CHAPTER 3.8

Grinding Circuit Design

Aidan Giblett and Brian Putland

Other chapters in this handbook reviewed the main grinding technologies used in mineral processing and under which circumstances they will be a more practical flow sheet component than others. For any project, the optimum grinding flow sheet will be subject to many influences and will require detailed consideration of numerous elements, including process plant capital; operating costs; specifics of metal recovery processes; product quality requirements; and geographical, social, and environmental factors. These project-specific considerations can be even more significant than the technical criteria in many cases and have a significant bearing on final grinding technology selection.

This chapter focuses on the technical factors that must be considered during project development to optimize the initial process design, after the preferred grinding circuit flow sheet has been selected. These technical considerations are critical to the successful performance of the project post-commissioning. The authors strongly believe in the value of robustness when it comes to sample selection and ore characterization protocols, and the application of well-validated methods for the sizing and selection of grinding mills. Failure to execute these stages of development with the required level of diligence can have serious ramifications for the profitability of the project. Professionals can readily speculate how many of the circuits that are currently secondary-crushing their semiautogenous grinding (SAG) mill feed could have avoided the added capital expense of that retrofit, and the preceding years of below-nameplate plant production, if they had been more rigorous in their initial ore characterization or in the selection and application of methods to size the original grinding mills.

Fundamentally, the selection of a specific mill for a given grinding application requires several critical inputs:

- Sampling and testing to measure the hardness characteristics of the deposit
- Consolidation of this ore characterization data to establish a design basis for the mill sizing
- Identification of baseline mill operating conditions
- A method to estimate the specific grinding energy requirement for each stage of liberation
- A method to scale the specific grinding energy requirement to a mill input power
- A reliable model to predict the power draw of a mill, and thereby to confirm the selection of an adequate mill size for the task

The methods and protocols of determining the specific energy for a given grinding duty are covered in detail in Chapter 3.12, "Testing and Calculations for Comminution Machines." Implicit in this calculation is an understanding of the input and output particle size distributions of the grinding circuit. In addition, selecting the grinding mills will require estimation of specific operating conditions such as mill rotational speed, ball load, total volumetric filling, sizing screen apertures, recirculating loads, and mechanisms for managing critical size material. Several of these parameters will affect the power draw of the mill; others will influence the specific energy calculation of primary mills depending on the calculation method used. Additional factors that influence mill operability, availability, and grinding efficiency are discussed in this chapter. These include motor selection, liner selection, sizing and classification system design, and grinding media size.

DETERMINING THE DESIGN POINT

The significance of good sampling and testing protocols to determine reasonable grinding circuit design assumptions cannot be understated. The penalties for executing a limited or poorly designed sampling and testing campaign as a basis for grind circuit development can be severe. Production losses and the need to spend significant additional capital on expensive plant upgrades after start-up to achieve target production levels are heavy penalties to pay. In the worst cases, a poorly executed testing program can lead to project failure. The cost of a well-considered, inclusive sampling and testing program will usually be validated by predictable plant performance, achieved in a short time frame after start-up.

The tests that may be applied for grinding circuit evaluations are covered in detail in Chapter 3.12, "Testing and Calculations for Comminution Machines." In respect to which

Aidan Giblett, Senior Technical Advisor—Mineral Processing, Newmont Mining Services, Subiaco, Western Australia, Australia
Brian Putland, Principal Metallurgist, Orway Mineral Consultants, Perth, Western Australia, Australia

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samples and how many should be selected for testing, there is no agreed standard. However, as the required accuracy of project design increases, the number of samples tested must also increase. The broad concepts of sampling a deposit in support of grinding circuit design are generally well accepted:

- Major ore domains must be defined in meaningful terms. Typically, especially early in the development of a project, they are defined in geological terms, reflecting characteristics such as host rock types (lithology) and alteration zones. A diligent testing program will challenge the metallurgical validity of the initial geological domains and, as necessary, establish more robust geometallurgical classifications for the deposit. As discussed in Chapter 1.10, "Geometallurgy," a valid geometallurgical domain will exhibit consistent properties throughout the project life. This will, in turn, lead to a consistent and predictable performance response in the process plant when treating either single-domain ore or domain blends. The cost of establishing valid geometallurgical domains is often seen as large, but it is small compared to both the cost of defining ore body geology and the present value of a robust project. Moreover, it is an investment that continually generates project value through reduced design risk, rapid project commissioning and increased confidence in mine planning, and estimating project optimization benefits.
- Sufficient samples within each major domain should be tested to reliably represent the typical domain characteristics as well as the extremes of ore hardness. Ideally, sufficient samples should be tested to allow definition of the full range of variation within each domain at least in spatial terms, and with depth in economic zones of the deposit. Good guidance on the quantity of hardness tests that should be initially conducted is given by Morrell (2011), who specified a minimum of 10 samples where low variability is observed and a minimum of 40 samples when high variability is observed. This guidance is best applied to individual ore domains rather than entire deposits to ensure that sources of variability within the deposit can be isolated and studied in detail. In design studies these initial numbers may be increased as the level of detail required by the project increases from the scoping stage to final bankable feasibility (Lane et al. 2013). It is particularly important that ore variability in the early (payback) years of the project is fully defined, to ensure the financial success of the project.
- Domain hardness modeling must be sufficient to generate a realistic prediction of the hardness progression by period (months, quarters, or years) throughout the project life. Important periods underpinning a commercially successful project are the commissioning period (typically year 1) and the payback period (typically up to year 5). If there are insufficient samples in the set representing the commissioning period, in particular, then urgent supplementary testing is necessary on suitable samples from within the commissioning ore zone. A reliable hardness model can guide designers in the scheduling of milling expansions, allow the evaluation of direct-tip (single handling) and blended-ore (double or even triple handling) plant feeding designs, and may result in a rescheduling of the mine plan to smooth annual production capacity, if the geology and mining allow. Every time the mine plan changes during design, the hardness prediction model needs to be rerun.

Because of the frequently observed variability in ore properties with increasing deposit depth, maintaining the accuracy of geometallurgical models in operating plants requires ongoing sampling and testing of samples, often on a yearly basis. It is not unusual in larger deposits for 100–200 samples to be taken and tested for this purpose per year. The number of samples required is dictated by the variability of the deposit and, contrary to what some literature suggests, is not related to the type of comminution test performed.

In some cases, typically early-stage studies, there may be insufficient information to divide the deposit into well-defined domains that reliably define the physical characteristics of the ore. In these instances, the hardness test results will typically be lumped together and a single design point chosen for provisional grinding equipment sizing and development of the capital estimate. In this case, a conservative design point should be selected to provide confidence that the grinding equipment size will be adequate to accommodate variations in hardness through the deposit. The conservative design point may reflect a database percentile approach, where a higher value than the average, such as the 75th or 85th percentile value of the data set, will be chosen. Where the hardness data set is very small, perhaps consisting of only a handful of results, the design basis is often selected as simply the hardest value in the data set. Such design approaches recognize and address the risk of false representation of the deposit's hardness characteristics that may be provided by small data sets.

As the project progresses, the intent should be to conduct sufficient hardness tests on carefully selected samples chosen to represent the observed variations in geological characteristics, deposit depth, and spatial distribution to provide statistically robust hardness data sets by ore domain. Under such circumstances, the average ore domain characteristics can be combined with tonnage distributions by domain in the mine plan to provide a robust basis for the sizing of grinding mills. When the average ore characteristics of a resource are used as the design basis, it is common to also apply a safety factor to the mill sizing. This factor is normally applied to the estimated grinding power requirement, which provides the design contingency that may otherwise be given using a more conservative percentile value design basis. Safety factors of 10%–20% are typically applied depending on the extent of testing performed and the design point selected. Larger safety factors will be applied when database average ore characteristics are used for the design point, and in early stages of project evaluation when the testing data set is small and less likely to accurately reflect the distribution of ore characteristics through the resource. In deciding to use these factors, the reader must understand that there is an inherent minimum precision of each testing method, limited accuracy of the design methods, and an inability of operating plants to maintain ideal conditions in the process at all times. All these risks and inefficiencies must be understood in selecting a design contingency and operating envelope that will deliver a circuit that will achieve the desired annual production capacity.

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DETERMINING THE GRINDING ENERGY REQUIREMENT

The methods available to determine specific energy are discussed at length in Chapter 3.12, "Testing and Calculations for Comminution Machines." The most appropriate method of estimating specific energy can vary depending on the duty at hand. For example, there is no contention that the methods of Bond and Rowland are the optimal approach to size a rod mill. The approaches of Bond, Rowland, and Morrell are suited to sizing ball mills in primary or secondary grinding duties. Several methods, most at least to some degree proprietary, are currently used for the sizing and selection of autogenous grinding/semiautogenous grinding (AG/SAG) mills. In the case of stirred milling and compressive grinding technologies, machine-specific in-house testing procedures and equipment sizing methodologies have been developed that require significant vendor participation in the mill sizing process before the vendor will consider warranting equipment performance.

