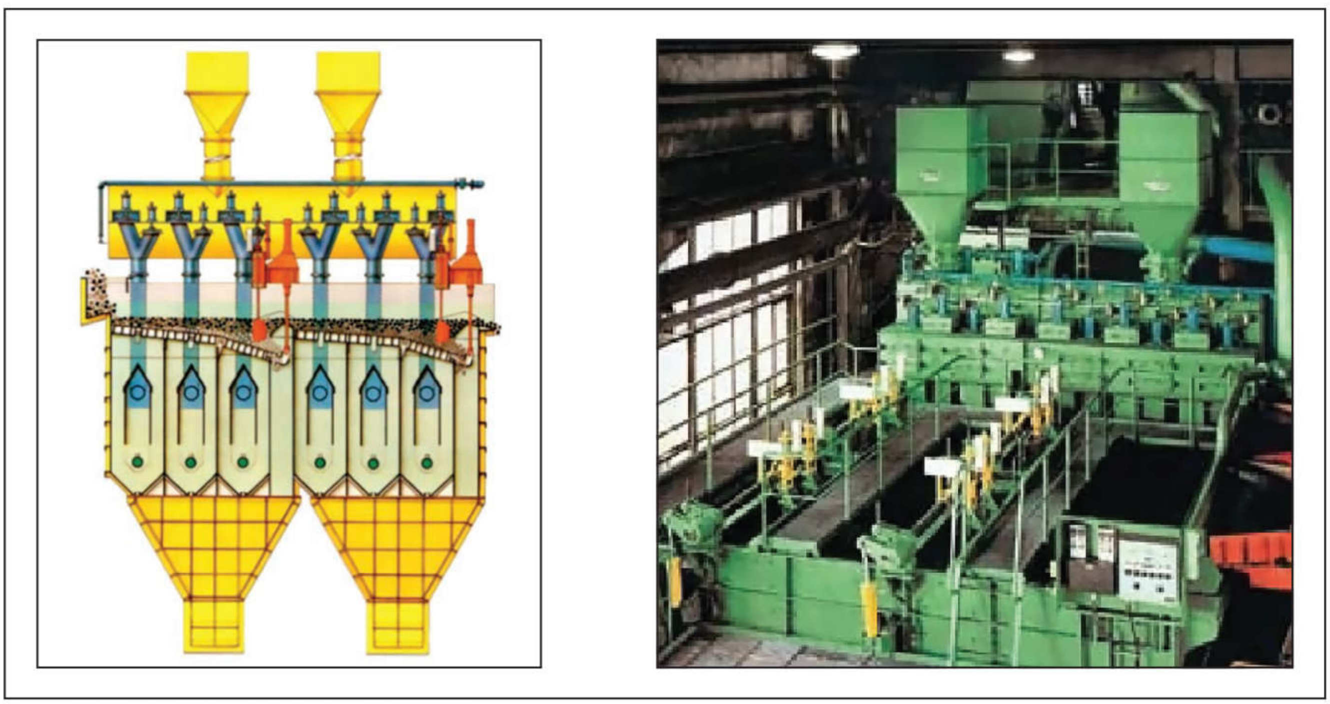
Operating controls are necessary to control bed density and level, and these are usually achieved by varying the rate of product removal. Level sensors are usually of the balanced-float-compartment type. Some jigs have submerged streamlined floats in the bed to sense density and to automatically control either the air entrance or exit to modify the jig



Source: Indian Institute of Technology Kharagpur, n.d.

Courtesy of Metso

Figure 4 Baum jig (left) and McNally Baum-type jig (right)

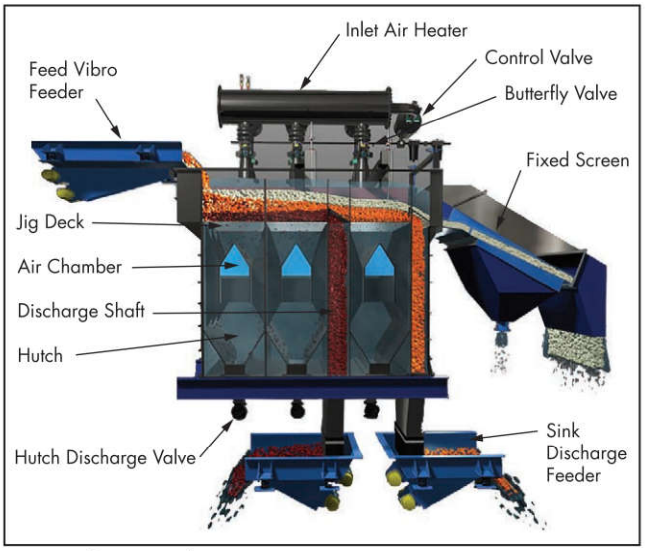


Source: MBE Coal and Minerals Technology GmbH 2011

Figure 5 Batac Jig

stroke. In standard plant practice, at least two jigs are operated in series, the first making a finished clean product and the second making a middling and tailing product. The middling product may be crushed and treated in another jig or by other methods.

Batac jig. The demand to increase throughput that was not achievable in a Baum jig, given the difficulty of generating a uniform bed pulse across a large particle bed, resulted in the development of the Batac jig (Figure 5). The jigging motion in the Batac jig is generated in air chambers located underneath the jigging bed. Low pressure, high-volume air from a blower is intermittently supplied to these air chambers and then discharged by means of an electronically controlled plate valve system. The frequency and profile of the stroke can be easily modified using the jig's programmable logic controller. The stroke is imparted to the water inside the jig as a function of the pressure change generated inside the air chambers. Makeup water is added at the lowest point of every jigging



Courtesy of Tenova Delkor
Figure 6 Apic jig

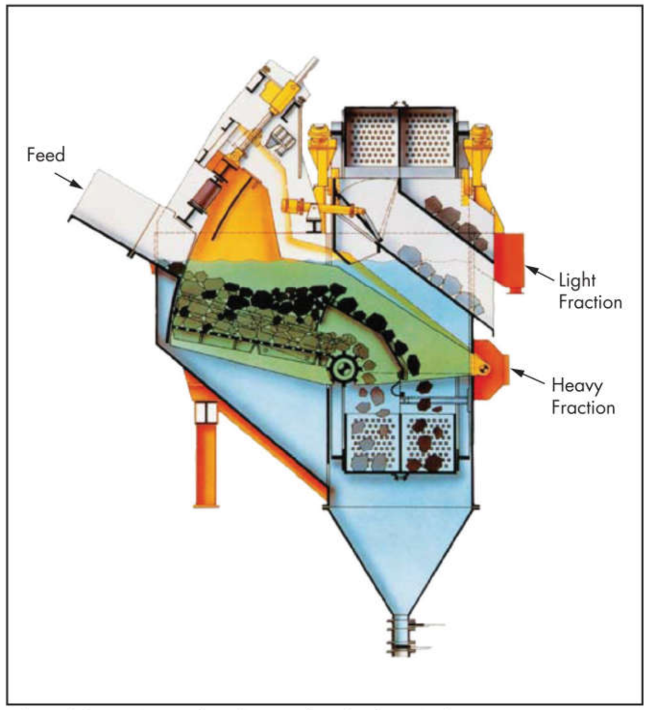
chamber to intensify the upward current and to restrain the downward current. The pulsating motion of the water stratifies the feed material according to its density (Takakuwa and Matsumura 1954). As a result, very efficient separations have been realized in industrial units up to 7 m wide.

The benefits of the Batac jig have been summarized by Sanders et al. (2002). Batac jigs are usually used to treat coal with a size range from 60 to 6 mm. However, the jig is capable of handling run-of-mine coal with a top size up to about 150 mm. Treatment rates range from 68 to 90 t/h/m jig width.

A short Batac jig, which is a jig with a length of only two or three chambers, can be used to de-stone raw coal ahead of a dense medium circuit. The key difference between a Batac jig for de-stoning applications and a Romjig is the size range of the material to be beneficiated.

Apic jig. The Apic jig, shown in Figure 6, is a relatively recent advancement in under-pulsed jigs like the Batac jig. The Apic jig provides enhanced control of the airflow into and out of the chambers and improved gate control for removing low-density particles (Loveday and Jonkers 2002). The Apic gate, a vertical-shaft sink discharge mechanism, added to the advantages of the jig, because it eliminates the vertically operated or conventional discharge mechanism, thereby minimizing the risks of low-density material short-circuiting and in the mixing of bed layers. The other advantage is unhindered smooth discharge of heavy product.

The underbed air pulsation of the water in the jig submits the bed of particles on the screen deck to vertical fluid pulses of extension and compression phases and results in densimetric stratification of the particles. The denser, heavier particles (sinks) settle, and the lighter, less-dense particles (floats) rise to the top of the material bed. The number and widths of the compartments in the jig are selected to achieve the most efficient separation. Where necessary, a ragging bed is installed for the recovery of fine material. The discharge mechanisms of the jig permit a smooth and accurate evacuation of sinks and middlings. The gate discharge is controlled by a float that is continually positioned at the desired density interface within the bed. The gate design is chosen to suit the particular feed size distribution and the proportion of sinks.



Adapted from MBE Coal and Minerals Technology GmbH 2011
Figure 7 Romjig

The pulse characteristics are designed to maximize the separation of products in the jig, and electronic control provides a precise and consistent operation regardless of feed variations. The pulse timings are automatically adjusted for bed-level changes. Each compartment is timed and adjusted independently. The inlet air pressure is automatically regulated to provide the correct pulse for the application. The pulse downstroke can also be regulated to control the suction stroke and thus the settling characteristics.

Romjig. Beneficiation of coal close to the mining face to remove coarse rock from coal was the subject of extensive studies by German researchers in the 1980s. The objective was to reduce the cost and increase production of underground operations (Sanders et al. 2000). The result was the development of the Romjig, shown in Figure 7. The Romjig uses a moving jig screen, the jigging motion provided by a mechanical hydraulic arm. In this way, the feed end of the screen plate moves up and down while the discharge end is pivoted. The lifting cycle is repeated 38–43 times per minute. During the jigging action the material separates, with the heavy particles collecting next to the screen and progressively forming a bed as the material moves along the screen, while the lighter coal moves to the top of the bed. The jig plate is a screen panel with 15-mm slotted openings.

Three factors ensure that the material moves horizontally from the feed end to the discharge end:

1. Slope of the screen deck (pressure from the feed)
2. Linear downstroke
3. Circular upstroke

Table 1 Romjig details

Construction	Units	Process Details	Units
Length	~6.0 m	Feed rate	350 t/h
Width	~6.5 m	Feed size	350 –40 mm
Height	~8.0 m	Stroke amplitude	500 mm
Bed width	~2.0 m	Stroke frequency	38–43/min
Full weight	98 t	Makeup water requirement	10–15 m^3/h
Installed power	110 kW	Specific energy	0.3 kW·h/t
		Cut-point range	1.6–2.1 relative density

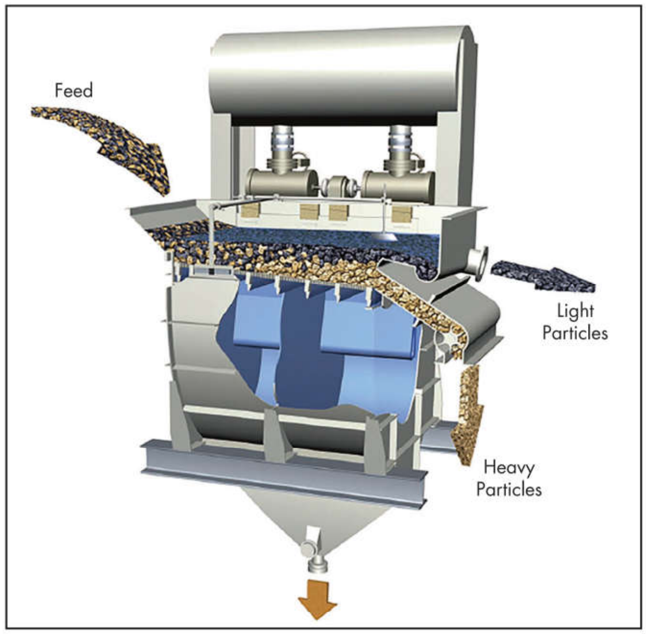
Source: Ziaja and Yannoulis 2007

Products are collected in a divided discharge wheel with one compartment for the coal and one for the discard. Particles smaller than the screen panel apertures fall through into the hutch compartment. Material accumulates in the hopper at the bottom and is discharged periodically through a double-gate valve system to join with the –40-mm feed screen underflow material for further treatment or as final product.

To ensure that design and manufacturing remain competitive, the Romjig is available in only one size, with a 2.0-m-wide bed, rated at up to 350 t/h for a nominal 350 –40 mm feed. Details of the construction and operational design are summarized in Table l. The metallurgical performance of a Romjig is discussed in the paper by Sanders and Ziaja (2003).

Alljig. The Alljig is similar to the Baum jig, with the addition of a control system for discharge, enabling the machine to work autonomously once the control system is adjusted (Figure 8). Since its introduction in the late 1980s, the Alljig has seen many improvements, enabling it to process a growing array of different materials. Electronic sensors are used to automatically monitor and control the precise discharge of heavy particles contained in the feed. Alljig jigging machines create a physically stable and individually adjustable optimal jigging stroke at minimal energy consumption by means of air-pulsed water. These machines provide capacities ranging from 5 t/h to 700 t/h. The process of efficient separation and cleaning of feed material is applicable to particle sizes from 150 mm to <1 mm.