DETERMINING THE MOTOR POWER REQUIREMENTS

After determining the mill-specific energy requirement at the pinion, or mill shell, as estimated by common power-based calculation methods, the user must convert this figure to a motor sizing that will supply the power demand of the plant at the target production rate. The final motor selection will depend on the design-specific energy requirement (S) in kilowatt-hours per ton, design throughput rate (t) in metric tons per hour, and factors for motor power utilization (PF_1) and drivetrain efficiency (PF_2) by the following equation:

$$\text{motor power, hp} = \frac{1.341 \times S \times t}{PF_1 \times PF_2} \tag{EQ 1}$$

In operation, variability in the feed plus other plant-based disturbances affect the power draw of tumbling mills, causing it to fluctuate. Consequently, it is not possible to operate mills for any period at their maximum rated output of the motor for fear of overloading the motor when these variations cause high power excursions. The long-term sustainable average power draw of mills is therefore, by necessity, somewhat lower than the motor rated power. The result is that the maximum power that the motor can deliver is rarely fully available for ore processing purposes, and predictions of sustainable throughputs of grinding mills should not use the maximum motor power. For AG/SAG mills, a reasonable assumption is that the sustainable average power draw that will be available for long-term throughput is approximately 90% of the installed motor capacity, although this can approach 0.95 for well-controlled variable-speed mills. This factor can also be as low as 0.85 for ores with highly variable properties. For ball mills, 95% is normally assumed because of the more stable operating environment for these machines. As a result of this stability, the occasional ball mill draws greater than design motor output if the shell size is sufficient and motor winding temperatures are carefully monitored. One issue with ball mills is that the shell dimensions are often undersized, and therefore the mill is incapable of drawing full installed power. This is a waste of applied capital in the design, and sizing should be scrutinized when selecting a ball mill. Typical values for PF_1 are 0.85–0.90 for SAG mills and 0.95 for ball mills.

Drivetrain efficiency (PF_2) reflects the difference between the motor input power and the power delivered to the mill shell or pinion, the latter being the value generated by traditional calculations of specific energy. This efficiency is a function of the type of motor used and the transmission system. Many technical articles describe how system efficiency varies for each of the common drive systems, notably Grandy et al. (2002) and Ploc and Peters (2010). Those authors ascribe similar drivetrain efficiency values, in the context of PF_2 influencing the transfer of motor input power to the pinion shaft, as the efficiency associated with the motor, gearbox, slip system, and ring gear components for the most common drive systems. From this analysis, typical PF_2 values for wound rotor induction motor systems will be 0.93, for low-speed synchronous motor systems will be 0.94–0.96, and for gearless drive systems will be 0.96–0.97.

Ancillary equipment may also be required upstream of the motor for to it operate correctly. For example, a variable-speed single pulse width modulated (PWM) synchronous motor requires an additional transformer and variable-frequency drive, both of which will consume power, approximately 4% total as shown in Table 1. Such ancillaries will not directly influence the size of the motor chosen but must be considered when determining the overall capital cost and total electrical operating cost.

DETERMINING THE MILL DIMENSIONS

After the required motor has been estimated, the dimensions of the mill(s) required to deliver the design power can be estimated using a tumbling mill power model, such as the Morrell model described in Chapter 3.12, "Testing and Calculations for Comminution Machines." Mill vendors also have their own calibrated power draw models. Care must be taken to use practical estimates of mill speed, ball charge levels, and total mill filling to ensure that the design power draw can be achieved under sustainable operating conditions. For example, if the design power draw requires the mill to operate at high speed, high ball charge, and/or high volumetric filling, there is little allowance for normal variations in operating conditions and the difficulty of maintaining precise control over charge levels.

In the case of AG/SAG mills, aspect ratio has a significant influence on the estimated specific energy and is discussed in the following section in more detail. Therefore, the aspect ratio that is used for estimating specific energy must also be used when choosing the mill dimensions to deliver the required power. Conversely, some iteration may be required to ensure that the specific energy values and power draw assumptions used are applicable to the final mill dimensions.

Alternatively, the mill can be estimated with reference to mill installation lists available from mill suppliers, based on the target mill motor size. If a power model is used to determine the mill dimensions, it should also be cross-referenced against the vendor's equipment lists to validate the selected dimensions. A mill with the dimensions and power that a vendor has built previously will be less expensive and probably be delivered more quickly than a mill with dimensions that the vendor has not engineered in the past.

Ultimately, the mill vendor will be charged with supplying a mill that can achieve the design power draw under practical operating conditions. Nonetheless, the project engineer must ensure that there is adequate contingency in the mill design so that the required performance is achieved, that the operating conditions assumed in design remain appropriate, and that the selected mill can deliver the expected performance, as the

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Table 1 Mill drive system efficiency

Speed	Type*	Component Efficiency, %						
		Transformer	Variable-Frequency Drive	Motor	Gearbox	Ring Gear	Other	Total
Fixed	Single synchronous	N/A†	N/A	96–97	N/A	98.5	N/A	94.6–95.5
	Single WR	N/A	N/A	95–96	99	98.5	N/A	92.6–93.6
	Dual quad	N/A	N/A	96–97	N/A	98.5	N/A	94.6–95.5
	Dual WR	N/A	97	95–96	99	98.5	97‡	89.9–90.8
Variable	Single PWM synchronous	98.3	97.8	96–97	N/A	98.5	N/A	90.9–91.9
	Single WR	99.7	99.7	95–96	99	98.5	N/A	92.1–93.1
	Single PWM induction	98.3	97.8	96.5–97.5	99	98.5	N/A	90.7–91.4
	Dual PWM synchronous	98.3	97.8	96–97	N/A	98.5	N/A	90.9–91.9
	Dual WR	99.7	99.7	95–96	99	98.5	N/A	92.1–93.1
	Dual PWM induction	98.3	97.8	96.5–97.5	99	98.5	N/A	90.7–91.4
	Gearless	98.8	99	96–97	N/A	N/A	99§, 99.5**	92.5–93.5

Source: Ploc and Peters 2010
*PWM = pulse width modulated; WR = wound rotor induction motor
†N/A = not applicable
‡Slip resistor
§Harmonic filter
**Motor ventilation

owner—not the vendor—will bear the real cost of any loss in available operating power over the life of a project.

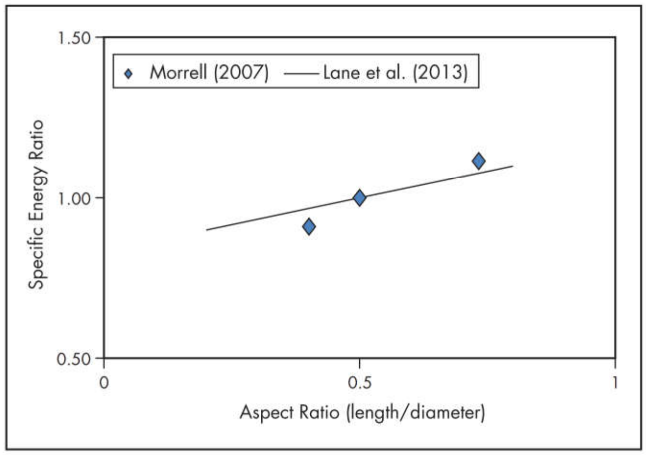
MILL DESIGN ASSUMPTIONS

AG/SAG Mill Aspect Ratio

Popular power-based modeling approaches, such as those applied by Morrell (2004; 2006) and Lane et al. (2013), observe that mill aspect ratio has a material impact on AG/SAG mill capacity when operating in primary grinding duty, as illustrated by Figure 1. When discussing mill aspect ratio, note that aspect ratio is quoted in terms of either mill length divided by mill diameter (L:D) or mill diameter divided by mill length (D:L). In this discussion, the diameter-over-length methodology is observed, defining high D:L ratios for high–aspect ratio mills. Figure 1, however, observes the L:D representation, reflecting the visualization produced by Lane et al. (2013).

For the same power draw and feed characteristics, a high–aspect ratio mill will have a lower specific energy, higher throughput, and coarser transfer size than a low–aspect ratio mill when operated in open circuit. This will be reflected in a lower specific energy ratio for the low aspect mill, with reference to Figure 2. For example, a 30 ft × 15 ft effective grinding length (EGL) SAG mill has, by definition, an aspect ratio (D:L) of 2 and a relative specific energy factor of 1.0. Should the mill length be increased to 20 ft to increase the mill power draw, the aspect ratio decreases to 1.33 D:L, and by Figure 2 the specific energy ratio is nominally 1.1 for L:D = 0.75. This reflects a 10% increase in SAG mill specific energy and a commensurate reduction in SAG mill unit capacity. The precise impact of aspect ratio on SAG mill specific energy will be subject to other factors; however, the fact that aspect ratio will influence open-circuit SAG specific energy should not be overlooked.

This effect must be considered when sizing both primary and secondary mills, as these variations in transfer size determined by the primary mill aspect ratio will vary the work load of the secondary grinding stage. Should it be necessary to increase the primary mill power draw at any point of the design process to accommodate harder ore at sustained throughput rates, the designer should attempt to conserve the primary mill aspect ratio wherever possible, or otherwise allow for the



Adapted from Lane et al. 2013

Figure 1 Influence of SAG mill aspect ratio on specific energy

impacts of altering the aspect ratio on primary mill capacity and the transfer size to the secondary grinding stage.

In addition to the estimation of specific energy, modeling of grate and pulp lifter slurry flow capacity should be taken into account when selecting the aspect ratio of the mill, as discussed by Putland et al. (2011) in relation to the design of single-stage SAG mills. Grate open area is particularly important in the case of single-stage SAG mills or very low specific energy open-circuit SAG mills. The diameter of the mill should be selected first based on the required slurry flow with typical grate and pulp lifter performance. The mill length is then selected to achieve the required power draw. The use of too small a mill diameter will result in slurry pooling, which will negatively affect power draw. This may not be a significant problem if considered in the initial selection and design of the mill but can be an issue if the mill is unable to generate design power draw, resulting in reduced capacity. The presence of a slurry pool does not necessarily affect grinding efficiency for single-stage mills, as most of the required energy input is low in intensity. However, inefficiencies and overgrinding can occur when attempting to achieve a coarse

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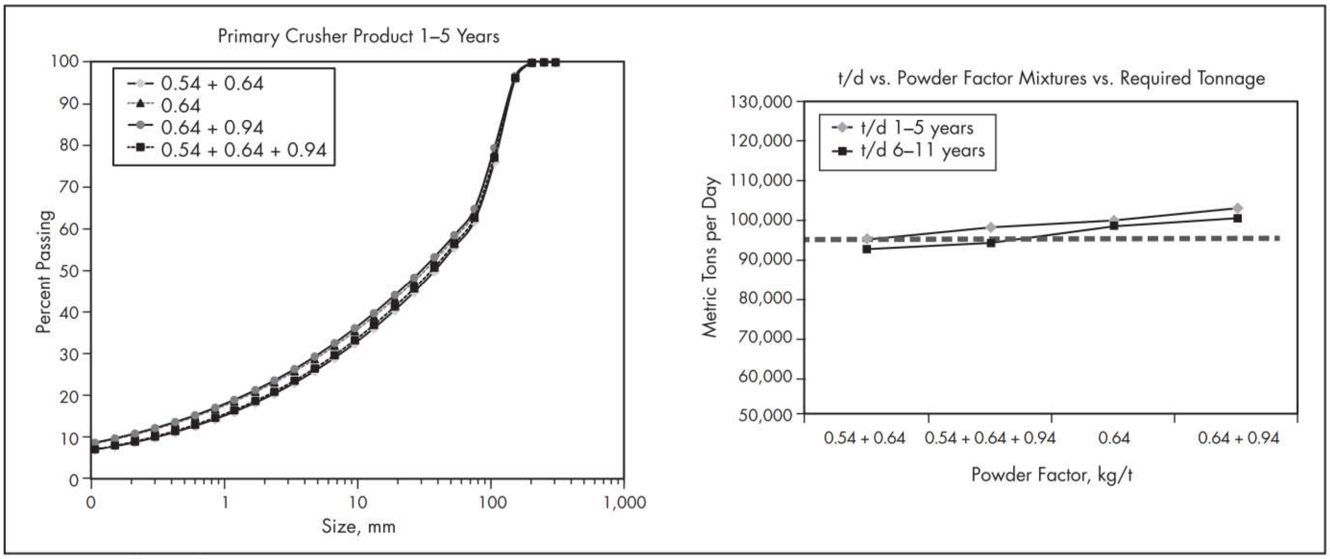

Source: Fernandez et al. 2015

Figure 2 Impact of blast design on run-of-mine size distribution and milling circuit capacity

grind from a competent ore and the slurry pool is providing a cushion, reducing the effectiveness of coarse ore breakage. The issue with accurate assessments of design slurry flow requirements is estimating an appropriate circulating load when not determined by a population model or piloting. In general, very high circulating loads or a coarse grind are generated by a lack of media, while very low circulating loads and overgrinding result from excess coarse rock. These two conditions should occur only if the incorrect single-stage SAG mill configuration has been selected. In the absence of these extreme conditions, circulating load is primarily influenced by the cyclone configuration. The size of cyclone, pressure, and conditions (spigot and vortex selection) that need to be operated to achieve the product size within the target cyclone overflow range determine the circulating load range. Standard vendor cyclone models can be used to estimate optimum cyclone operating conditions and therefore predict the likely circulating load range.

Mill Feed Size Distribution

In practice, mill feed size distribution has a significant impact on AG/SAG mill capacity. Morrell and Valery (2001) provided an example of a SAG mill where feed size variations, in F_{80} terms, of 3–4½ in. were consistent with tonnage rate and SAG specific energy variations of up to 30%. The same authors also illustrated the impact of variations in the fine end of the size distribution on mill capacity, with large variations (50%) in mill throughput achieved for similar F_{80} values, because of a measurable difference in fines content in the SAG mill feed. The amount of material in the SAG mill feed that is finer than the SAG mill discharge screen aperture is an important performance indicator, as this represents material that should readily pass through the SAG mill to the secondary grinding stage, increasing the primary mill capacity.

Such concepts are at the core of mine-to-mill optimization programs and require the design engineer to consider factors that influence mill feed size distribution during the mill design process. It is critical that the design consider the influence of in situ ore parameters, such as fracturing and compressive strength, in addition to drill-and-blast program parameters, such as powder factors, drill-hole diameter, drill-hole spacing, and stemming practices. At the design stage, this requires liaison with the geology and mining engineering groups, and the use of blast simulation software by a suitably skilled practitioner, to ensure these important influences on run-of-mine (ROM) ore distribution can be incorporated in the mill design. The influence of a full ROM particle size distribution data set on grinding circuit performance can only be determined by the application of a population balance simulation process. Such a detailed simulation approach to demonstrate the resultant impact on milling circuit capacity can identify situations where modifications to a standard mill design approach will be required to accommodate deposits and mining strategies that may deliver "nonstandard" ROM feed size distributions. A recent example of such a study was provided by Fernandez et al. (2015), who applied blast simulation software and population balance modeling of the comminution circuit to demonstrate the influence of blast design and geotechnical characteristics of the ore on the ROM size distribution, and subsequently the grinding circuit capacity. Figure 2 reflects the outcomes of this analysis, showing the variation in ROM size distribution and the magnitude of impact on mill capacity. The methods are equally applicable to the optimization of fragmentation in operations and the validation of mill sizing for greenfield projects. Care must be taken to not use an overly optimized design for greenfield projects, as the marriage of optimal mine fragmentation and mill operation (grate and lifter design) may take years to achieve and will impact the ramp-up of a project if used as the basis for design.

Critical Size Management

When the flow-sheet design considers primary autogenous or semiautogenous milling, it will be necessary to estimate how much critical size material will be generated during the primary grinding stage and determine how this material will be managed. In both autogenous and semiautogenous milling, the accumulation of critical size material has been shown to be detrimental to mill capacity as this material is,

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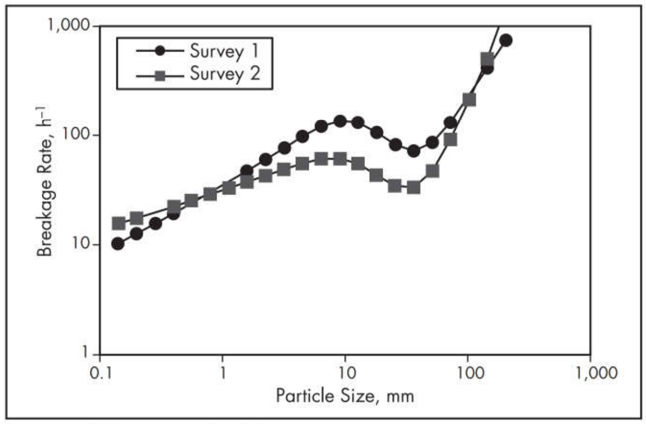
by definition, slow to grind in the mill. The relative grinding rates of critical size material are effectively illustrated by the typical JKSimMet SAG mill breakage rate curve (Figure 3), which reflects a deterioration of breakage rates in the ½–3 in. size range. While operational considerations can influence the position and relative depth of this trough in the breakage rate distribution, it will exist in some form for all autogenous mills. The most effective method to address the negative impacts of critical size accumulation is to implement pebble crushing in combination with the use of large aperture grates, more commonly termed *pebble ports*. Pebble ports allow ready removal of this critical size material from the AG/SAG mill and more effective breakage by pebble crushing. Pebble crushing is a very effective means to reduce the critical size material to the point where it can be more effectively ground. Ideally, the pebble crusher closed-side setting or gap will be set in concert with the AG/SAG discharge screen panel aperture to produce material that passes the AG/SAG mill discharge screen and directly reports to the secondary grinding stage. Crawford et al. (2009) demonstrated the material impact of effective pebble crushing to produce SAG screen undersize material on a semiautogenous grinding mill, ball mill, and pebble crusher (SABC) circuit throughput at the Telfer gold mine in Australia.

The extent to which an autogenous mill will produce pebbles varies as a function of the mill feed size distribution, the strength of the ore, and the mill operating conditions. To appropriately design the pebble handling system and then determine the size of the pebble crusher, the engineer must estimate the pebble production rates of the mill, normally expressed as a percentage of the fresh mill feed rate. In many cases, the use of generic assumptions based on the pebble production rates of other facilities has been unsuccessful and often leads to an oversized pebble crushing circuit. An oversized pebble crusher is difficult to choke feed without batch processing, which can be detrimental to AG/SAG mill stability. Inability to choke feed makes the crusher prone to rapid and localized liner wear, which in turn reduces the crusher's ability to maintain the target product size gradation. Underestimating the pebble production rate to the extent that the pebble handling system is undersized is similarly undesirable, as the material handling capacity of the pebble crusher feed system can become the milling circuit throughput bottleneck. A good initial estimate of pebble handling system requirements can be generated from rule-of-thumb pebble production estimates, such as 14.5 m^3/h of pebbles per square meter of pebble port open area as provided by Morrell in the JKSimMet training package (Morrell 1999). As with most design efforts, benchmarking of operating plants processing comparable material is invaluable to sanity-check any design inputs.