Gekko inline pressure jig. The inline pressure jig (IPJ) is a single-hutch circular jig that is completely enclosed to operate at greater than atmospheric pressures. The IPJ components are shown in Figure 9. An internal screen is pulsed by a hydraulic ram by means of a shaft that extends through the bottom of the cone and is sealed by a diaphragm. The screen frame is sealed to the enclosure by a flexible diaphragm. The IPJ is fed through the center and under pressure to a distributor. The feed is distributed throughout the screen, with the heavy minerals passing through the screen and the light minerals moving to the top and continuously to the tailings outlet. The vessel is completely full of water or slurry. The screen is pulsed by means of the hydraulic ram and the pulse is a sawtooth pattern. Ragging material is on the screen. Hutch water is added at the bottom of the concentrate hutch. The completely submerged feed aids in mineral separation and reduces the hutch water requirement. The quality of hutch water is not an issue, and water of almost any quality, including seawater or water with up to 5% solids, is suitable. The IPJ can recover minerals from 100 μm to 5 mm in size. Because it operates under pressure, it can be incorporated in a cyclone feed line or other pressurized systems. Industrial applications of the jig are provided by Gray (1997).



Source: Allmineral 2013
Figure 8 Alljig

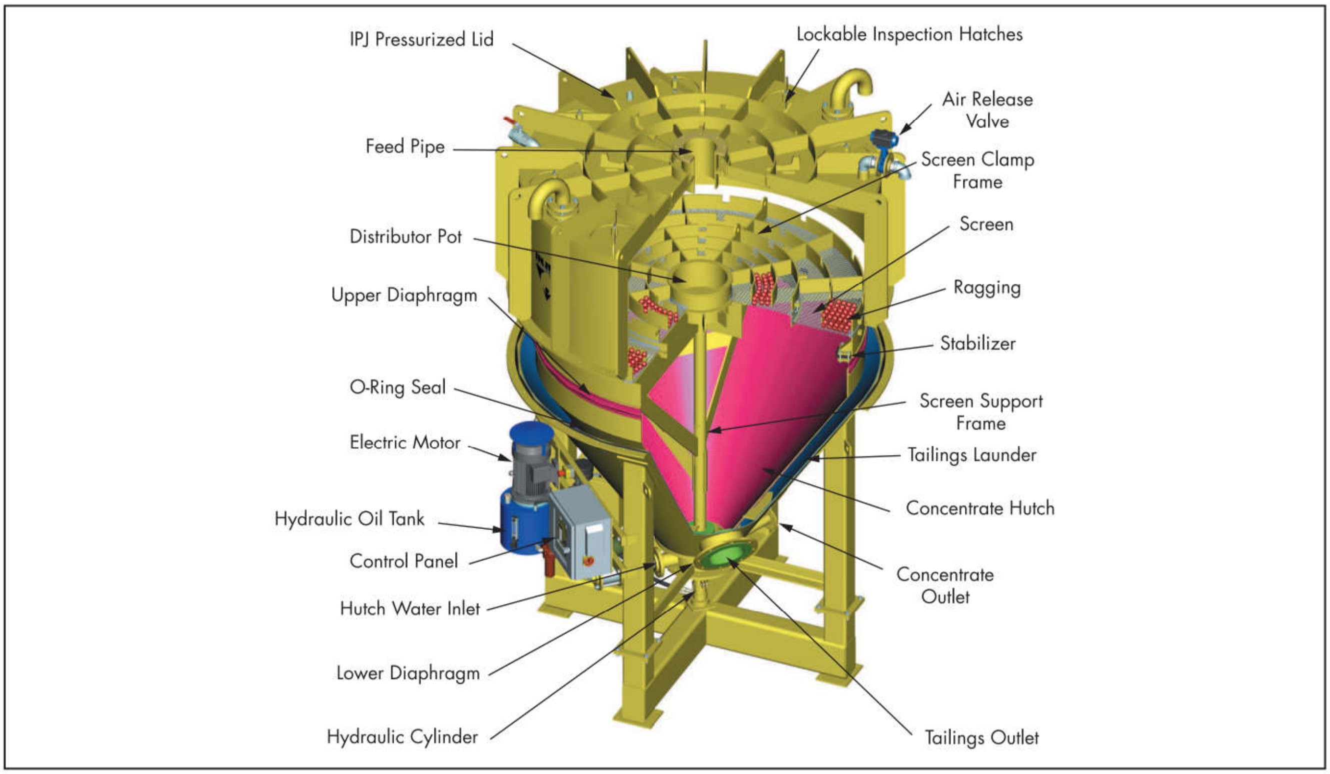
The IPJ includes the following operating variables:

- **Feed rate.** The feed rate will ultimately dictate unit performance. Too high a feed rate will impact recovery, though this is mitigated by a higher specific-gravity difference between gangue and valuable mineral.
- **Screen and ragging.** Screen aperture should be slightly greater than the largest particle fed and is generally of wedge-wire design. The ragging material can be natural minerals or steel, lead shot or ceramic beads. Consistent ragging size and shape will enhance unit repeatability and ultimately performance. Synthetic ragging material made from polyurethane or ceramic is available from Gekko.
- **Pulse.** Both stroke length and frequency of stroke, the upstroke and downstroke, can be controlled independently. The upstroke is generally slower than the downstroke, creating a sawtoothed wave profile. This wave profile causes increased acceleration of higher specific gravity particles and thus improves the separation.
- **Hutch water.** Adding water creates a positive flow through the screen bed, aids in concentrate flow, and reduces the suction created during the upstroke. Too low a water flow will produce a low-grade concentrate and cause excess force on the machine.

The IPJ is available in four sizes, with specifications as shown in Table 2.

SLUICES

Sluices can be used to make a rough gravity concentration, provided the valuable mineral is free, not too fine, and possesses a fairly wide size range. Sluice boxes can provide a much higher concentration ratio than most other gravity concentrators. They are also very reliable, inexpensive, and simple to operate, which explains why the sluice box is still the most important placer gold concentrator (Clarkson 1990). The use of sluices has diminished greatly over the last three



Courtesy of Gekko Systems

Figure 9 Inline pressure jig showing individual components

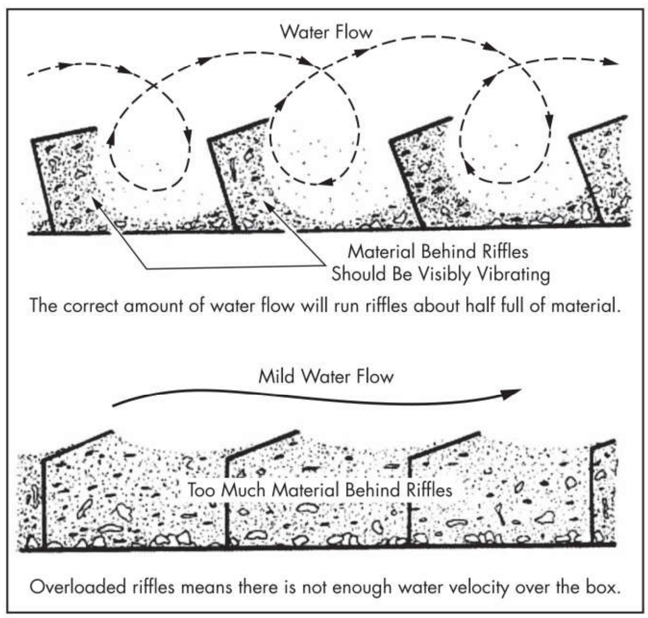
Table 2 Specifications of inline pressure jigs

	Model Number			
Specification	**600**	**1000**	**1500**	**2400**
Maximum feed rate, t/h	1.5–3	15–25	35–50	80–100
Maximum particle feed size, mm	6	20	20	50
Hutch water flow rate, m^3/h	2.0	10–20	15–35	15–50
Typical yield to sinks/concentrate, t/h	0.6	1–4	5–8	5–40
Overall height, mm	1,300	2,105	2,500	3,200
Footprint diameter, mm	800	1,350	1,800	2,500
Screen diameter, mm	520	1,030	1,330	2,000

Courtesy of Gekko Systems

decades, more so in the tin industry (Min 2006; Hutahaean and Yudok 2013); jigs (Pan-Am and the circular jig) are preferred, especially when using a dredge for mining.

A sluice box is an inclined rectangular flume containing riffles on matting, through which a dilute slurry of water and alluvial gravel flows. Sluice boxes operating under ideal conditions are actually centrifugal concentrators whose riffles overturn ribbons of slurry to form vortices (Figure 10). At the bottom of these vortices, centrifugal and gravitational forces combine to drive placer heavy particles into matting, breaking up clumps of clay and cemented particles. Effective recovery of fine minerals depends upon a fairly loose, active bed of sand between the riffles. At the same time, the rapidly moving coarse gravel above the riffles must not interfere with the access of sand to the riffles or cause too much disturbance. Generally, a smooth flow represents a hard bed and poor recovery.



Source: McCracken 2013

Figure 10 Formation of vortices between the riffles of a sluice box

Types of Sluice Boxes

The appropriate selection of material for a sluice box is fundamental to the successful recovery of gold particles of different sizes and shapes.

Courtesy of Madden Steel
Figure 11 Large sluice box for gold (~5 × 12 m)

Courtesy of Royal Manufacturing
Figure 12 Small sluice box for miners

- **Standard sluice box.** Sluices come in all sizes depending on throughputs, for single operators to large-scale mining, as shown in Figures 11 and 12. The sluice length can vary from around 1.5 m to more than 30 m (nominally 10 m) and principally depends upon the character of the material to be treated. Coarse and very high-density minerals settle quickly and require only a short length of sluice. The length also depends upon the requirement to break up the alluvial ground. Usual practice is to use sluice boxes between 1.2 m and 1.8 m in width. The slope must be adequate to transport the pebbles along the sluice and also prevent sand packing within the riffles. The slopes can vary between 7° and 14°. High slopes are used where there is a great deal of clay or coarse material. The recommended volumetric feed rate is around 0.75 m^3/h/m width of sluice. Excessive feed rates are one of the major factors contributing to gold losses. Lower feed rates lead to minor improvements in recovery. Water requirements are variable and for low feed rates, a good starting point for gold particles less than 1 mm is 40 L/s/m. For high feed rates, water requirements of 80 L/s/m are adequate for recovering gold particles larger than 1 mm.
- **Oscillating sluice box.** An oscillating sluice box consists of a pair of sluice runs suspended from a frame with cables. A direct current electric motor is mounted between and above the sluice runs and rotates a weighted bent shaft through an angle drive. The motor–drive combination creates a horizontal circular panning motion with a 16-mm-diameter circle oscillating at 130–180 rpm. Oscillating sluice boxes are used for gravels containing a high proportion of high-specific-gravity minerals such as magnetite or a high percentage of clay leading to inter-riffle packing (Clarkson 1990).
- **Triple-run sluice box.** Triple-run sluice boxes rely on the ability of the stationary distributor punch plate to screen fine gravels to the side runs (Figure 13). Most of the water entering the distributor remains above the punch plate to move large rocks. Fine gravels and gold are inevitably trapped in these excessive volumes of turbulent water and are swept off with the boulders at high speed down the center run. At times with high feed rates the distributors become inefficient and gold recovery is reduced by underutilizing the side runs and overloading the center run with large pebbles and fine gravel. Triple-run sluice boxes fabricated with large distributors (>30 m) that contain large holes (+20 mm) in their punch plate, sluice gates to control flows to the side runs, adjustable side run slopes, and manually controlled wash monitors are the most efficient (Clarkson 1990).

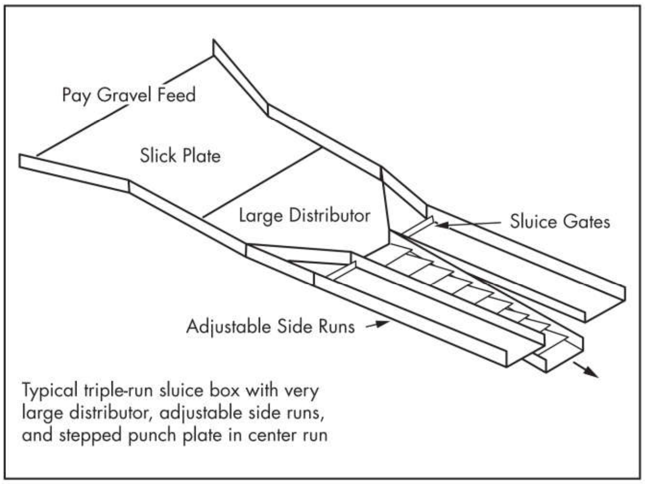
Types of Riffles

The appropriate selection of riffle type is fundamental to the successful recovery of gold particles of different sizes and shapes. The attributes of the more well-known riffle types are as follows:

- **Expanded metal.** Coarse expanded-metal riffles are effective at recovering gold particles finer than 1 mm; however, 25-mm angle iron riffles are required to efficiently recover gold coarser than 1 mm. Coarse gold (+1 mm) losses with expanded metal can be very dramatic as the feed particle size increases. Angle iron riffles require higher water flow rates (1.21 m^3/min) and specified gaps (38–64 mm) and inclinations (–15°) for optimum gold recovery. Angle iron riffles of 25 mm do not tend to pack as readily as larger angle iron riffles. Doubled expanded metal and flat bar riffles are not recommended because of their susceptibility to packing and creating excessive turbulence, respectively (Clarkson 1990).
- **Matting.** Unbacked Nomad matting appears to be the best matting in common use because it does not interfere with riffle operation. Matting made from the "fur" of coconuts, artificial turf, and Monsanto matting are not ideal.
- **Hydraulic.** Hydraulic riffles consist of alternating 50-mm flat bar riffles and 25-mm square tubing that is perforated on the bottom of the sluice. Low-pressure water introduced into the square tubing keeps full riffles loose, unlike conventional riffles that rely on the formation of vortices.