Ideally, the pebble crushing system will be designed based on an approach that considers the mill feed size distribution, ore hardness and competency parameters, mill grate and pebble port apertures, and mill operating conditions. Pilot mills have been traditionally used to indicate pebble production rates, although they are now less commonly a basis for SAG mill design. Population-based simulation models are the most capable of considering these factors, specifically ball load, mill filling levels, and grate and discharge screen aperture. A simulation approach that considers a range of measured ore characteristics and operating conditions can provide a meaningful estimate of typical levels and ranges of pebble production rates as the basis for crushing system design.



Courtesy of Metso Australia, Ltd.

Figure 3 Example JKSimMet SAG mill breakage rates

Ideally, the pebble crusher will be sized to treat an average or typical pebble production rate, with the screening and conveying system designed with an additional safety factor to allow peak loads to be accommodated. A pebble feed bin and feeder system is desirable to allow the pebble crusher to be choke fed at all times, and an overflow system should be considered to allow pebble production in excess of crusher capacity to be diverted to a static storage pile or directly back to the mill.

Callow and Meadows (2002) provided some practical guidance on pebble crushing system design, with key recommendations including the following:

- Allow for no less than 1 hour of live pebble storage capacity ahead of the crusher. Ideally, a pebble stockpile will be incorporated in the design.
- Favor multiple smaller crushers over a single large crusher for improved operating performance.
- Design for good grinding steel removal ahead of the pebble crushers. Consider combinations of trommel magnets, cross belt magnets, and metal detectors. A minimum of two stages of metal removal prior to a metal detector is recommended.

The pebble crushing system in the Telfer SABC circuit as described by Crawford et al. (2009) provides an example of good but capitally intensive design practices and the significant impact the ability to optimize a pebble crushing circuit can have on SABC circuit capacity. The Telfer circuit includes two-stage screening of the SAG mill discharge using a trommel (⅝-in. aperture) and pebble screen (½-in. aperture), a 3,000-t stockpile ahead of the pebble crushers, pebble crusher feed bins, and two pebble crushers for each grinding line. The consistent crusher feed rate afforded by the stockpile, and the ability to run single or multiple crushers as required were specifically cited by the authors as significant contributors to grinding circuit optimization.

The inclusion of a recycle crusher is detrimental to single-stage milling (Putland et al. 2011), except in certain circumstances. This is because the critical size material that holds up an open-circuit mill causing a fine transfer size is actually the media in the charge required to generate a finished product in single-stage SAG. Recycle crushing in single-stage mills is usually effective only for the treatment of very competent

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ores where the aim is to produce a coarse product size when the mill is an AG mill.

When used in a single-stage circuit, the pebble crusher is used for balancing load control with grind size. The crusher is brought online if the mill load is building up with the grind size getting too fine, and it is taken out of circuit when the load drops and achieving grind size becomes difficult with a coarser-than-desired value produced. Alternatively, the amount of pebble crushing can be proportionally adjusted with a percentage of the pebbles bypassing the crusher varied by the control system on the same basis. This configuration is best operated with optimization of the pebble crusher closed side set to maximize power draw. If pebbles are being crushed, they should be crushed fine for maximum value. In a single-stage mill with pebble crushing, smaller grate apertures are typically installed when compared to a SAG mill in SABC configuration. Typically, apertures <50 mm are used. This preserves a reasonable portion of coarse rock for use as media. Given the finer grate apertures, finer closed-side sets on pebble crushers and finer trommel or discharge screen apertures can be used.

Mill Speed

The critical speed concept applies to tumbling mills and is loosely described as the speed at which the mill will centrifuge and the charge of the mill will become pinned to the mill shell. At this point, the charge will no longer demonstrate the required tumbling action, and mill performance will significantly deteriorate. The critical speed (*Nc*) for a given mill diameter (*D*, ft) inside liners is determined by the following formula:

$$Nc,\text{ rpm} = \frac{76.6}{\sqrt{D}} \quad \textbf{(EQ 2)}$$

Or for a mill diameter (inside liners), in meters:

$$Nc,\text{ rpm} = \frac{42.3}{\sqrt{D}} \quad \textbf{(EQ 3)}$$

The concept of peripheral speed (*Np*) of the mill shell is an important factor in the design of rod and pebble mills, as discussed by Crocker (1985). The following equation from Rowland (2002) is used to calculate peripheral speed in units of feet per minute (fpm) or meters per minute (m/min), dependent on what units are used for the mill diameter (*D*, inside liners). *N* is the mill speed in revolutions per minute.

$$Np = \pi DN \quad \textbf{(EQ 4)}$$

High peripheral speeds in rod mills promote rod breakage and increased wear, and in pebble mills will promote faster pebble breakdown. Maximum peripheral speeds recommended by Crocker (1985) were 540 fpm for rod mills and 850 fpm for pebble mills.

AG/SAG mills typically operate in a wide range of critical speeds, dependent on mill operating conditions such as ball charge and total charge filling levels, feed particle size, and hardness. This speed range will generally be between 60% and 80% of critical. Lower speeds are used for softer ores and lower mill filling levels to avoid shell liner damage. Higher speeds are used for higher mill filling levels and more competent ores that require greater energy input to promote breakage. For all grate discharge mills, particularly SAG mills, the impact of mill speed on pulp discharge capacity must be understood. At high speeds, the mill discharge capacity may reduce as flowback or carryover increase. This is most commonly an issue in high-capacity open-circuit SAG mills or in closed-circuit SAG mills where the volumetric throughput of the mill is high.

The speed range of ball mills is much tighter, normally 72%–76% of critical, although higher speeds up to 80% of critical are used in small-diameter mills below 6 ft in diameter. Analysis performed at the Bougainville Copper Limited operation in Papua New Guinea (Plavina and Clark 1986) provided useful insight into the effect of mill speed on the performance of an 18-ft-diameter variable-speed ball mill. A series of trials concluded that variations between 70% and 80% of critical speed had no measurable impact on mill performance. Speeds exceeding 80% were observed to cause coarser grinds. Prior upgrades to increase the speed of the ball mills from 68% to 71% had been observed to increase plant throughput by approximately 3%.

Rod mill speeds are higher in smaller-diameter mills, with mills under 9 ft in diameter operating between 70% and 76% of critical speed; larger mills operate at slower speeds of 64%–70% to minimize liner wear and to prevent tangling of the rods. As noted by Rowland (1985), higher operating speeds increase the elevation of the rod charge and the spread between the rods, resulting in elevated rod wear and a coarsening of the mill discharge.

Pebble mill speeds are normally in the range of 70% to 75% of critical (Rowland 2002).

Ball Mill Volumetric Capacity

High slurry velocities and low slurry retention times in ball mills have been observed to be detrimental to ball mill grinding efficiency by studies performed by authors, including Rowland (1988) and Morrell (2001). Both investigations identified methods for determining critical slurry velocities through a ball mill, above which operational issues will be encountered that include coarse particle short-circuiting, excessive grinding media carryover, and reduced grinding efficiency.

Rowland (1988) provided Equations 5–8 to calculate the slurry residence time and a slurry velocity through the mill, and offered empirical guidance on targets for these values. A target mill residence time not less than 1.2–1.4 minutes and a slurry velocity not exceeding 6 m/min were recommended.

$$Vas = \frac{D^2}{4} \cdot \pi \cdot L(0.4Vp + Vdo - Vp) \quad \textbf{(EQ 5)}$$

where

- Vas = mill volume available for slurry
- D = mill diameter inside liners
- L = mill effective grinding length
- Vp = fraction of mill volume occupied by the ball charge
- Vdo = level of the discharge opening expressed as a fraction of the mill volume

$$RT = \frac{Vas}{Vs} \quad \textbf{(EQ 6)}$$

where

- RT = retention time, in minutes
- Vs = slurry volume per minute

Slurry velocity is then calculated with the following equations:

$$Aas = \frac{D^2}{4} \cdot \pi \cdot (0.4Vp + Vdo - Vp) \quad \textbf{(EQ 7)}$$

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Table 2 Basic arrangements of mill power transmissions

Motor Type	Speed	Pinion	Motor Speed, rpm	Primary Driver	Secondary Driver	Power Supply
Wound rotor	Fixed	Single or dual	900–1,200	Ring gear	Gearbox	Direct
	Variable	Single or dual	900–1,200	Ring gear	Gearbox	Slip energy recovery
Induction	Fixed	Single or dual	900–1,200	Ring gear	Gearbox	Direct
	Variable	Single or dual	900–1,200	Ring gear	Gearbox	Pulse width modulated
Synchronous	Fixed	Single	180–200	Ring gear	None	Direct
		Dual	180–200	Ring gear	None	Quadramatic
	Variable	Single or dual	180–200	Ring gear	None	Load commutated inverter, pulse width modulated, or cycloconverter
	Variable	None	9–15	Gearless	None	Cycloconverter

Source: Grandy et al. 2002

where *Aas* is the area available for slurry flow.

$$N = \frac{Vs}{Aas} \quad \text{(EQ 8)}$$

where N is the slurry velocity in distance per minute.