FLOWING-FILM CONCENTRATION

The behavior of solid particles in suspension depends, to a large extent, upon the pulp density and the size of the suspended



Adapted from Clarkson 1990

Figure 13 Triple-run sluice box

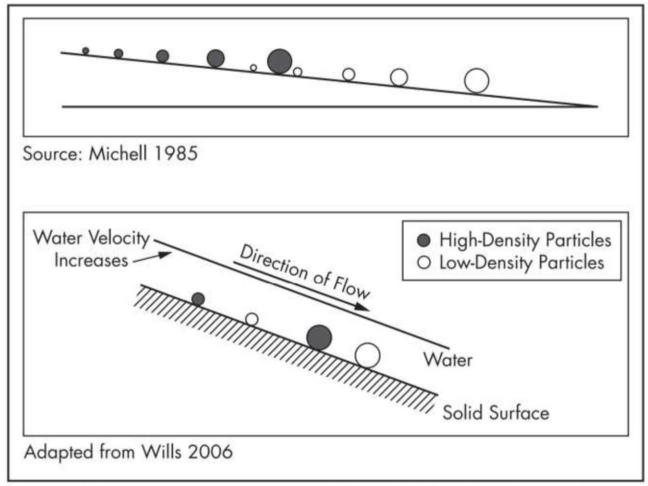


Figure 14 Arrangement of particles of differing size and specific gravity under the influence of a streaming current

particles. In a fairly dilute suspension, such as that normally used when dealing with small particle sizes, the behavior of particles in a flowing film results from two effects. These are the lateral displacement, which is determined by the time taken for each particle to penetrate the flowing film and to reach the solid surface, and the resistance offered by each particle to further displacement after it has reached the solid surface.

The initial penetration through the flowing film depends on the size and density of the particle and the thickness and viscosity of the film. As a result, the smaller particles will migrate further before their movement is retarded relative to larger particles of the same specific gravity. The behavior on reaching the solid surface depends on whether there is a single particle layer or, as is more often the case, a multiple layer or thin bed of material that is sufficiently dilated, which permits the penetration of higher-density particles.

When a thin film of liquid flows over a plane's solid surface, the layer next to the surface remains at rest, but the velocity of the film increases with the distance from the surface and becomes a maximum near, but not quite at, the free surface. Therefore, a particle in suspension in such a film is acted upon by a greater force near the upper part of the film than at the lower part, resulting in an overturning effect. After a particle reaches the separating surface or an accumulated densely packed bed of other particles, the liquid flow causes it to move downstream by rolling, sliding, or a movement involving alternating suspension and deposition (saltation).

In rolling and sliding, which are brought about by a substantially non-eddying stream, the large submerged particles are acted upon to the greatest extent and they move more rapidly than smaller ones, notwithstanding their larger mass. When two particles of the same size but of different specific gravity are considered, the higher-density one moves more slowly because of its greater mass. Consequently, the particles tend to become arranged in the manner shown in Figure 14.

When a thin film of water flows across a flat surface, a fluid shear occurs at the interface between the surface and the water, which creates a fluid velocity gradient. The velocity increases from the surface and through the flowing film. High-density particles within the thin film settle through the high-velocity streamlines and into the low-velocity region at the solid surface, whereas low-density particles are unable to settle from the high-velocity streams and thus report with the majority of the fluid flow.

One of the earliest forms of flowing film separators was the buddle (Davies 1902), followed by the Reichert cone concentrator (Reichert 1965). The unit was developed in response to a need for high-capacity gravity concentration for heavy mineral sand applications (Ferree 1973). Present-day flowing-film gravity separation devices include shaking tables and spirals.

Shaking Tables

The shaking table is a gravity separation device that has been in use for a long time, having been commercialized at the beginning of the 19th century. Little has changed in the design, although multi-deck (up to three levels) tables have led to capacity increases relative to floor area. Shaking tables are normally used only on cleaning stages because of their low capacity. The principle of separation is the motion of particles according to specific gravity and particle size moving in a slurry across an inclined table, which oscillates backward and forward essentially at right angles to the slope. Wash water is provided as a flowing film across the slope of the table. The riffles in the table hold back heavier particles, which are closest to the deck, while lighter particles flow over the riffles. This motion and configuration cause the fine high-specific-gravity particles to migrate closest to the deck and be carried along by the riffles to discharge uppermost from the table, while the low-specific-gravity coarser particles move or remain closer to the surface of the slurry and ride over the riffles, discharging over the lowest edge of the table (Figure 15).

Shaking tables include the following operating variables:

- Angle of deck (a steeper angle means less weight to concentrate).
- Length of stroke (the longer the stroke, the more sideways the motion and hence more weight to concentrate, up to a maximum).
- Frequency of stroke (similar to length; that is, the more frequent the stroke, the more sideways the motion, up to a maximum).

- Splitter positions (the position of the splitters on the concentrate launder will determine the mass to the concentrate).
- Feed rate and density (above a maximum of typically 2 t/h per full-size table and density typically 40% solids, depending on the type and particle size of the feed) separation will be reduced.

Shaking tables have been the most popular of the vibrating separators and remain common today for the recovery of gold, tin, and other heavy metal minerals in the particle size range of 1.65 × 0.074 mm and coal in the 6.7 × 0.15 mm particle size range. Many suppliers provide shaking tables, and of these, Deister and Holman-Wilfley are the best known.

Deister Shaking Table

The deck of a Deister shaking table has a one-piece, ultra-smooth rubber cover, in either black or white, with riffles for sand or slime application (Figure 16). Launders are molded high-density polyethylene with adjustable cutting pans. Five models are available, as shown in Table 3, and technical information is provided in Table 4.

Holman-Wilfley Shaking Table

The Holman-Wilfley shaking table is driven by a head motion system with a 1.5-kW motor and with a nominal stroke adjustment of between 8 mm and 16 mm. The deck is supported on 42 carriers and is vibrated diagonally along the entire length. Good-quality timber with standard 5-mm rubber covers for maximum durability (color white or black options) is used for the decking. Different riffle patterns are available to maximize performance, depending on the application. Table tilt can be easily adjusted using a hand wheel, even while the machine is in operation. All feed/product launders are polyurethane lined.

Three models are available: 2000, 3000, and 8000. The 8000 model is available in single- or double-deck configuration. The Model 8000 is shown in Figure 17 and technical information is provided in Table 5.

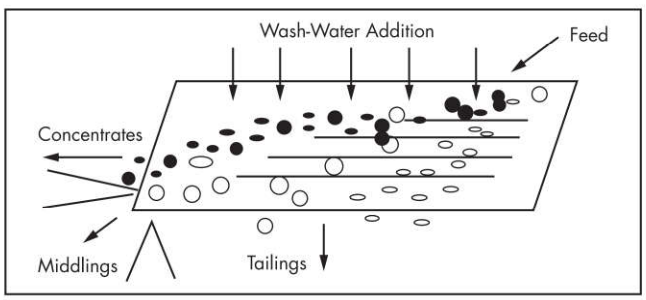


Figure 15 Flow of particles on the deck of a shaking table

Gemeni Shaking Table

The Gemeni shaking table has been specifically designed for the recovery of fine gold to a directly smeltable concentrate (Figure 18). The new direct drive system incorporates a geared motor direct driving a crank connected to the table deck. The crank incorporates a sprung connection system to absorb overrun. The bump stop system has been maintained to provide a fine-tuning mechanism. Table tuning is achieved by adjustment of a single screw (Mineral Technologies, n.d.). Gemeni tables are available in three models, as shown in Table 6.

Spiral Concentrators

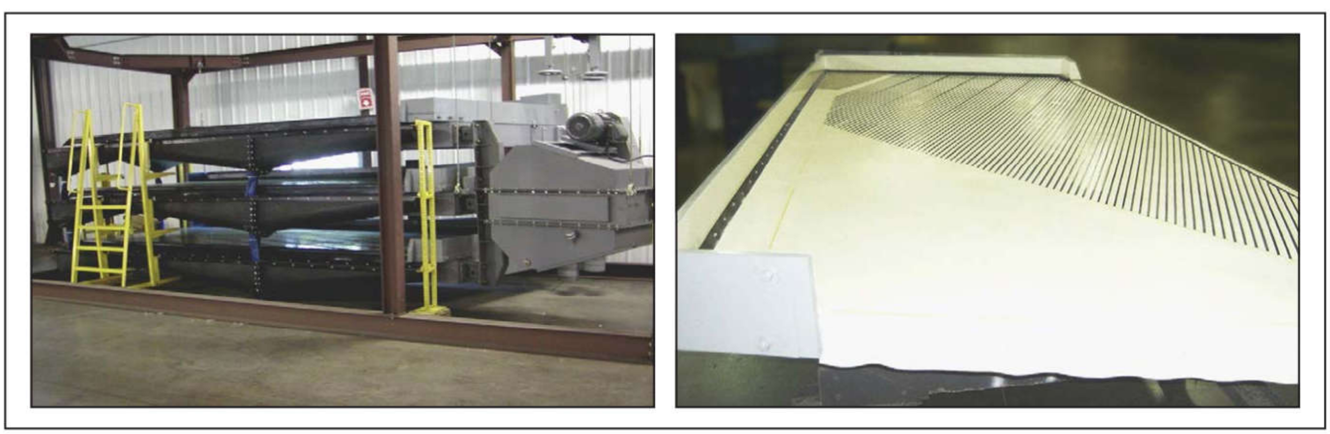
The spiral concentrator has been widely used in the minerals industry to achieve efficient fine particle density-based separations since 1943 when the Humphreys spiral was introduced (Humphreys 1975). The separation is a result of a circulating flow, as shown in Figure 19. Light particles are lifted by the circulating flow from the inner part of the trough and carried to the outer area of the trough, whereas high density particles settle onto the trough surface and move toward the inner part of the trough. Internal channels along the vertical axis and splitters located at the end of the trough are used to collect the light and heavy particle streams.

The spiral concentrator incorporates the use of wash water to enhance the separation, and drawpoints are located

Table 3 Deister shaking tables

Model No.	Description
15-S	Laboratory table
14	Pilot-sized table
6	Full-sized table
99	Double deck
999	Triple deck

Courtesy of Deister Concentrator



Courtesy of Deister Concentrator

Figure 16 Deister triple-deck tables and slime deck covering

Table 4 Technical information for Deister shaking tables

	Particle	Capacity per Deck, t/h			Dressing Water per Deck, m^3/h		
Type of Feed	Top Size, mm	999/99/6	14	15-S	999/99/6	14	15-S
Coarse sand	2.0	2.3	1.15	0.115	2.3–3.6	1.1–1.8	0.11–0.18
Medium sand	0.30	1.4	0.70	0.070	1.8–2.7	0.9–1.4	0.09–0.14
Find sand	0.15	0.9	0.45	0.045	1.1–2.0	0.7–1.1	0.07–0.11
Slime	0.044	0.4	0.20	0.020	0.7–1.1	0.5–0.9	0.05–0.90

				Deck Adjustments, mm/m	
Type of Feed	Feed % Solids	Strokes/min	Stroke, mm	End Elevation, mm	Side Tilt, mm
Coarse sand	20	260–265	19	20	25
Medium sand	22	265–270	19	15	20
Find sand	23	270–275	13	10	15
Slime	25	275–280	13	5	10

Courtesy of Deister Concentrator
Note: The end elevation is measured parallel to the uppermost riffle, whereas the tilt is measured at a right angle to the riffles.

Courtesy of Holman-Wilfley
Figure 17 Holman-Wilfley Model 8000 shaking table

Table 5 Holman-Wilfley shaking table details

Model	Deck Area, m^3	Width, mm	Length, mm	Motor, kW	Water, L/min	Rate, kg/h
2000	2	887	2,445	1.5	5–20	<450
3000	3	1,030	2,515	1.5	10–25	<850
8000	7.5	1,600	4,900	2.2	20–35	<2,500

Courtesy of Holman-Wilfley

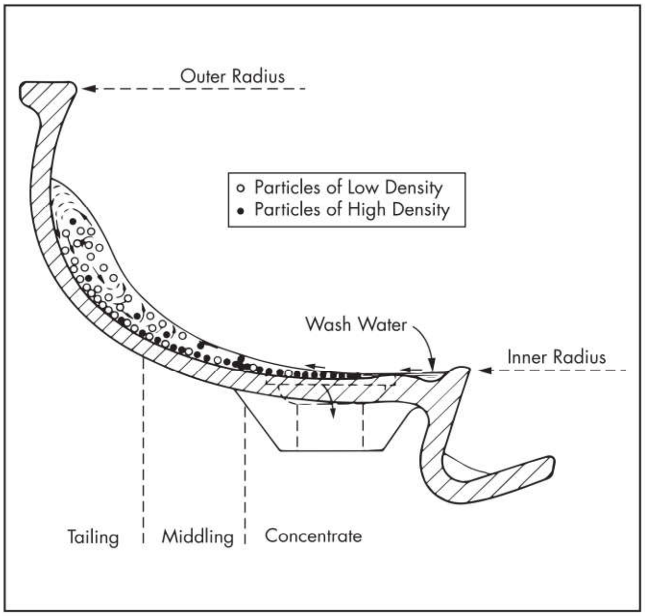
at various points along the bottom of the trough to remove the high-density particles. Commercial applications for several decades after its introduction were mainly for easy mineral separations such as enriched chromium placer ores, iron ore, and heavy mineral sands. The application limitations were mainly due to the construction from cast iron or cement that limited the ability to vary the profile and pitch of the trough. Around 1980, the advancement of construction materials led to spirals being constructed of fiberglass and spray coated with polyurethane. The new material construction allowed alterations to the trough geometry to address the needs of more difficult separation applications. The lightweight material construction permitted the entwining of two

Courtesy of Mineral Technologies
Figure 18 Gemeni shaking table

Table 6 Technical information for Gemeni tables

	Model		
Operating Data	**GT60**	**GT250**	**GT100**
Feed rate nominal, kg/h	30	115	450
Fed density recommended, % solids w/w	60–70	60–70	60–70
Feed size nominal top, µm	800–1,000	800–1,000	800–1,000
Nominal wash-water requirements, L/min	12	25	38
Nominal wash-water pressure, kPa	30	30	30

Source: Mineral Technologies, n.d.



Courtesy of Humphreys Engineering Company

Figure 19 Fluid flow pattern along the cross section of the spiral trough, which provides the density-based particle separation

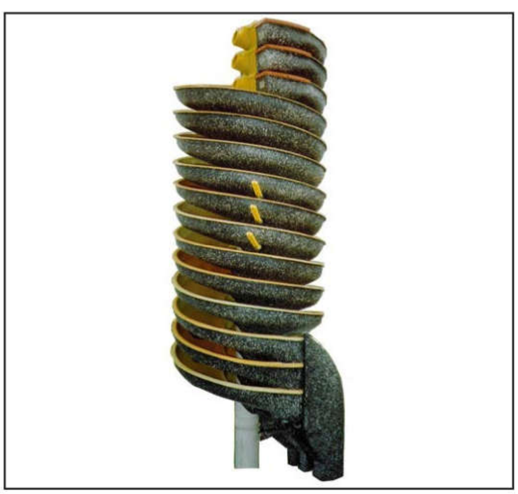
or three spiral starts around one central vertical stick or axis. For fine coal applications, the trough was designed to allow higher particle retention times by making the trough wider, changing the trough profile, and decreasing the vertical pitch (Richards et al. 1985). Higher throughput capacities per start were realized for mineral applications from the redesign of the trough. A summary of the spiral trough changes is described by Richards and Palmer (1997). Studies conducted by Luttrell et al. (1998) revealed that significant improvements in separation efficiency can be realized when re-treating light particle streams of a spiral concentrator in a second spiral unit in a rougher-cleaner arrangement.

There are many suppliers of spiral gravity separators, such as Mineral Technologies, Multotec, Outotec, and FLSmidth (Krebs spirals).

Mineral Technologies spirals. The Mineral Technologies spirals are fiberglass reinforced and polyurethane lined. The splitters are designed to cater for ease of operation regardless of spiral size and number of splitters. Concentrate diverters improve concentrate grade. Repulping devices incorporated in the spiral further enhance recovery of product. Accurate top or bottom entry feed distribution systems allow for improved recoveries. Replaceable modular cast polyurethane feed and concentrate boxes provide longer spiral service life. Spiral configurations come in single, twin, triple, and quadruple troughs per column to suit capacity requirements. Spiral banks of up to 48 starts are available, to maximize capacity and minimize floor space requirements. High capacity 12 × 3 start spiral modules can process in excess of 150 t/h. Figure 20 shows a single spiral and a bank of spirals.

A multitude of spiral designs are available for particular mineral types and applications in a spiral circuit. A brief description of the different Mineral Technologies designs is as follows (Mineral Technologies 2017a, 2017b):

- MG series—for feed material generally containing up to 25% heavy minerals (up to 40% in some applications).
- HG series—for high-grade feed material generally greater than 25% and as high as 90% heavy minerals.
- VHG model—for feeds with very high levels of heavy minerals.



Courtesy of Mineral Technologies

Courtesy of Iluka Resources Limited

Figure 20 Single spiral and bank of spirals

- FM series—for valuable heavy mineral particles in the range of 30 to 150 µm.
- WW series—utilizes wash-water addition for enhanced grade control in specific applications (e.g., iron ore).
- HC series—super high-capacity spirals specifically designed for more economical and compact plants. The facility to add wash-water is available on some models.
- LC3 coal spiral—a revolutionary new low-cut-point spiral producing a low cut point of 1.4 to 1.5 RD (relative density); and allowing superior product yield at conventional 1.7 to 1.8 RD cut points. The design allows for less buildup than conventional spirals, leading to improved beneficiation of ultrafine material.
- The LD7RC coal spiral—a combined rougher-cleaner spiral featuring seven turns consisting of three rougher turns and four cleaner turns. It offers reduced overall height when compared with standard two-stage systems. This spiral is particularly effective on feeds with high levels of near-gravity material and other difficult coal separations. The unique rougher-cleaner transition stage ensures thorough repulping of the slurry for maximum separation. This system operates within the trough, saving space/height, and dispenses with separate inter-stage components (Figure 20).

Multotec spirals. Six basic models with up to three combinations are available to cater to different mineral applications and circuit design. Units come in three to eight turns with high-, medium-, and low-grade profiles. The spiral trough is sprayed with polyurethane to a thickness of 3 mm ±0.5 mm. Splitters are of solid polyurethane casting. Collected material is channeled into a specific product box outlet.

The following Multotec models are available (Multotec 2016):

- SX4, MX7, and SX7 spirals—The SX4 is a single-stage four-turn spiral used for coal, while the SX7 (seven-turn, for coal) and MX7 are double-stage spirals. Both the primary and secondary stages are contained in one assembly.
- SC20/7 spiral—Used for rougher and scavenger applications.
- SC20 and SC21 spirals—Used for cleaning applications in the mineral sands industry.
- SC20 LG—Twin- and triple-start spirals.
- SC20 HC—High-capacity twin-start spiral with high-flow wash water.
- SC20 VC—Twin- and triple-start spirals.
- SC21—Twin- and triple-start spirals.
- SC21/5—High grade spiral, used for cleaner and recleaner.
- HX3/HX5—Twin- and triple-start spirals.
- HX5 spirals—Shallow in angle and used to treat low-grade ores.
- NHM spirals—For the rougher and scavenger stage. Specifically designed for heavy mineral sands applications.

Outotec spirals. The Outotec urethane fiberglass spirals either have wash water or are wash-waterless. Details of the spirals are provided in Table 7.

Krebs spirals. Krebs spirals are primarily designed to treat pulp streams with a heavy mineral content in the range up to 35% by weight. Each spiral consists of a backfed feed box, single-section helix, and a product box. Finger cutters on the helix divert separated minerals into channel areas. All splitters are located in the product box to divide the stream into concentrate, middlings, and tailings. The spiral can be used in rougher, scavenger, and re-treat duties. The spiral is primarily designed to treat pulp streams with a medium heavy-mineral content up to 15%–80% by weight.

Two models of spirals are available (FLSmidth-Krebs 2014b):

1. LM series—Capacity up to 2 t/h per start with concentrate removal up to 0.7 t/h per start. Feed pulp density up to 45% solids. Models available with three, five, and seven turns in simplex, duplex, and triplex configurations.
2. MM series—Capacity up to 2 t/h per start with concentrate removal up to 0.5 t/h per start. Feed pulp density up to 45% solids. Models available with five and seven turns in simplex, duplex, and triplex configurations.

FLUIDIZED-BED SEPARATION

Fluidized-bed separators (FBSs), also known as teetered-bed or hindered-bed separators, have been used for more than a century in coal and mineral processing plants, primarily for

Table 7 Details for Outotec spirals

Model	Pitch, mm	Starts	Number of Turns	Wash-Water Addition	Concentrate Takeoff	Concentrate Collection	Main Features	Suggested % Solids
CS2000	420	6	Single, double, or triple	No or yes	Splitter at each turn and at discharge splitter box	Inboard collecting trough	Larger diameter, high capacity	20–40
LC3000	410	5	Single, double, or triple	No	Splitter at discharge splitter box	Central channel into discharge splitter box	Low-grade application (wash-waterless)	25–40
LC3700	410	7	Single, double, or triple	No	Splitter at discharge splitter box	Central channel into discharge splitter box	Agitator bumps (optional)	25–40
MC7000	410	7	Single or double	No	Splitter at each turn and at discharge splitter box	Inboard collecting trough	Medium grade and wash-waterless	25–40
HC8000	410	7	Single, double, or triple	No	Splitter at each turn and at discharge splitter box	Inboard collecting trough	High grade and wash-waterless/easy adjustment splitter handle	25–40
H9000W	460	7	Single or double	Yes	Splitter at each turn and at discharge splitter box	Inboard collecting trough	High grade and wash-water/easy adjustment splitter handle	25–40

Adapted from PhySep, n.d.

particle size separations. FBS units can be operated in a manner that provides a very efficient density-based separation by utilizing the finest high-density particles in the feed stream to create an autogenous dense medium.

One of the oldest and perhaps most popular teetered-bed separator is the Stokes unit. The positive features of FBS units include the ability to achieve efficient density-based separations at relatively high mass flow rates (10–20 t/h/m^2), the capability of adjusting online to changes in feed characteristics as well as variations in feed rate, and their overall simplicity of operation. Their inherent disadvantages are the need for tight top-size control in the feed to prevent the recovery of low-density oversize in the underflow stream, the need of a mechanical underflow valve and control system that requires maintenance, the use of clean water in the injection system to prevent plugging, and the existence of components that are susceptible to wear in the bottom of the units.

Many commercial units operate in much the same manner as the Stokes classifier, including the Floatex (Mankosa et al. 1995; Elder et al. 2001), Lewis hydrosizer (Lewis 1990), Linatex hydrosizer (Deveau and Young 2005), Allflux separator, and the Hydrosort (Doerner 1997). The Allflux separator is a unique unit in which two stages of density-based separations using fluidized particle beds occur in a single unit (Short et al. 2001).

A problem associated with conventional FBS units is the turbulence generated by feed injection into the center of the unit at a depth of around one-third of the total height. The turbulence disrupts the settling of large and small high-density particles into the fluidized-bed zone of the FBS, and the upflow velocity of water leads to the loss of high-density particles into the overflow stream. To counter this problem, Mankosa and Luttrell (1999) equipped an FBS system with a unique feed system that gently introduces the feed slurry across the top of the separator using a transition box (i.e., the Crossflow separator).

More recently, Galvin et al. (2009, 2010; Galvin 2012) discovered the benefits of using closely spaced inclined channels, which promote laminar flow and a high shear rate at the plate surface. This leads to enhanced separation efficiency. A new separation mechanism, referred to as the laminar-shear mechanism, develops in the inclined channels. Relatively fine, dense particles segregate from the flow, sliding downward and returning to the lower fluidized-bed zone. However, lower-density particles over a broad range of particle sizes continue to convey upward through the inclined channels. The relatively large, low-density particles experience inertial lift, and hence fail to segregate onto the inclined channels. These relatively coarse particles become exposed to the higher fluid velocities, conveying easily to the overflow. The inclined channels also provide a significant capacity advantage over conventional fluidized beds, especially as the particle size decreases. Moreover, the system is insensitive to low feed-pulp density and can be deployed directly after a classifying screen without the need for thickening cyclones.

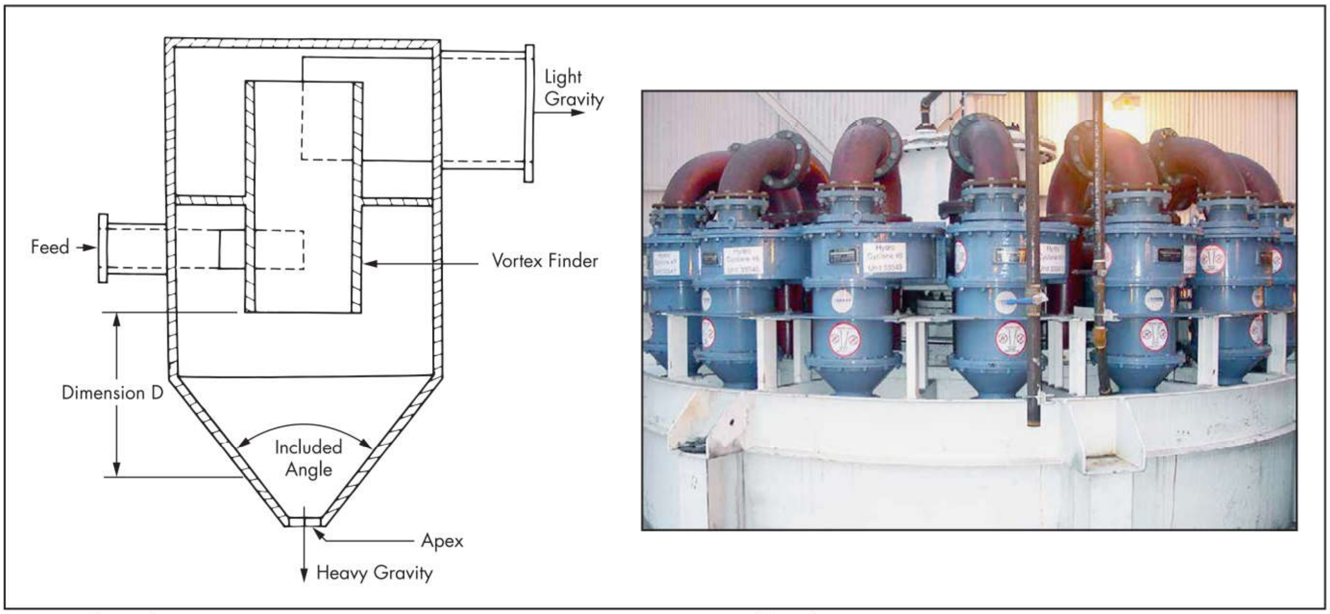
In an effort to increase the effective particle size ratio of separation, Mankosa and Luttrell (2002) developed a process referred to as the HydroFloat separator in which air bubbles are injected into the fluidized particle bed. Using this concept, the effective particle size range that can be treated in a modified FBS unit is around 6:1 (Kohmuench et al. 2001; Luttrell et al. 2006). The benefits of the units are discussed by Kohmuench et al. (2007).

WATER-ONLY CYCLONE

The water-only cyclone is a modified classification cyclone that makes specific gravity separations utilizing water as the separating fluid (Figure 21). Since the water-only cyclone does not employ an external heavy medium, the effect of classification forces (forces that separate on the basis of particle size and shape as well as density) must be minimized by the design features of the cyclone if a sharp specific-gravity separation is to be made. To minimize the classification forces, the design of the water-only cyclone, as compared with conventional classification cyclones, has a wider included angle (60°–120°) and a longer vortex finder. Along with these two design elements, other features, such as cyclone diameter and the sizes of the overflow, underflow, and feed orifices, also affect its performance. Extending the vortex finder toward the apex orifice reduces the vertical distance between the end of the vortex finder and the wall of the conical section of the cyclone. This dimension is one of the most critical design variables of the water-only cyclone. The relative shortness of this dimension means that the vertical path travelled by particles in the upward central current of the cyclone is reduced sufficiently so that large low-specific-gravity particles caught in the upward current will not reach their terminal settling velocity and settle out to the wall, but will be captured inside the vortex finder and discharged with the light-specific-gravity fraction.