In a comparable approach, Morrell (2001) calculated a superficial velocity from the slurry flow rate through the cross-sectional area of the slurry pool that exists above the ball charge, extending from the toe of the charge to the discharge trunnion. A critical superficial velocity of 0.32 m/s was defined, above which a ball mill would be expected to demonstrate the operational issues associated with low residence time and high slurry velocities as discussed earlier. The basis for the calculation of Morrell's superficial velocity was not provided; however, his analysis confirmed the influence of pulp velocities on the performance of overflow ball mills.

MILL DRIVE SELECTION

A range of options are available for tumbling mill drives, and the selection is primarily influenced by the amount of mill power required, whether the mill will require fixed- or variable-speed operation, whether bidirectional mill rotation is required, the desired efficiency and mechanical availability of the drive system, and the capital costs associated with the individual options. The continuity of power supply should also be considered and more electrically robust systems preferred where frequent disruptions to power supply are possible. For stirred mills, the mill drive system is a standard component of the mill supply and therefore will only be subject to more subtle refinement during the mill selection process, in close consultation with the mill vendor. Table 2 summarizes the key features of common mill drive systems.

For large tumbling mills, the mill power requirement has a significant influence on the drive selection. Continual advances in the production of pinion drives have increased the capacity of single-pinion drives to 8.5 MW and of dual-pinion drives to 17.0 MW, as reported by Kalra et al. (2013) referencing the operating Wushan (China) copper and gold project ball mills. This change is a material increase over the 15.6-MW twin-pinion drives used on the Boddington (Australia) ball mills (Hart et al. 2011). The increased capacity keeps the dual-pinion drive technologies compatible with all but the very largest ball mills in use today. Coupled with variable-speed capabilities, these developments increase the applicability of gear-driven mills in AG/SAG mill applications. The relative dominance of gearless mill drives in SAG mill applications in recent decades is reflected in the review of Andean SAG mills by Rayo (2014), which reports 21 of 25 mills reviewed using gearless drives, with geared drives in use only in SAG mills with motor sizings of 8 MW or less in that analysis.

Variable-speed capability is an important requirement for AG and SAG mills to enable mill load levels and liner wear to be controlled in response to variations in mill feed size, fragmentation, pebble crusher performance, and ore characteristics. Fixed-speed SAG mill drives ideally should only be specified when ore characteristics are consistent through the deposit, although there are many instances where fixed-speed drives are selected on the basis of cost minimization. Variable-speed drives are rarely justified in the case of rod, ball, or pebble mills, as the same issues with variable mill load levels that can exist with SAG mills are not as significant an influence in environments where the grinding media filling all but completely dictates the volumetric composition of the mill.

Bidirectional capability is often a valued feature for SAG mills, as the ability to change the mill direction and use both faces of the shell lifter as the leading contact surface with the mill charge can allow up to 20% improvement in shell liner life. Continuous rotation in only one direction is, however, required in grate discharge mills that use curved pulp lifter designs. Furthermore, the use of a trommel screen to classify SAG mill discharge often restricts the mill operation to a single direction of rotation by virtue of the internal discharging spiral or scroll, which operates effectively only in one direction.

The optimum motor and drive configuration for a given project is a project-specific decision and must be based on thorough consideration of individual vendor submissions. Balanced consideration must be given to vendor performance records, capital and operating costs, system availability, and process performance requirements such as total power requirement and fixed- versus variable-speed capabilities. As noted by Grandy et al. (2002), this requires input from the process, electrical, and mechanical members of the project team.

INITIAL OPERATING CONDITIONS

Grinding Media Requirements

For rod and ball mills, the optimum ball mill recharge size can be estimated using the equations of Bond (1961). Both models predict the upper media diameter required for effective breakage.

For ball mills, makeup ball diameter (B), in inches, is given by the following equation:

$$B = \sqrt{\frac{F}{K}} \cdot \sqrt[3]{\frac{sg \cdot Wi}{100 \cdot Cs \cdot \sqrt{D}}} \quad \text{(EQ 9)}$$

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where
F = mill feed size, µm, 80% passing
K = 350 for wet grinding and 335 for dry grinding
sg = specific gravity of ore
Wi = work index, kW·h/st
Cs = mill speed fraction of critical
D = mill diameter inside liners, ft

This empirical ball size estimation model was based on the benchmarking of primary and secondary ball mills, though not ball mills following SAG mills. The model will often predict too fine a ball size when used to predict the optimum ball size for ball mills following SAG mills, as the distribution is typically bimodal, often with particles significantly coarser than the 80% passing size. Neither the SAG transfer size nor cyclone underflow particle size distributions reliably approximate the F value for a primary ball mill or a rod mill product. A good general starting ball makeup blend for ball mills following SAG mills is a mixture of 2-in.- and 3-in.-diameter balls, with the relative distribution adjusted to accommodate variations in SAG screen panel aperture and ore strength and grindability characteristics. In operation, the ball top size should be adjusted downward where possible to maximize ball charge surface area without causing excessive scatting.

For rod mills, makeup rod diameter (B), in inches, is given by the following equation:

$$B = \frac{F^{0.75}}{160} \cdot \sqrt{\frac{Wi \cdot sg}{100 \cdot Cs \cdot D}} \quad \textbf{(EQ 10)}$$

Bond (1961) further stated that the rod diameter should be increased by ½ in. when the reduction ratio (F_{80}/P_{80}) is less than 8.

For pebble mills, Crocker (1985) prescribed that grinding media of equal weight to the ball size calculated in Equation 9 is adequate under appropriate grinding conditions. Pebble media must be prepared to allow for this equivalent weight to be achieved after the "rounding up" process, where the particles may lose 15%–35% of their mass while the pebble surface wears to a smooth condition.

Bond (1961) also provided an equation to describe the equilibrium ball size distribution that exists in the mill based on the recharge size (B) determined in Equation 9. Bond speculated that this formula should also hold for rod and pebble mills. The percentage (y) of the total equilibrium charge that passes size (x, in.) is given by the following equation:

$$y = \left(\frac{x}{B}\right)^{3.8} \quad \textbf{(EQ 11)}$$

Bond (1958) further provided nominal start-up grinding mill charges for rod and ball mills intending to best reflect the equilibrium ball charge distribution as a function of the optimal recharge size determined by Equations 9 and 10. Bond's start-up, first fill, or graded charge distributions, in terms of percentage weight distribution, are reproduced here for ball mills in Table 3 and rod mills in Table 4.

Another common grinding media size optimization approach uses the following formula developed by Azzaroni (1980) and revised by MolyCop to derive the optimum makeup ball size:

$$dB = \frac{6.06F80^{0.263}\ (ps\ Wi)^{0.4}}{(ND)^{0.25}} \quad \textbf{(EQ 12)}$$

where
dB = makeup ball size, mm
F_{80} = 80% passing fresh feed size, µm
ps = ore specific gravity, t/m^3
Wi = Bond ball mill work index, kW·h/t
N = mill speed, rpm
D = mill diameter inside liners, ft

The charge-specific surface or string area (m^2/m^3) produced by this optimal ball size is then replicated as closely as possible

Table 3 First fill charge distribution for ball mills

Makeup Size (*B*), in.	4½ in.	4 in.	3½ in.	3 in.	2½ in.	2 in.	1½ in.
4½	23.0						
4	31.0	23.0					
3½	18.0	34.0	24.0				
3	15.0	21.0	38.0	31.0			
2½	7.0	12.0	20.5	39.0	34.0		
2	3.8	6.5	11.5	19.0	43.0	40.0	
1½	1.7	2.5	4.5	8.0	17.0	45.0	51.0
1	0.5	1.0	1.5	3.0	6.0	15.0	49.0

Source: Bond 1958

Table 4 First fill charge distribution for rod mills

Makeup Size (*B*), in.	5 in.	4½ in.	4 in.	3½ in.	3 in.	2½ in.
5	18					
4½	22	20				
4	19	23	20			
3½	14	20	27	20		
3	11	15	21	33	31	
2½	7	10	15	21	39	34
2	9	12	17	26	30	66

Source: Bond 1958

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by using one or more conventional ball sizes in the makeup charge. The same approach can be used to estimate the graded ball size distribution in the mill, which can be replicated in the initial start-up charge.

The empirical method developed by Azzaroni (1980) suffers the same shortcomings as the Bond model when applied to balls mills following SAG mills because of the nature of the benchmarking database. In these SAB/C applications the model predictions should be validated against operating practices for comparable grinding duties. For SAG mills there is no such desktop method to reliably estimate the optimum ball size or top ball size for a given condition, although a suitably validated population balanced based simulation program can provide practical guidance for SAG ball size optimization. In many instances, the selection of the optimum ball size is more arbitrarily based on user experience. That said, other than for very fine feeds, the use of 5-in. balls for SAG mills is common, and the range of ball sizes applied in SAG is narrow, typically between 4½ and 6 in. The use of larger-diameter SAG balls, greater than 5-in. diameter, requires more careful consideration of mill operating conditions and liner design to avoid excessive liner breakage as a result of the nonlinear increase in ball weight and impact energy.

For fine and ultrafine grinding applications, the optimum media size is determined during laboratory- and pilot-scale testing programs, and is specific to the application and target grind size.