Operating variables that need to be addressed include the following:

- **Feed solids.** The performance of a water-only cyclone is greatly affected by varying feed concentrations. An increase in feed concentration increases the specific gravity of separation and slightly decreases the efficiency. This factor is responsible for difficulties in the control of water-only cyclones.
- **Feed orifice.** Increasing the diameter of the feed orifice on a water-only cyclone increases the capacity of the device approximately in proportion to the square of the orifice diameter. Increasing the feed orifice diameter decreases the retention time of the feed in the cyclone and thus increases the specific gravity of separation.
- **Overflow orifice.** Increasing the diameter of the overflow orifice has about the same effect as increasing the feed orifice diameter. The capacity is increased, the retention time is decreased, and the specific gravity of separation increases. An increase in the overflow orifice diameter increases the specific gravity of separation more than does a comparable increase in the feed orifice diameter.
- **Underflow orifice.** An increase in the diameter of the underflow orifice increases the capacity of the cyclone for handling refuse or reject material and decreases the specific gravity of separation. Efficiency decreases slightly with an increase in orifice diameter.
- **Orifice ratio.** The performance of a water-only cyclone is affected by the relationship of the diameter of the overflow orifice to that of the underflow orifice. This relationship determines the flow ratio of the cyclone (i.e., the ratio of apex flow to feed flow). Generally, the ratio of the diameter of the overflow orifice to that of the underflow



Source: Miller et al. 1985 Source: FLSmidth-Krebs 2014a

Figure 21 Cross-sectional view of a water-only cyclone and production cyclones

Table 8 Water-only cyclones available from Krebs*

Model	Maximum Feed Particle Size, mm	Effective Separation, μm	Dry Feed Range, t/h	Pulp Flow Rate Range, m^3/h	Pressure Drop Range, kPa	Maximum Feed % Solids	
						Weight %	Volume %
D10LB-S-218	2	147–104	4–7	43–59	55–103	10	7
D15LB-S-245	3	208–147	11–16	91–132	69–124	12	8
D15LB-S-327	6	208–147	11–23	116–164	69–138	12	8
D20B-S-260	13	295–208	23–41	186–238	83–138	15	11
D20LSB-S-333	13	417–295	32–54	250–341	83–138	15	11
D26-S-224	19	417–295	45–82	338–500	83–152	20	15

Adapted from FLSmidth-Krebs 2014a
*Typical operating parameters.

orifice for water-only cyclones ranges from 1.5:1 to 2:1. The feed orifice is typically slightly smaller in diameter than the overflow orifice.

- **Feed pressure.** Except in those cases where coarse particles, which may cause excessive wear, are being processed, water-only cyclones are generally operated at feed pressures of 100 kPa or greater to ensure high separation efficiencies. An increase in feed pressure increases the separating efficiency and also increases the volume of feed material processed. To extend the separation ability of the cyclone to process the finest sizes, higher feed pressures are required.
- **Cyclone included angle.** This can influence separation efficiency. Some units have an included angle of 60°–95°. Furthermore, modifications of the shape or profile of the conical portion of the cyclone have been developed. Another modification features a spherical bottom for the cyclone.

Table 8 shows cyclones available from Krebs with feed volume and particle sizes that may be processed.

CONCENTRATION CRITERIA FOR GRAVITY SEPARATION

A consistent feed rate and feed density are essential for shaking tables and spiral concentrators, whereas jigs and centrifugal devices are relatively insensitive to feed rate and pulp density. A concentration criterion (CC) provided by Taggart (1945) provides an indication of the practicality of gravity concentration (other than dense medium separation):

$$CC = (D_h - D_f)/(D_l - D_f)$$

where

D_h = density of the heavy particles
D_l = density of the light particles
D_f = density of the fluid (usually water at 1.0)

Gravity concentration will usually be successful if $CC > 2.5$. When $CC < 1.25$, gravity concentration is virtually impossible. When $2.5 > CC > 1.25$, gravity separation is very difficult but may be possible to some extent with narrow feed size classification.

ENHANCED GRAVITY SEPARATION

Enhanced gravity separation involves the application of centrifugal forces to target relatively fine particles. Particles that would normally settle according to Stokes' law experience a centrifugal force with an effective acceleration of G times the normal acceleration, g, due to gravity. This g-force produces a significant increase in the terminal velocity of each particle. Moreover, the settling regime shifts toward the intermediate regime (Vance and Moulton 1965) in which the terminal velocity scales directly with the particle diameter. This means that the dependence of the particle velocity on the particle size decreases, which in turn leads to an enhanced level of separation performance.

The need to utilize centrifugal force to recover granular minerals from ore based on density difference dates back to the late 1800s (Seymour 1893; Ponten 1910; Bradbury 1912). Eccleston (1923) developed the first fully continuous EGS. Several similar types of EGS units were developed and commercially used in Russia and China throughout the late 1900s (Burt 1984). EGS units having capacities from 0.5 to 14 t/h when utilized for the recovery of gold, cassiterite, tungsten, iron, cinnabar, and other heavy minerals.

The developments that led to the EGS units commonly used in the industry today occurred over the last two decades of the 20th century, and detailed reviews of these technologies are provided by Luttrell et al. (1995) and Cole et al. (2012). The technologies vary by their separation mechanism and the magnitude of the applied centrifugal field. Each process is reportedly effective over a particle size range of ~1 mm to 10 µm. For material having a heavy mineral content less than about 1% by weight, semicontinuous units are recommended. Fully continuous units are available for feed streams containing more than 1% heavy minerals. Mass throughput capacities are as high as 1,000 t/h for semicontinuous units and 300 t/h for the continuous units.

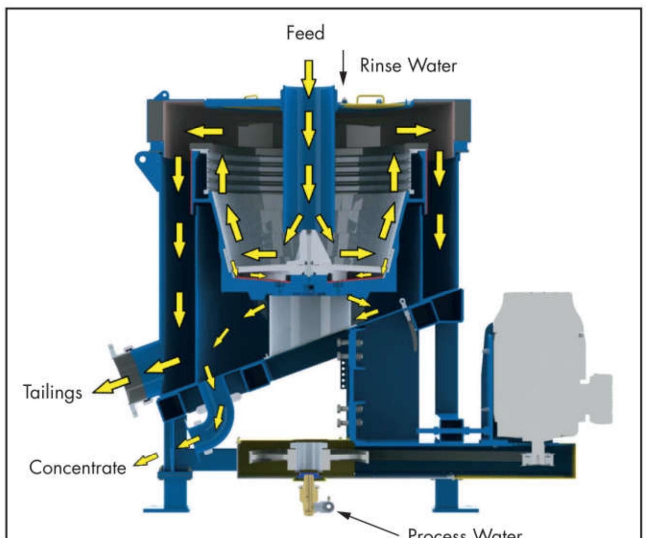
Falcon Centrifugal Separators

A review of the historical development of the Falcon is provided by McAlister and Armstrong (1998). Three models are available: a semi-batch (SB) unit, a continuous (C) unit, and a unit to treat ultrafine (UF) material.

The Falcon SB concentrator uses a rotating bowl to generate the high gravitational forces required for mineral separations. Slurry is fed through a fixed center pipe to a distribution plate. The SB bowl has a smooth-walled bottom section and fluidized, riffled upper collection zone. The smooth-walled section allows higher-density particles to stratify and move to the wall, and lighter particles are displaced toward the center of the bowl. The collection zone is vertical and the riffles are fluidized by water injected from the back of the riffles. The fluidization elutriates and cleans the heavy concentrates in the riffles. The term *semi-batch* is used to describe the cycle; the unit is fed for a predetermined time, then the feed and machine are stopped. The concentrate is flushed with water to a concentrate discharge pipe at the bottom center of the unit. The lighter material proceeds over the lip of the collection zone to a tailing launder. The Falcon SB units have a treatment range of 1 to 400 t/h and are typically used in a grinding circuit either on cyclone underflow or cyclone feed. The SB concentrator can operate between 50 and 200 g and typically operates between 90 and 120 g. The g-force operation is determined by metallurgical testing. Clean, high-quality fluidization water is required to operate fluidized-bed-type centrifugal gravity concentrators. The flow is controlled by the process automation system, and water is injected through small holes into the fluidized concentrate bed. The holes through which water is injected in the unit are short, relatively large in diameter, and radially drilled. The material selected is stainless steel that is highly resistant to particle embedment (Sepro Mineral Systems 2018b). Figure 22 shows the features of the Falcon SB unit and operating data for the different SB models is provided in Table 9, including information on the laboratory batch unit, the L40.

The Falcon C concentrator also utilizes a spinning bowl to generate gravitational forces. The bowl is inclined outward and has smooth walls to allow for stratification of higher density particles. At the top of the bowl is a ring of specially designed hoppers followed by pneumatically controlled valves similar to muscle valves. These valves are regulated to produce a constant stream of concentrate. The C machine is able to produce a high concentrate mass. Concentrate masses can vary from 5% to 40% depending on the application. There are four sizes of C concentrators ranging from 1 t/h to 100 t/h. Feed must be screened to 1 mm to prevent blockage of the concentrate valves. These machines are able to operate between 50 and 300 g but typically between 160 g and 200 g. The unit can concentrate particles as fine as 10 µm (Sepro Mineral Systems 2018a). Figure 23 shows a schematic of the Falcon C unit, and Table 10 provides the operating data for different models.

The Falcon UF concentrator is a semicontinuous unit that is focused on the treatment of particle sizes between 37 µm and 3 µm. The UF unit utilizes the same spinning bowl as the SB and C concentrators. The UF concentrator has a smooth, outwardly inclined bowl and retains concentrate through a pneumatically controlled lip at the top of the bowl. The lip slowly expands through the sequence of a cycle. The UF unit operates in semi-batch mode with a shutdown to clean the concentrate. Typical cycle times are approximately 3 minutes. Typical feed for the UF unit is desliming cyclone overflow. The machine operates at up to 600 g and can recover minerals down to 3 µm. The target application is scavenging of slime reject streams (Sepro Mineral Systems 2018c).



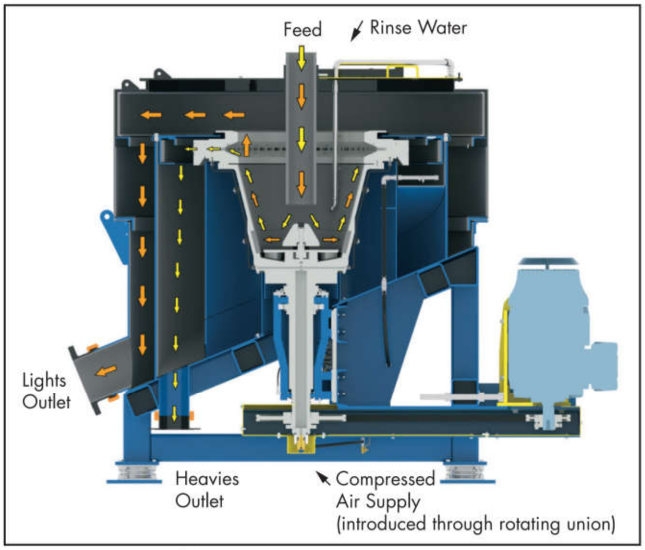
Source: Sepro Mineral Systems 2018b
Figure 22 Falcon semi-batch unit

Table 9 Operating information for the Falcon semi-batch units

	Model				
Operating Data	**L40**	**SB750**	**SB1350**	**SB2500**	**SB5200**
Recommended solids capacity, t/h	0–0.25	10–80	50–150	100–250	200–400
Approximate maximum slurry capacity, m^3/h	2.3	100	200	300	450
Concentrating surface area, m^2	0.03	0.46	1.08	2.14	3.37
Upper g-force range	200	200	200	200	200
Lower g-force range	50	50	50	50	50
Machine weight, kg	35	1,250	2,900	4,560	7,720
Motor power, kW (hp)	0.4 (0.5)	7.5 (10)	18 (25)	45 (60)	75 (100)
Process water consumption, m^3/h	0.24–1.2	8–12	12–20	15–28	25–35
Continuous water supply pressure, bar	2–3	2–3	2–3	2–3	2–3
Recommended maximum feed particle size, mm	1.0	2.0	2.0	2.0	2.0

Adapted from Sepro Mineral Systems 2018b

Clean, high-quality fluidization water is required to operate centrifugal gravity concentrators. Water is injected through small holes into the fluidized concentrate bed, and the flow is controlled by a process automation system. The holes through which water is injected in the unit are short, relatively large in diameter, and radially drilled.



Source: Sepro Mineral Systems 2018a

Figure 23 Falcon continuous concentrator

iCON Concentrator

The iCON concentrator shown in Figure 24 is an innovative, small gravity concentrator. The iCON is designed for artisanal duty. Ore is fed through a fixed pipe in the top of the unit. Fluidization water is introduced through the main shaft to the capture zone in the bowl. The unit utilizes a variable-speed drive capable of 150 g, which allows the capture of very fine gold not recoverable by traditional techniques of artisanal miners. To recover the gold the unit is shut down and gold concentrate flushed through the center bottom to a concentrate outlet pipe. Details of the iCON concentrator are shown in Table 11.

Knelson Centrifugal Concentrator

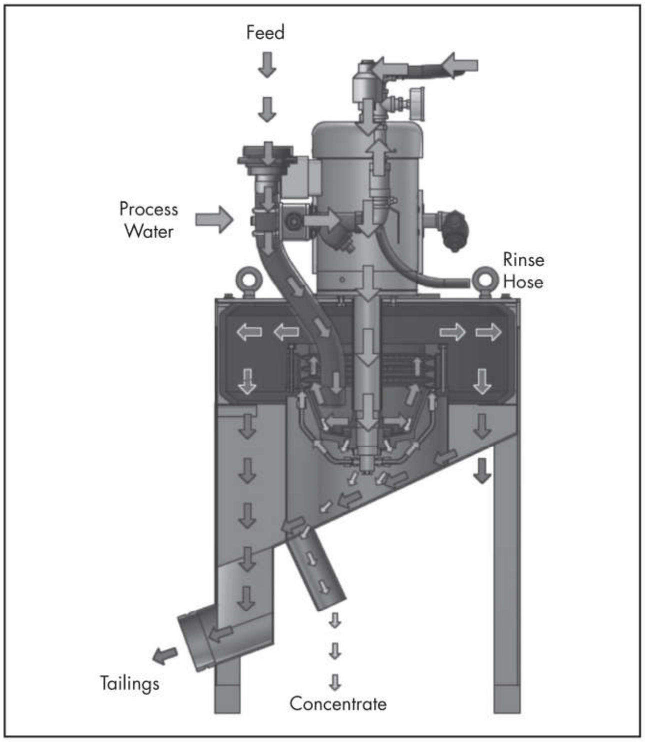
The concept of the Knelson concentrator was patented in 1986 by B.V. Knelson and has been applied worldwide to the treatment of a variety of minerals (Knelson 1992). Both the semicontinuous and the continuous variable discharge (CVD) machines employ the concepts of fluidized particle bed separation in a mechanically applied centrifugal field.

The Knelson semi-batch concentrator operates by introducing water through a series of fluidization holes located in rings that circle the circumference of a bowl (Figure 25). The bowl, which has a truncated cone shape, is rotated at speeds that provide a centrifugal field from 60 to 200 times gravity. Feed slurry is introduced through a pipe that directs the material toward the bottom center of the machine. The heavy particles settle into the bottom of each slot of the concentrating bowl and are retained until the machine is stopped and the

Table 10 Operating data for the Falcon continuous units

	Model			
Operating Data	**C400**	**C1000**	**C2000**	**C4000**
Recommended solids capacity, t/h	1–5	5–27	20–60	45–100
Maximum slurry capacity, m^3/h	17	74	210	400
Maximum particle feed size, mm	1	1	1	1
Minimum effective capture size, μm	10	10	10	10
Concentrate % solids	65–72	65–72	65–72	65–72
Maximum feed % solids	40–45	40–45	40–45	40–45

Adapted from Sepro Mineral Systems 2018a



Source: Cole et al. 2012

Figure 24 iCON concentrator

Table 11 Specifications for the iCON concentrator

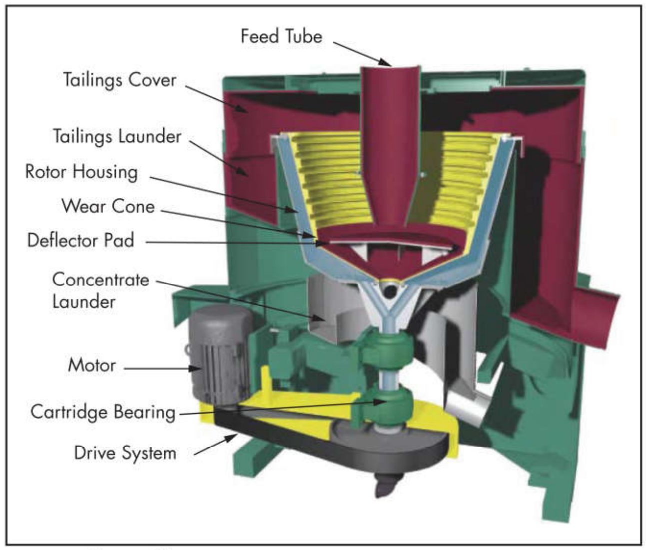
Operating Data	Units
Solids capacity	2 t/h
Maximum slurry capacity	100 L/min
Concentrating surface area	968 cm^2
G-force range	60–150 g
Machine weight	115 kg
Motor power	1.5 kW
Process water requirement	17 L/min
Water pressure requirement	1.0 bar
Maximum feed particle size	2 mm
Dimensions	0.6 × 0.6 × 1.29 m

Adapted from Cole et al. 2012

bowl is flushed. The duration of the concentrating cycle will vary, depending on the application. In hard rock milling applications, cycle durations typically range from 15 to 90 minutes. In alluvial applications, cycle durations range from 1 to 4 hours.

Four variations of the concentrating cone provide options to adjust the quantity of fluidizing water and so change the quantity of concentrate collected (Table 12). Operating information for the commercially available Knelson semi-batch concentrators is provided in Table 13, while information for the laboratory units is shown in Table 14.

The Knelson CVD concentrator was developed to address the limitation with respect to mass yield (concentrate) of the batch unit. The batch machines are limited to a mass yield of about 0.1%, while the CVD mass yields can be varied from about 0.1% to 50%, depending on the feed characteristics. The operating principles of the CVD concentrator are similar to those of the Knelson semi-batch concentrator but they allow the concentrate to be emitted from the fluidized bed continually. A series of pinch valves, located at the base of the fluidized rings, are kept closed by air pressure. By releasing the air pressure periodically, concentrate can be emitted without an interruption in production. Figure 26 shows a schematic of the CVD-32 model concentrator. The mechanism of separation and recovery is quite similar to the batch machine. The operating variables affecting the quantity of concentrate produced are the pinch valve opening and closing times. Operation information for the CVD unit is provided in Table 15.



Courtesy of FLSmidth

Figure 25 Knelson semi-batch gravity concentrator

Table 12 Concentrating cone details for the Knelson semi-batch unit

Cone Style	Fluidization Water, m^3/h	Concentrate Weight, kg	G-Force Range, g
G4	30–39	31–45	60
G5	17–24	18–29	60
G6	26–39	31–45	60–120
G7	14–27	34–59	60–200

Adapted from FLSmidth-Knelson 2013

Kelsey Centrifugal Jig

The Kelsey jig consists of a series of hutches that are rotated around a central feed pipe (Figure 27).The unit is capable of generating centrifugal fields up to 100 g. A cylindrical wedge-wire screen is mounted across the top of each hutch to retain ragging material. Feed slurry enters the unit through the central feed pipe and flows outward across the bed of ragging. Hutch water is added under pressure and the water pulsed by means of push arms acting on a flexible diaphragm. The pulsations create oscillations in the bed that differentially accelerate particles based on differences in density. Low-density particles flow across the ragging material and overflow the top of the unit, while high-density particles pass downward

Table 13 Operating information for the Knelson semi-batch industrial concentrators*

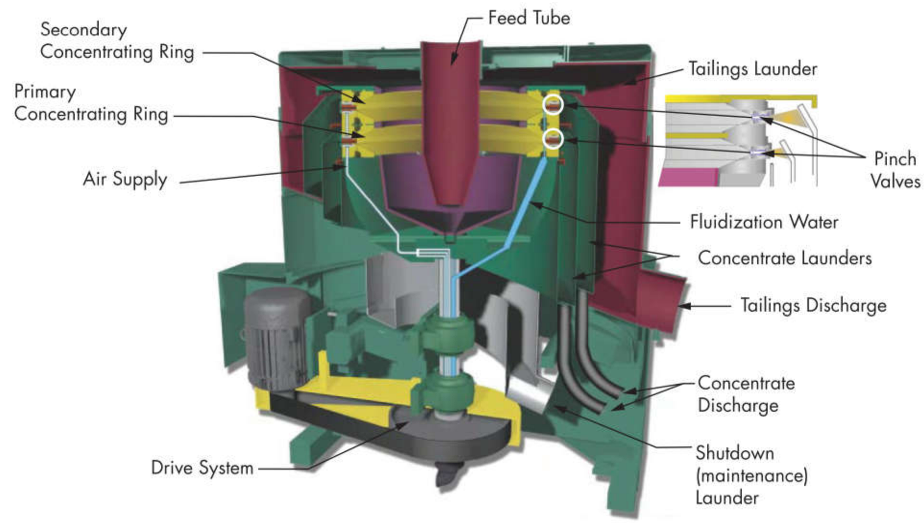
Model†	Feed Solids, t/h	Feed Volume, m^3/h	Concentrate Solids, kg	Concentrate Volume, L	Fluidizing Water, m^3/h	Motor, kW
CD10	8	10	2–3	1.2	3–5	1
CD12	20	27	5–7	1	6–10	1.5–3.8
CD20	80	110	9–11	5	8–11	5.5–7.5
CD30	100	135	23–29	13	17–24	11
XD20	80	110	9–11	5	8–11	5.5–7.5
XD30	150	205	23–29	13	17–24	11–22
XD40	250	340	35–44	19	27–35	30–56
XD48	400	545	34–43	19	41–52	30–75
XD70	1,000	1,360	95–125	81	68–86	150–375
QS30	150	205	23–29	13	17–27	11–22
QS40	250	340	35–44	19	27–35	30–56
QS48	400	545	34–43	19	41–52	30–75

Data from FLSmidth 2014
*Data is for the G5 bowl only.
†CD = Center Discharge; XD = Extended Duty; QS = Quantum Series.

Table 14 Operating data for the batch Knelson laboratory units

Model	Feed Solids, kg/h	Feed Volume, L/min	Concentrate Solids	Concentrate Volume	Fluidizing Water, L/min	Motor, kW
MD7.5	680	95	1,200–1,600 kg	0.7 L	45–68	0.6
MD4.5	275	18	200–300 g	0.18 L	11–19	0.6
MD3	45	8	80–150 g	58 mL	0.7–4.5	0.1

Data from Knelson, n.d.(b)



Source: Knelson, n.d.(a)
Figure 26 Knelson CVD separator

through the ragging/screen and are discharged through actuated valves. Key operating parameters include rotational speed, pulse rate, stroke length, specific gravity and particle size of the ragging material, and the hutch water addition rate. In most cases, the unit forms its own ragging material from coarser and heavier feed particles.

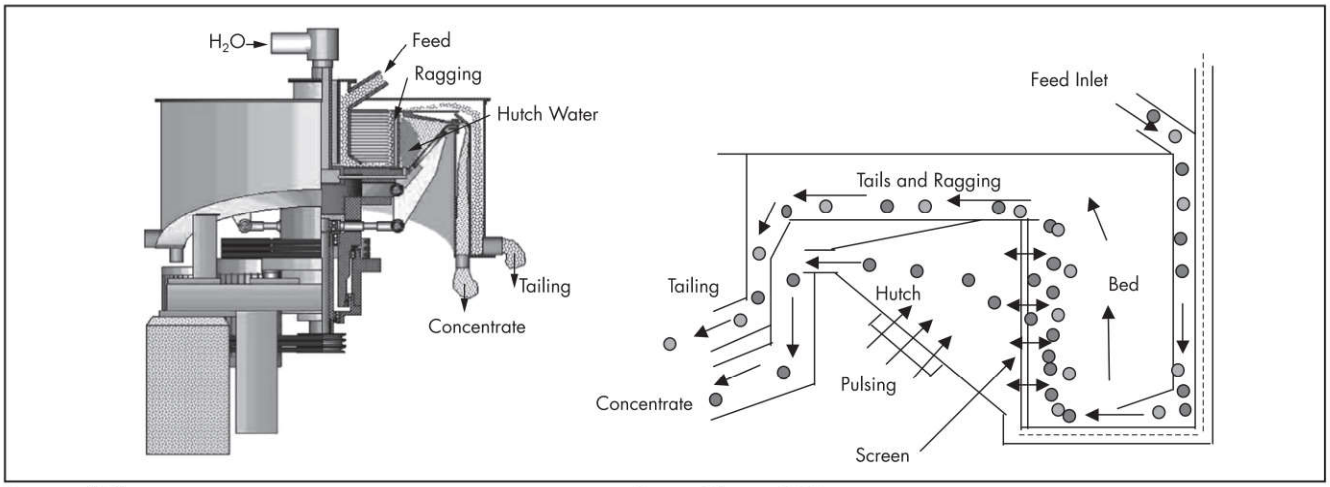
Applications for the jig are provided by Richards and Jones (2004) and Cole et al. (2012). The Kelsey jig is available in three models:

- J200—laboratory test unit, with nominal capacity of 15-100 kg/h solids

Table 15 Operating information for the Knelson CVD units

Model	Typical Fluidization Water Requirement, m³/h	Maximum Total Volumetric Throughput, m³/h	Maximum Feed Capacity (solids), t/h	Feed Density (solids weight), %	Maximum Feed Size, mm	Motor Specifications		Concentrate Mass Yield % of Feed Variable	Concentrator Net Weight, kg
						hp	kW		
CVD64	9–27	635	300	0–50	1.0	100–200	75–150	1–50	18,200
CVD42 1 ring	11–23	250	120	0–50	1.0	40–50	30–38	1–50	7,000
CVD32 2 rings	16–34	170	80	0–50	1.0	40	30	1–50	6,800
CVD20 1 ring	3–9	75	35	0–50	1.0	15	11	1–50	2,500
CVD6 1 ring	1–2	4	2	0–50	1.0	1.5	1	1–50	230

Courtesy of FLSmidth



Courtesy of Kelsey

Source: Falconer 2003

Figure 27 Kelsey jig

- J1300—smallest commercial unit, with nominal capacity of 2–30 t/h solids
- J1800—largest commercial unit, with nominal capacity of 5–60 t/h solids

Multi-Gravity Separator

Perhaps the most efficient EGS is the multi-gravity separator (MGS), which utilizes table riffling principles (Mozley 1990; Tucker et al. 1992). The unit consists of three main components (i.e., cylindrical rotating drum, internal scraper network, and variable-speed differential drive). Selective separations of fine particles are achieved along the internal surface of the rotating drum using the same basic principles employed by a conventional shaking table. However, replacing the table surface with a rotating drum subjects the particles to many times the normal gravitational pull. This feature allows the MGS to separate much finer particles than would otherwise be possible using conventional flowing-film separators. Figure 28 shows a schematic and industrial units of the MGS. The MGS is suitable for the treatment of fines and ultrafines with a maximum particle size of approximately 150 µm and lower limit of ~5 µm. The maximum treatment rate is around 4 t/h.