The optimum design media filling level can be readily estimated from a good power draw model after the determination of specific grinding energy requirements (kW·h/t) and the design mill capacity. A good example is the Morrell power model described in Chapter 3.12, "Testing and Calculations for Comminution Machines," and available online at www.smctesting.com. For overflow ball mills with high volumetric flow rates and high ball filling levels, the influence of slurry velocity on ball retention must be considered, and in some instances, the use of a ball retaining ring may be required to allow the design ball charge to be maintained. In SAG mills, the effective removal of slurry by the pulp discharge system is required to ensure that the design power draw is achieved at the modeled charge level. In the extreme instance where slurry pooling exists, it is likely that a slightly higher ball charge will be required to deliver the target power draw.

For open-circuit SAG mills, the optimal, nominal, and maximum ball charge levels selected are dependent on the scale of the project, feed size, competency, and design contingency. These parameters become even more important for single-stage mills with the required ball charge and circuit success dictated by the ore characteristics, feed size, and circuit product size (Putland et al. 2011). It is prudent to specify a maximum ball charge level above that ever intended for operation to safeguard the mechanical integrity of the mill. In cases of extreme ore abrasiveness, the mill diameter may be deliberately increased to allow the operating ball load to be reduced or even eliminated to manage the ongoing cost of high grinding media consumption rates.

Consideration should be given at the design stage to the influence of grinding media specifications on metal recovery and final product performance. Several examples of improved flotation response and concentrate quality attributed to the use of high-chrome grinding media have been observed for precious and base metals ores. A recent example was provided by Greet et al. (2012) of the Newcrest Ridgeway Cu/Au concentrator in Australia, where converting the ball mill charge to high-chrome grinding media achieved a 1% increase in copper recovery and a 2%–4% increase in gold recovery by flotation. Reversing the trial and converting the mill back to carbon steel grinding balls resulted in a 1% loss in copper recovery, with an inconclusive influence on gold recovery. The use of other inert grinding media, such as silica sands or ceramic media, in fine and ultrafine grinding applications (Greet 2008; Rule 2011) delivers similar metallurgical process performance and product quality benefits. Where it is essential that final products have minimal iron contamination, the use of pebble milling and primary AG often becomes a preferred technology selection.

Mill Liners

Mill liners are often stated to have two major purposes: (1) to provide an effective wear lining for the body of the mill and (2) to effectively impart energy to the grinding charge to induce effective size reduction. These objectives are noted by Powell et al. (2006) to conflict with each other but also be "optimized simultaneously with suitable liner selection." It is important that the shell liner design achieves the desired charge motion, normally represented by two-dimensional trajectory profiles made popular by discrete element modeling (DEM) techniques, to promote effective size reduction in the mill. The larger the spacing between lifters, the higher the lifter can be above the shell plate and the larger the required relief angle on the lifter. The larger the lifter, the longer the lifter wear life; however, large spacing exposes the shell plates or section to wear. An optimal design must balance production, lifter, and liner wear. These goals often have opposing requirements, and a balance must be found. Feed head liners are designed with the primary objective of prolonging wear life, as is the case for discharge head liners. In the case of grate discharge mills, the discharge head design must also consider the effective removal of material from the mill via grates, pebble ports, pulp lifters, and the discharge cone.

There are many examples of significant issues with the original mill liner design after start-up, specifically packing and liner breakage. Consequently, careful consideration of the mill liner system design prior to mill start-up is well justified. While it is an impractical objective to fully optimize a mill liner profile at the design stage, several factors should be considered to ensure acceptable mill performance on start-up. These include liner material selection and liner profiles, including the appropriate height and spacing of lifters. These are the key considerations to get right with mill liner designs, beyond which further refinement of the liner design is unlikely to deliver significant improvements in grinding efficiency. The objective is to ensure that liner materials are suitably robust for the duty, liner spacing is adequate to avoid packing, and lifter profile is appropriate to mobilize the mill charge but not to overthrow and cause excessive liner impacts.

Liner Materials

A comprehensive review of industry practices by Powell et al. (2006) noted the use of high-carbon chrome moly steel as the most common material used for SAG mill liners, high-chrome irons in rod and ball mills, and rubber liners traditionally used in secondary and tertiary grinding applications. Composite liners, using a rubber base with steel inserts at the wear surface, have been successfully used in many primary

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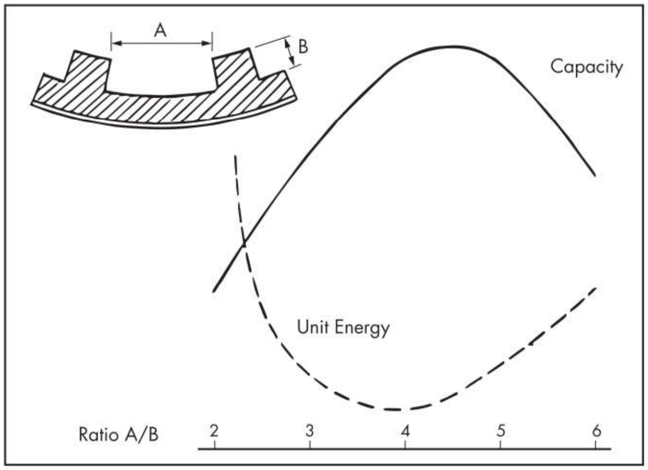

Source: McIvor 1981

Figure 4 Effect of liner spacing-to-height ratio on mill capacity

and secondary milling applications, including large-diameter AG mills (38 ft) and SAG mills (32–34 ft).

Claims of significantly improved grinding efficiency have been made when steel linings are used in preference to rubber in ball mills, although this has not been conclusively demonstrated in published studies. One recent case study (Maclean et al. 2014) of the conversion of the 20-ft Batu Hijau (Indonesia) ball mills from steel to composite linings demonstrated no material difference in grinding efficiency as a function of mill lining in an extended head-to-head trial of composite and steel shell lining systems.

Lifter Spacing

Extensive pilot-scale AG mill test work by Meaders and MacPherson (1964) established the original benchmark lifter spacing ratio, represented as spacing over height, or A:B ratio, as nominally 4:1. This is represented by Figure 4 from McIvor (1981) based on the original pilot work.

The 4:1 spacing ratio was confirmed as an optimum during field trials at Highland Valley Copper (British Columbia, Canada) in the early 1990s reported by Bigg and Raabe (1996) and is implicit in the formula from optimal A:B ranges for rubber-lined mills reported by Moller and Brough (1989), which also considers mill speed. This formula is given in the following equation, where *FCs* is the fraction of critical speed.

$$B = (1 - FCs) \times A \quad \textbf{(EQ 13)}$$

where A is the spacing between lifters, and B is the lifter height above the liner plate.

AG/SAG Mill Liners

Notwithstanding the supporting information for a spacing-to-height ratio of 4:1 provided by these references, apparently in practice, particularly for large-diameter SAG mills, it is more commonplace to design initially for a lower A:B ratio and accept that the ratio cannot be optimum for the duration of liner life. This is consistent with the recommendations of Powell et al. (2006) to start with a low ratio and finish with a higher value at the time of liner replacement, and consider an ideal variation of A:B ratio of ±1 across the liner life. This is evident in shell liner designs reported for operations such as Yanacocha in Peru (Burger et al. 2011) and Peñasquito in Mexico (Palmer et al. 2011), shown in Figure 5. These designs clearly show an initial A:B ratio of less than 4:1, even when high-low profiles are used to reduce the average height above the liner plate.

DEM is a valuable tool to support initial liner design, particularly to optimize lifter height, face angle, and spacing. This process is particularly important for SAG mills where the potential for shell liner damage because of large-diameter balls overthrowing the toe of the mill charge is more apparent. While increased lifter height is well known to promote longer wear life, when combined with a steep face angle it can result in increased frequency of charge overthrow and liner breakage. The documented experiences at the Freeport Grasberg concentrator in Papua, Indonesia (Coleman and Veloo 1996; Staples et al. 2001) are instructive in this regard, particularly with the use of DEM to optimize the shell liner design to improve the grinding charge trajectory. Ideally, DEM and other instructive simulation techniques, such as smoothed particle hydrodynamics (SPH), and high-fidelity and computational fluid dynamics, should be applied as often as possible in support of good initial liner design in the first instance. They should not be reserved for use only as a tool to correct suboptimal performance after mill commissioning.

Some useful trends in mill liner design for SAG mills are apparent based on recent industry practices and publications presenting optimized liner designs and performance improvements. Quoted improvements are associated with mill throughput, power draw, elimination of packing, reductions in liner breakage, and overall increases in liner service life. The documented experiences at the following operations demonstrate the approaches available for SAG mill liner optimization:

- Grasberg (Coleman and Veloo 1996; Staples et al. 2001)
- Cadia in Australia (Hart et al. 2006)
- Alumbrera in Argentina (Sherman 2001)
- Peñasquito (Palmer et al. 2011)
- Los Pelambres in Chile (Villanueva et al. 2001)
- Collahuasi in Chile (Villouta 2001)
- Candelaria in Chile (Miranda et al. 1996; Kendrick and Marsden 2001)
- Yanacocha (Burger et al. 2011)

As observed by Royston (2007), SAG mill lifter face angles have progressed from steep angles of 5°–15° to a more common range of 22°–35°, large-diameter mills (>24 ft) are tending to favor high-high liner designs over high-low designs, and the number of rows of lifters has on average decreased. Lifter face angles and profile style are relatively easy to change in comparison to lifter spacing, which is dictated by the mill shell drill pattern. Historically, SAG mill linings followed the rule of thumb that the shell should have twice the number of lifter rows as the mill diameter, in feet, often referred to as a 2 × D profile, or comparable variations on that theme. Examples of the 2 × D design are the 72 row lifter sets used in the 36-ft-diameter SAG mills at Batu Hijau (Burger et al. 2006) and Candalaria (Miranda et al. 1996). The Batu Hijau operation is a unique example that reported a different experience than other authors regarding the effects of SAG mill lifter spacing. Trials of reduced lifter rows from the original 72-row design, including 48-row and 36-row designs, were unsuccessful and the operation reverted to the original lifter spacing. In general, however, greater awareness of the occurrence and detrimental impacts of packing on mill performance, compared

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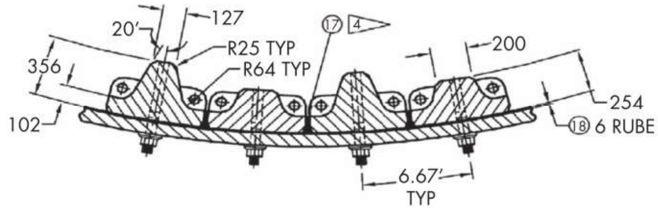
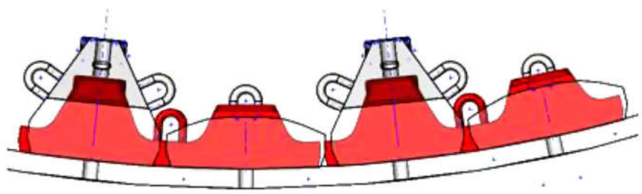


Minera Yanacocha Original Shell Lifter Profile
Source: Burger et al. 2011



Alternative Minera Peñasquito SAG Mill Shell Lifter Profile
Source: Palmer et al. 2011

Figure 5 Recent SAG mill liner profiles

with observations of improved mill performance with wider lifter spacing in practice, have led to a reduction in the number of lifter rows for SAG mills with designs in the range of 4/3D to 5/3D made popular by the experiences of operations such as Cadia (Hart et al. 2006), Alumbrera (Sherman 2001), and Collahuasi (Villouta 2001). This reduction has also been facilitated by the increased use of variable-speed drives on SAG mills. Historically with fixed-speed mills, the presence of packing with a new design was required to prevent ball overthrow and protection of the liners while still maintaining a profile that would maintain reasonable lift at the end of the mill's life. Now with variable-speed drives, the ball trajectory can be optimized throughout the life cycle of the liner. This does, however, impact available SAG mill power draw over the life of the liners. Liner profiles reflecting 1 × D spacing have also been implemented at Yanacocha (Burger et al. 2011) and Los Pelambres (Villanueva et al. 2001). Clearly, the initial drill pattern of the mill shell must consider the optimum lifter spacing, in addition to contingency patterns, should the desired performance not be achieved after commissioning. This has led to the use of approximately 2 × D designs, divisible by 3, so that approximately a 1.33 × D design can be implemented with the same shell drill pattern.

Similar considerations hold for the design of the mill discharge liner system when it comes to the application of advanced modeling techniques in support of discharge liner design, and the planning of the drill pattern for the discharge head to accommodate optimization of the pulp lifter design. A good example of the latter point is provided by Barratt and Sherman (2002) regarding the Inco Clarabelle SAG mill in Canada, which was drilled to accommodate both radial and curved pulp lifter designs. This may prove an important consideration if there is a need to significantly improve mill performance by moving from radial to curved and/or twin-chamber pulp lifter design, as has been achieved at Cadia (Hart et al. 2006) and at Cortez in Colorado, United States (Stieger et al. 2007). Alternatively, the operation may realize up to a 20% increase in shell liner life as observed by Royston (2007) by employing radial pulp lifters and bidirectional rotation of the mill shell. For very high-capacity open-circuit SAG mills, or closed-circuit SAG mills such as in single-stage milling modes, there have been several instances where decreased mill performance was caused by slurry pooling from underdesign of the mill discharge system. Subsequent modeling using techniques such as DEM and SPH has illustrated issues with slurry flow back into the grinding chamber, flow restrictions at the discharge cone, inadequate pulp discharge chamber capacity, and pulp carryover in the discharge vanes. In all cases, such issues can be readily identified at the design stage through detailed simulation and the knowledge applied to optimize the pulp lifter design. Thereby any potential production constraints because of an undersized pulp discharge system can be readily addressed during mill design.

Grate Design

The discharge grate in AG/SAG mills has a significant impact on mill performance, being critical to retaining effective grinding media in the mill charge, while maintaining a working mill load and effectively allowing ground product and critical size material to discharge the mill. The initial grate design can be supported by advanced simulation but will typically be derived from benchmarking mills of comparable size and duty.

Grate apertures will be slots or ports ranging from ⅝ in. to 4 in. Square, slotted, or elliptical designs are common. The amount of the grate surface area that is open for material flow can range from 2% to 14% and is influenced by grate aperture size, ore particle size and competence, and the nature of the pebble management system. SABC-B circuits will tend toward larger apertures and more open area to maximize material flow to the pebble crushers. However, the pebble port size influences the operable closed-side set of the pebble crusher and therefore SAG mill discharge screen aperture. The Los Pelambres operation (Powell and Valery 2006) is an example

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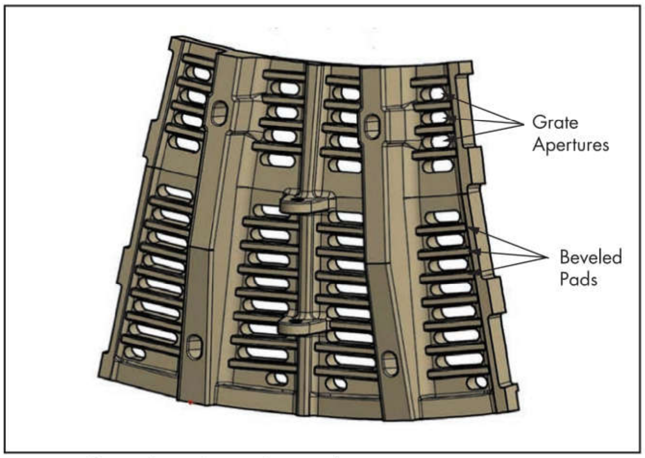

Courtesy of Growth Steel Australia Pty Ltd.
Figure 6 SAG mill grate design

of SAG mills operating with high open area (10%–14%) to maximize pebble discharge rates. The use of very high grate open area can facilitate slurry flowback from the pulp lifter to the grinding chamber of the mill, and the impact of pulp discharge system efficiency should be considered in any grate open-area optimization exercise.

Common issues with discharge grates that need to be managed in operation are peening and plugging of the grates. Peening refers to the closing up of the grate opening because of metal flow from contact with large-diameter grinding steel. Plugging is the blocking of the grate aperture with worn grinding steel, or in some instances with pebbles. Plugging is managed by relieving or tapering the grate apertures so that the aperture is larger on the discharge side of the grate, more readily allowing material to be released through the grate. Rowland described 3.5°–5° relief on either side of the slot as nominal. The use of rubber or composite grates is a common solution to plugging issues with smaller grate apertures because of the greater flex of the rubber while still maintaining strength. Peening can be managed by using protective lifters between grates and using beveled pads surrounding the grate aperture (Figure 6). However, care must be taken not to use too large a protective lifter, as this can result in reduced extraction rates. The beveled pads act as a point to absorb metal impacts, confining the metal flow to the pad rather than on the edge of the slots.

The discharge grate design usually incorporates a lifter as either an integral or a replaceable section. The lifter helps to slow down the wear of the grate face plate section but also has a bearing on the amount of effective grate open area, creating a shadow effect as material flows over the lifter and down onto the grate face as the mill rotates. The grate section thickness and the lifter height must be optimized in unison to ensure both suitable wear life of the grate and effective use of the available grate open area.

Rod Mill Liners

Rowland (2002) reported the popularity of single-wave alloy steel or cast-iron shell liners in rod mills. Figure 7 illustrates a modern rod mill single-wave liner design. Nominal design features include a total number of shell lifters equivalent to double the mill diameter in feet, wave thickness of 2.5–3.5 in. on top of 2.5–3.0-in. liners, and rubber backing to protect the mill shell and heads from washing and corrosion. End liners should be smooth and free of features that may disrupt the rod action and cause tangling. Rubber or wear-resistant cast-iron end liners are prone to damage and are not recommended. Rubber shell liners have been successfully applied in the smaller-diameter rod mills running at slow speeds and reduce the noise generated by the mill. When using rubber liners, close attention must be paid to rod quality and removing broken or thin rods from the charge.

Ball Mill Liners

Wave liners are the most common steel liner design for ball mills, with either single-wave or double-wave designs being popular. Rowland (2002) noted the recommended spacing of 13.1/3.3 × the mill diameter (*D*) for double-wave liners and 2 × D + 2 for single-wave liners when the mill diameter is expressed in feet. Wave liners for ball mills are usually made of alloyed steel or wear- and impact-resistant cast irons. End liners for ball mills conform to the slope of the mill head and can be made of rubber, alloyed cast steel, or wear-resistant cast iron. To prevent racing and excessive wear, end liners for ball mills are furnished with integral radial ribs and/or replaceable lifters. A variety of alloys are suited dependent on the mill duty, with rubber and composite liner designs also suited in many applications including modern, large-diameter ball mills. The use of rubber liners in dry milling applications is not recommended. While rubber liners have been associated with a loss in mill volume and power draw in some primary milling applications, they are well suited for use in ball mills with small-diameter balls. Composite liners have been a viable alternative to steel liners for even the largest-diameter ball mills. Rubber and composite liners are typically designed with separate liner and lifter segments (Figure 8).