The MGS has the following operating variables:

- Drum rotational speed or spin (increased spin enhances the centrifugal g-force imparted to the particles, making it more difficult for the particles to move up the drum, hence resulting in a smaller mass and cleaner concentrate).
- Drum stroke length and frequency (increased length and frequency within limits will tend to increase the forces moving the particles up the drum, resulting in a greater mass of concentrate at a lower grade).
- Drum wash water will increase the washing of the slurry particles as they try to move up the drum, thus producing a cleaner concentrate.
- Drum tilt angle (increased tilt will produce a cleaner concentrate).

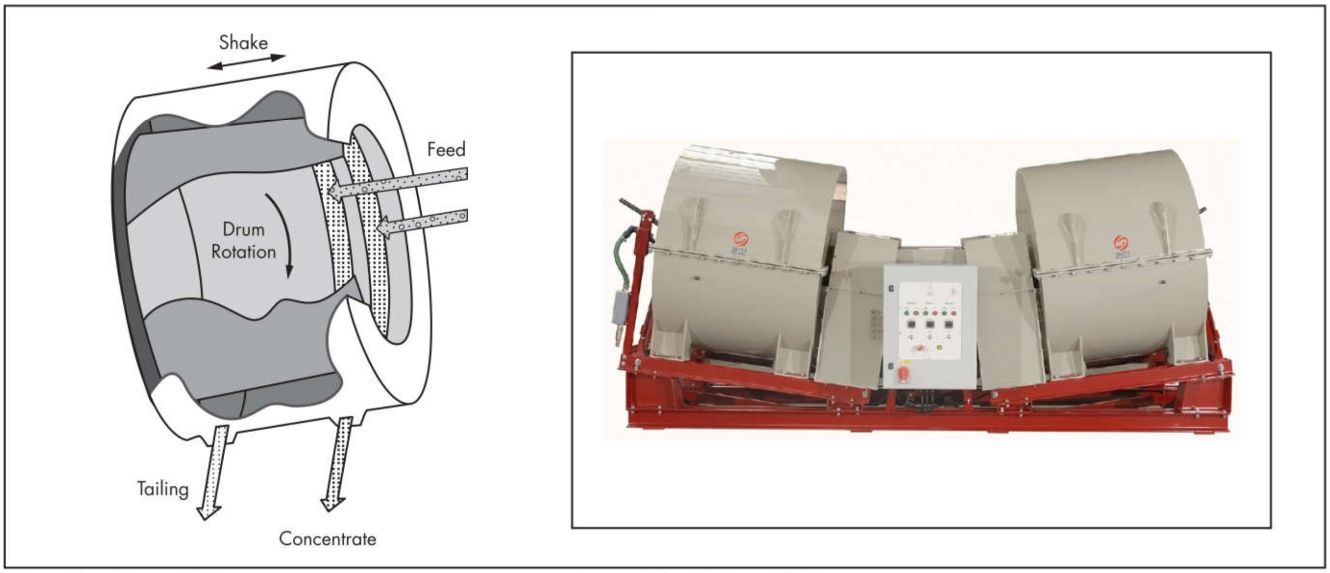
Gekko In-Line Spinner

The Gekko inline spinner (ISP) is a batch centrifugal concentrator. Feed is introduced to the base of the ISP spinning bowl. The bowl operates full of slurry and the rotation causes dense particles to flow into the riffles on the bowl surface. The lighter particles travel up the bowl and exit through a tailings launder. The concentrate is flushed through a central concentrate outlet after the unit is taken off-line. Figure 29 shows an ISP cutaway and Table 16 provides the technical specifications for the ISP. The typical application for the spinner is to upgrade low-grade gold gravity concentrates.

DRY GRAVITY SEPARATION

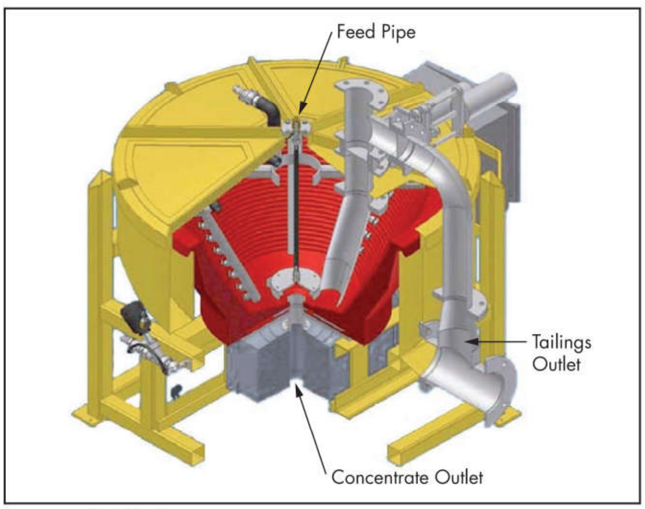
Pneumatic Density-Based Separations

The use of air as a medium to achieve density-based separations was the focus of significant development in the early part of the 20th century. The primary application was in coal



Adapted from Brown 2000 Source: Salter Cyclones 2015

Figure 28 Multi-gravity separator



Courtesy of Gekko Systems

Figure 29 Gekko inline spinner

cleaning, with industrial installations mainly in the United States and Europe. The estimated amount of coal cleaned by pneumatic processes in 1939 was 30–40 Mt/yr (million metric tons per year; Gaudin 1939). According to Arnold et al. (1991), the amount of coal processed in the United States through dry cleaning plants reached a peak in 1965 at 25.4 Mt. The largest dry-based cleaning plant processed 1,270 t/h of –19 mm coal using 14 cleaning units.

Pneumatic density-based separators have been widely applied in the food processing industry and the recycling industry for copper recovery from used electric wire and the separation of minerals. Jarman (1942) reported several potential applications for mineral separations, including the recovery of mica from silica sand, gold from sand, fluorspar from quartz, and calcite and garnet from kyanite. In one application, an "air flotation" table was used to concentrate tungsten in a 1 × 0.5-mm particle size fraction from 4% to 64% WO_3 (tungsten trioxide) with only a trace of tungsten in the tailings, which accounted for 85.8% of the feed.

The pneumatic technologies incorporate the same basic mechanisms used in wet separators, including dense medium separations, pulsated air jigging, riffled table concentration, and air-fluidized launders.

Air Jigs

The early development of pneumatic devices using jigging principles resulted in many technologies, including the Plumb and Hooper jigs (Gaudin 1939). These technologies used either pulsating air through a sieve or a moving sieve in a steady stream of air to generate the jigging action. The most commonly used air jig in the United States was the Stump super air-flow jig. It included a vibrating bed, a single fan with a mechanism for producing pulsations, and a multiple-sink product discharge mechanism. Recent modifications to the Stump jig have enhanced the operational characteristics and performance of the unit. The changes include (Kelly and Snoby 2002)

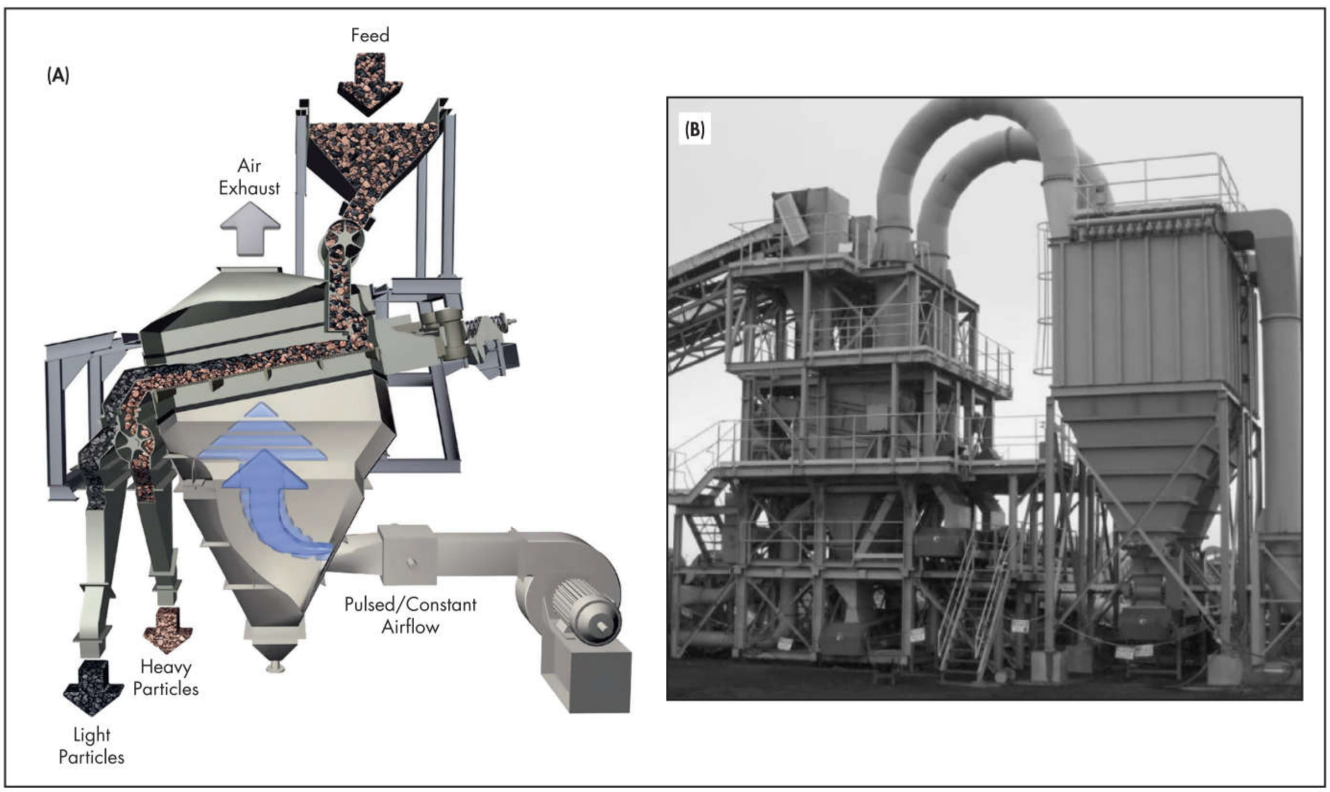
- A star gate to meter homogeneous feed evenly across the entire width of the jigging bed;
- A constant-velocity fan (working air) to loosen the bed of material;
- Installation of external vibrators to enhance particle transport in the jig;
- Addition of a pulsed air fan to optimize stratification by offering independent control of stroke frequency, amplitude, and acceleration; and
- Application of an automatic bed-level control system to achieve a constant separation performance and product quality over large variations in the feed ash content.

The Allair jig shown in Figure 30 has an automatic bed-level sensor that utilizes nuclear density instrumentation. The information from the nuclear density instrument is used to activate a refuse discharge star gate valve to remove refuse at a rate proportional to the amount of impurities entering in the feed. By using only one refuse removal mechanism, the air

Table 16 Gekko inline spinner specifications

	Model		
Details	**Mini**	**02**	**30**
Length, mm	800	1,000	1,650
Width, mm	500	1,000	1,600
Height, mm	710	1,200	1,500
Maximum feed rate, m^3/h	0.25	10	100
Maximum feed rate, t/h	0.05	2	30
Maximum particle size, mm	2	4	6
Typical concentration time, minutes	5–30	5–30	10–30
Typical time off-line for dump, minutes	5	0.5–2.0	1–2
Volume solids per dump, L	<0.5	1–4	8–12
Mass per dump, kg	0.4–0.6	1.5–8.0	12–25
Volume slurry per dump, L	12	45	200
Concentrate discharge	Manual	Automatic	Automatic
Water required per dump, L	5	15	40
Air quality required	Instrument air quality at 600 kPa	Instrument air quality at 600 kPa	Instrument air quality at 600 kPa
Motor speed, rpm	1,440	1,440	1,440
Installed power, kW	0.37	2.2	3
Bowl			
Maximum inside diameter, mm	335	560	882
Depth, mm	100	350	450
Volume, L	6.04	30	160
Material	Polyurethane	Polyurethane	Polyurethane
Revolutions per minute, nominal (40–60 Hz)	100	84–126	88–132
G-force at nominal rpm (top of rotor), g	1.9	2.2–5.0	3.8–8.6

Adapted from Gekko Systems 2010



Courtesy of Allmineral

Figure 30 Allair jig: (A) basic operation and (B) full-scale unit at a coal processing facility

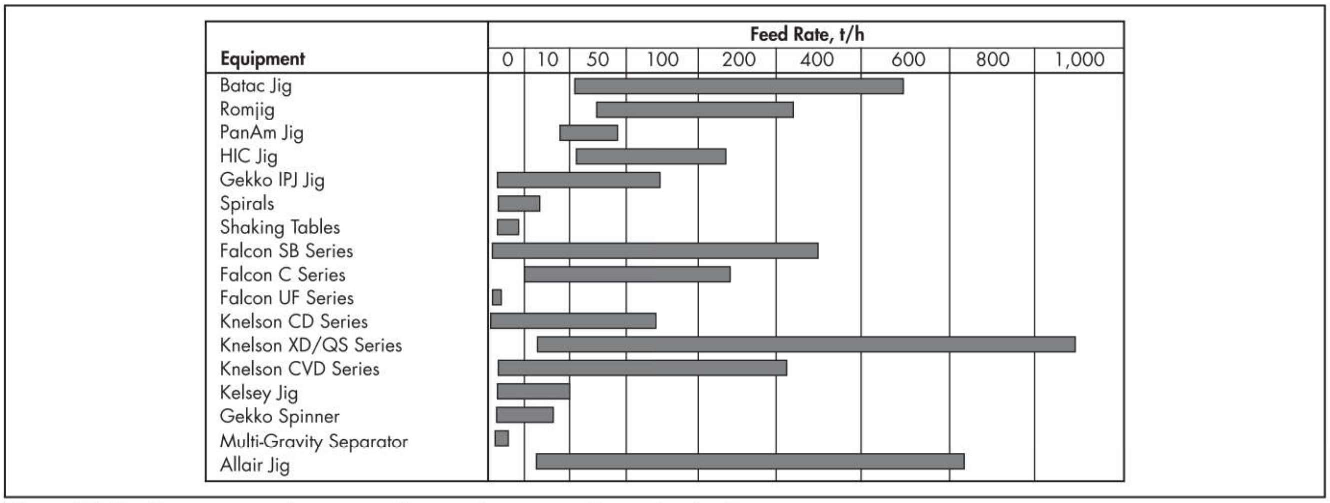


Figure 31 Equipment capacity ranges for gravity separation technologies

jig constantly maintains a reserve layer of high-density material over the outlet. This keeps the lower-density coal particles at a fixed distance from the screen bed, thereby minimizing misplacement.

Several industrial installations use air jigging technology worldwide, including operations located in Spain, India, Brazil, and the United States. Although air jigs are applicable for any lightweight material, most installations involve coal cleaning for the purposes of ash and/or sulfur reduction. Commercial air jigs are capable of treating up to 60 t/h of coarse (75 × 12 mm) coal and up to 40 t/h of fine (12 × 1 mm) coal (Snoby et al. 2009).

Air Tables

Riffled tables using air to create a fluidized bed of high-density fine particles were the most commonly used pneumatic separators in the early 1900s. Common technologies were the Birtley and Sutton-Steel tables. These technologies provided effective density separations for coal coarser than 6 mm and minerals having a particle size as small as 0.3 mm (Jarman 1942). Throughput capacity was around 5 t/h per table. Recent modifications to air tables include vertical suspension of the table, modern dust collection technology, and automation instrumentation, which increased throughput capacity to 120 t/h per table when treating coal in the particle size range of 75 × 6 mm (Lu et al. 2003).

The separation data obtained from coal studies conducted in China indicate that a riffled table has the potential to provide an effective separation for particles as coarse as 80 mm to a lower size limit of around 3 mm. The operational data also indicate that the process is relatively insensitive to surface moisture up to a value of around 7%–10% by weight. Performance evaluations conducted on full-scale units indicate the ability to provide a relatively high separation density (RD_{50}) of around 2.0 RD while achieving probable error values that range from 0.15 to 0.25 (Lu et al. 2003). A detailed evaluation conducted in the United States using a 5-t/h air table unit found that the table has the ability to provide significant upgrading for all ranks of coal (Honaker et al. 2008).

SUMMARY DATA FOR GRAVITY SEPARATORS

Equipment Treatment Rates

Capacities for commonly applied gravity concentration devices are shown in Figure 31, based on more detailed references, primarily in Cole et al. (2012) and Laplante and Gray (2005) as well as equipment specifications provided by manufacturers.

Feed Particle Size Range for Separators

The particle size ranges that can be treated in the more common gravity separators are shown in Figure 32.

GRAVITY SEPARATION TEST WORK

As with all test work, the samples need to be selected and subsampled correctly. Furthermore, samples should be properly characterized in respect to mineral composition, liberation, and chemical analysis. These topics are discussed elsewhere in the handbook.

Gekko-Wilfley Laboratory Table Test

The test procedure applied by Gekko Systems to establish yield–recovery relationships for in-line pressure jig (IPJ) applications is described by Cole et al. (2012). This test can be similarly applied for analysis of any continuous gravity concentration application. A quarter-size laboratory Wilfley shaking table is used to replicate the expected mass and metal recovery from an IPJ (Coward 2007). The IPJ can be configured in an open- or closed-circuit mode in industrial applications, and the laboratory procedure can simulate both modes of operation. The simulation of a closed-circuit process consists of four steps:

1. A sample preparation of the feed (usually 20 kg of –1-mm material) is fed as a slurry onto the shaking table and a table concentrate of ~10% of the feed mass is collected.
2. The table tailings are dewatered and the tailings are ground to a particle size P80 of 500 µm. The ground product is re-tabled and, again, a concentrate of ~10% of the feed is collected.

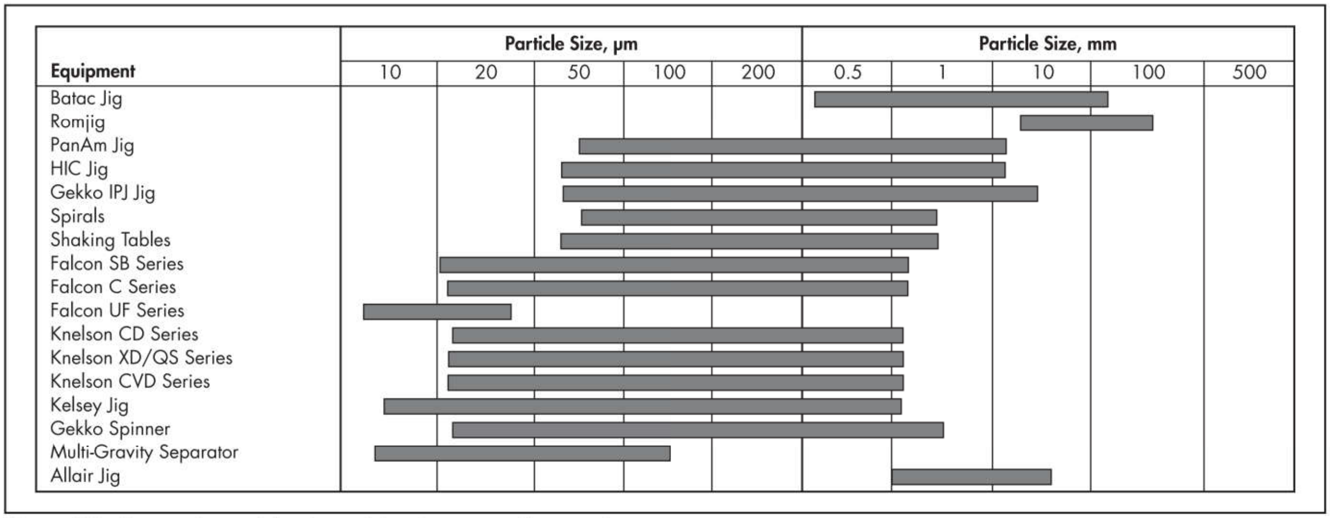


Figure 32 Comparison of feed particle size range for gravity separators

3. The table tailings are collected, dewatered, and reground, this time to a particle size P80 of 106 µm before tabling again. A concentrate of around 5% of the feed mass is collected for this test. The table tailings from this test are the final tailings product, and it is filtered, dried, and portions removed for chemical analysis.
4. The three table concentrates from steps 1, 2, and 3 are combined (i.e., some 25% of the original mass) and then re-tabled. Four samples are collected from this test (high-grade, medium-grade, low-grade, and low-grade tailings). The products are filtered, dried, and weighed, and assay portions are removed for assaying.

The data from the test-work program are used to develop a cumulative concentrate mass recovery versus metal recovery curve and allows for a decision to be made on the optimum mass of concentrate to be collected for the ore being evaluated. For an open-circuit IPJ simulation, a very simple shaking table procedure can be adopted whereby only a single test is performed and four concentrates and a tailing are collected.

Test-Work Methodology for Knelson and Falcon Centrifugal Separators

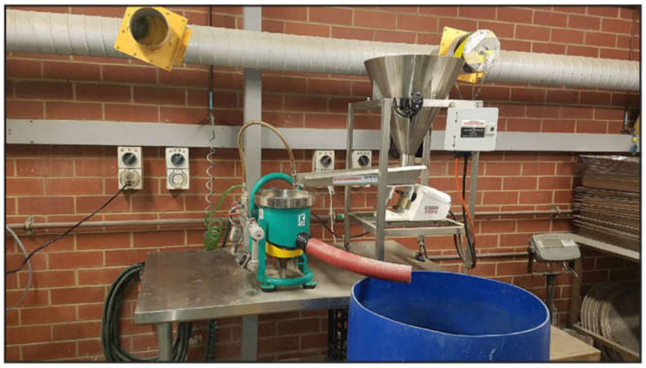
The laboratory test procedure for both the Knelson and Falcon laboratory batch machines are similar; however, operating parameters (water addition and agitation speed [g-force]) are particular to each machine as well as the ore being tested (Cole et al. 2012). The operating manuals provided with the laboratory machines provide guidance as to the operating parameters for setup. The design and operating parameters of the batch laboratory machines are similar to those of the larger industrial machines as described previously. The quantity of concentrate produced from both the Knelson and Falcon laboratory machines is similar, around 100 g (±20 g), depending on the relative density of the concentrate being collected. The volume of concentrate collected remains constant regardless of the amount of material fed to the machine. On this basis, the concentrate mass percent generated during a gravity test is mainly a function of the feed mass used for the test. For example, if 10 kg is fed to the batch machine, then the mass recovery to the concentrate (~100 g) will be 1%. For 1 kg of feed, the mass recovery to the concentrate will be around 10%, whereas 0.1% mass of concentrate is provided when 100 kg of feed are treated in the batch machine. The fluidization water flow rate is best determined by inspecting the concentrate retained in the riffles of the bowl after a setup test. The concentrate should not be packed hard but should just start to slump out of the lower riffle. Too high a water pressure will give a low concentrate mass, while too low a water pressure will have a higher-than-optimal mass recovery. Generally, the applied g-force is a function of the feed particle size. For coarse particle size, the g-force should be low (<50 g), whereas for very fine particles, it will be high (>200 g). The maximum particle size of the feed sample to the batch machines is usually around 1 mm.

Continuous Separations—Continuous Knelson Test

A procedure using a laboratory-scale Knelson concentrator is described by Sakuhuni et al. (2014) to establish mass (yield)–recovery relationships for continuous applications of centrifugal concentrators. This gravity release analysis consists of a series of rougher-scavenger-cleaner gravity tests based on the 3-in. laboratory Knelson concentrator (model MD3). The starting feed sample is a representative 5-kg (–1 mm) sample riffled from the original sample. The 5-kg sample is further riffle split into 1-kg subsamples. Each 1-kg subsample is treated twice in an MD3 concentrator, collecting the concentrate after each run. The tailings from each subsample are combined to constitute the final tailings. The concentrates are combined to constitute an ~1-kg rougher concentrate which is rerun in the MD3 four times, collecting the concentrates after each run. The concentrates are weighed and assayed for the target elements. The final tailing is weighed and a representative sample is assayed for the required elements.

Gravity Recovery Gold Test Procedure

The most commonly applied test procedure for precious metals gravity-circuit design and optimization is the gravity-recoverable gold (GRG) test procedure described by Laplante and Clarke (2006) and Giblett et al. (2013). The GRG test requires a large sample of ore (typically 80–100 kg) to be processed through a Knelson manual discharge MD3 concentrator at three successive liberation sizes, known as



Courtesy of Gold Technology Group, Curtin University
Figure 33 MD3 treatment circuit for gravity test work

Stages 1–3. In Stage 1, the entire sample is crushed to 100% –850 μm and processed through the concentrator, generating a Stage 1 GRG concentrate. The Stage 1 tailings are then split to produce a smaller sample (25–35 kg), which is ground to 45%–55% passing 75 μm and then run through the Knelson MD3 in Stage 2 of the test, producing a Stage 2 GRG concentrate. Finally, the Stage 2 tailings are ground to 80% passing 75 μm and run through the MD3 again, producing a third (Stage 3) Knelson concentrate. A final mass balance determines the total amount of gold recovered as GRG and the percentage recovered in each individual stage of the test. In the absence of sufficient information to estimate the required sample mass, the guidance of Laplante and Doucet (1996) should be followed and a sample mass of between 40 kg and 70 kg selected for initial gravity testing, ideally following the Laplante GRG test procedure (referenced to avoid confusion with the single-stage test). However, for samples of lower gold grade (<1.5 g/t) and coarse gold particle size, up to 150 kg of sample may be required (Laplante 2000). Where the single-stage GRG test is to be used for gravity gold characterization, such as for variability testing, a smaller sample mass of nominally 20 kg can be applied (Laplante and Clarke 2006). Interpretations of test results are provided in Laplante (2000) and Giblett et al. (2012).

The Knelson MD3 concentrator is the recommended laboratory unit for the GRG test. Figure 33 shows an installed Knelson concentrator with dedicated feed bin and feeding system to allow consistent material flow to the concentrator. Access to water for feed dilution, concentrator fluidization, and cleanup is required. The ability to handle large volumes of dilute gravity unit tailings and to extract dust from transfer points when working with dry feed should be considered in the laboratory system design.

Gravity Circuits for Precious Metal Recovery

Gravity circuits in precious metal recovery applications are typically situated in the grinding circuit to allow the recovery of free gold before sufficient damage or coating of the gold particles occurs to make gravity separation, flotation, or cyanidation less effective. Where the grinding circuit involves semiautogenous grinding and ball milling, the gravity circuit will typically be located in the secondary grinding stage. Where flotation is applied downstream, it may also be feasible to apply gravity separation in the concentrate regrind circuit, where a recirculating load of finer free gold particles may occur as a natural result of classification by hydrocyclones (Fullam 2010).

Gravity concentration has also been successfully applied to the treatment of carbon-in-pulp tailings in gold flow sheets to recover gold-bearing sulfides (Delahey et al. 1992; Butcher and Laplante 2003).

The treatment of gravity concentrates by tabling or intensive cyanidation often requires a batch processing approach, which is facilitated by having suitable concentrate storage capacity to ensure the continuity of the process. Where batch-intensive cyanidation is employed, there is a need for sufficient concentrate storage volume to match the full capacity of the leach reactor. This allows concentrate to be accumulated in preparation for the next leach cycle and therefore metal production rates to be maximized without leach reactor downtime due to insufficient concentrate stocks. The efficiency of upgrading gravity concentrates by shaking tables is significantly improved by having a consistent solids feed rate and pulp density. Ideally, sufficient concentrate storage should be provided ahead of the table to ensure a constant feed rate during the operating cycle of the table.

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