Stanley (1987) and Powell et al. (2006) describe the popularity of pocketed grid liners in South African mills, with Stanley observing 26% of South African gold mills using these lining systems at the time of his publication. These liners trap grinding media to act as a self-regenerating sacrificial liner, resulting in a low liner profile and requiring mill operation at high speeds, approximately 90% of critical speed, for effective grinding action. These types of liners are only used in smaller-diameter low-aspect mills. Norrgran (2009) and Ellsworth et al. (2007) describe the application of permanent ceramic magnetic liners in the U.S. iron ore industry, with Norrgran quoting the installation of more than 400 magnetic liner sets worldwide. Magnetic liners are limited by the structural integrity of the liner to application in 10-ft-diameter mills for rubber-encased magnets or 18-ft-diameter mills for steel-hosed magnets. Such systems can offer the advantage of faster and safer installation over conventional steel lining systems, as well as significantly higher liner service life.

Pebble Mill Liners

Crocker (1985) observed that pebble-mill liner design is similar to ball-mill liner design with the primary difference that lifting the lighter specific gravity autogenous media must be accommodated to ensure functional charge motion. Centrifugal force does not hold the light pebble against the shell to the same extent that steel balls are held. Most pebbles are very abrasive, and excessive slip must be prevented to provide reasonable liner life. As in ball mills, the size of media, percent ball load, speed of the mill, and media size are all considered when selecting the proper profile of the shell liner.

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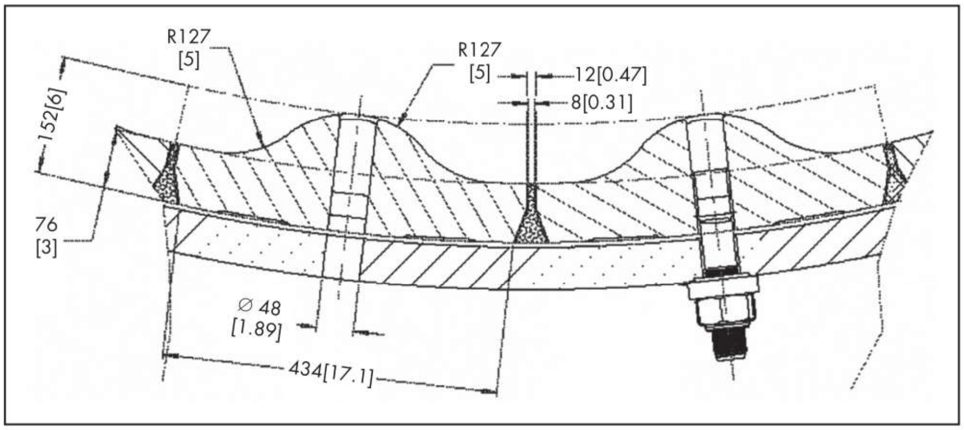
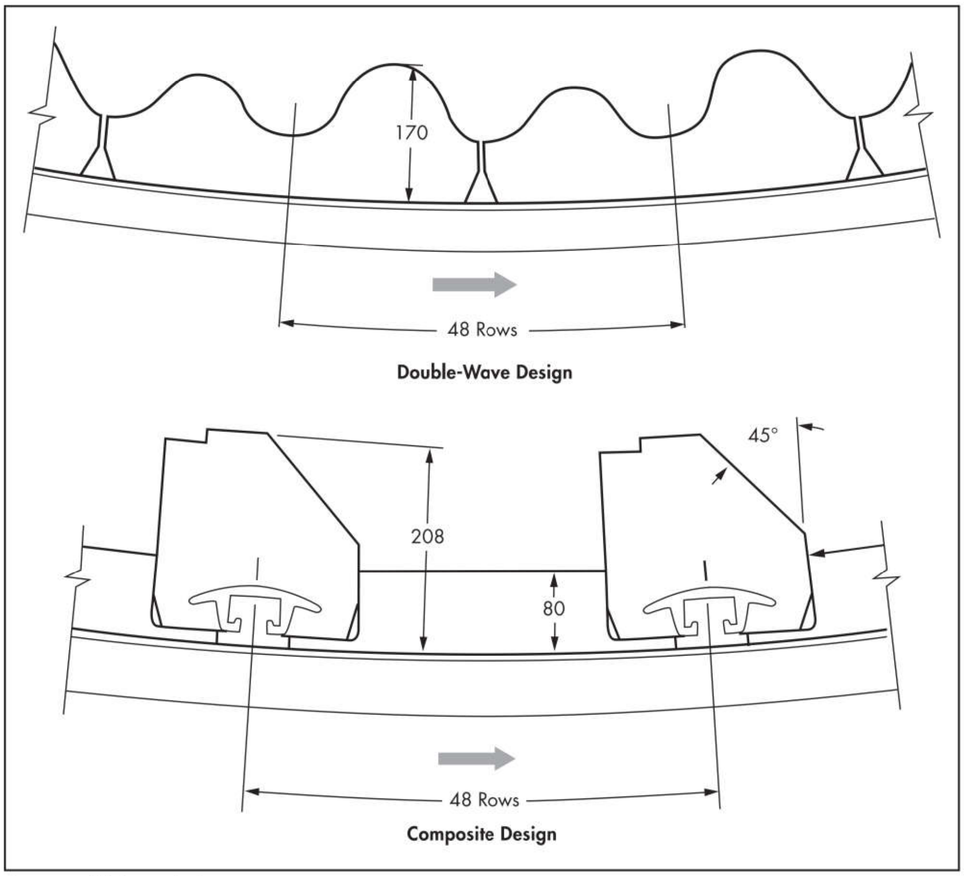


Courtesy of Growth Steel Australia Pty Ltd.

Figure 7 Rod mill single-wave lifter profile



Courtesy of Polycorp Ltd.

Figure 8 Ball mill liner profiles

As pebble mills are not too violent in action, smaller bolts can be used to hold the liners in place. Shell liners may be made of white iron, Ni-Hard, rubber, or chrome moly. The profile of the liner designed to suit the size of media and speed of the mill is the important factor in a successful operation. Feedhead liners are designed with radial and staggered radial lifters. Again, the profile must be higher than would be used with steel balls, usually the same lift as on the shell liners.

CLASSIFICATION

Appropriate classification design is essential for the successful design of a comminution circuit, as poor classification can be a major contributor to grinding circuit inefficiency. Many equipment options exist, including hydrocyclones, screw classifiers, vibrating screens, and sieve bends. However, in most circuits, hydrocyclones are used because of fine product sizes (–250 µm), low capital costs, operability, and their suitability to a wide range of circuit capacities.

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Table 5 Hydrocyclone capacity guide

Cyclone Diameter	Capacity, Mt/yr
10 in. or 250 mm	–1.5
15 in. or 400 mm	1.0–5.0
20 in. or 500 mm	2.0–8.0
26 in. or 650 mm	5.0–15.0
33 in. or 800 mm	+10

Screw classifiers were historically used in circuits targeting a coarse cut size but are only applicable to low-throughput circuits and are rarely used in modern designs. Sieve bends are used in some circuits, typically when a coarser cut size is required. A common use of sieve bends is in bauxite grinding where a coarse grind is required and the classifier encounters hot caustic process solutions.

Circuits that may have previously used sieve bends are now often installed with high-frequency vibrating screens. These screens classify more effectively at lower cut sizes with apertures down to 75 µm commonly used. Screens are commonly applied in circuits of low to moderate throughput when the overgrinding of valuable high specific gravity components of the ore is detrimental to downstream processing. Examples include small-tonnage lead–zinc grinding circuits, rare earth, tin, and tantalum mineral comminution.

Recent examples of the successful application of fine screens in small-capacity grinding circuits in duties normally reserved for cyclones are provided by Valine et al. (2012). In these applications, high classification efficiency and low circulating loads are consistently achieved. The authors cite examples where grinding circuit capacity has been increased because of the increased classification efficiency provided by high-efficiency fine screens. Because of the engineering and capital requirements associated with fine-screening technologies, this approach traditionally has been limited to smaller (<300 t/h) grinding circuits, as demonstrated by Valine et al. (2012).

A recent example of the application of screens in preference for hydrocyclones in a coarse-grinding, high-throughput application is the Swakop Uranium Husab mine in Namibia. The operation began commissioning activities in late 2016 and is a 15-Mt/yr nameplate operation treating uranium-bearing Alaskite granite. The comminution flow sheet consists of a primary gyratory crusher, a coarse ore stockpile, and an SABC-A grinding circuit using banana screens for ball mill classification. The design throughput rate is 1,875 t/h and the target P_{80} is 355 µm ahead of leaching with sulfuric acid. Milling is in mild acid conditions, as there is no separation of process waters between comminution and leach. The SAG mill has 10.8-m-diameter inside liners, an EGL of 5.29 m (36 ft × 18 ft shell), and is fitted with twin 6.75-MW variable-speed, geared pinion drives. The SAG mill discharges through a trommel screen with the oversize directed to pebble crushing and crushed pebbles returning to the SAG mill. Trommel undersize is pumped to a distributor box and then to the four (3.66 m × 8.23 m) banana screens, fitted with 0.8-m polyurethane or rubber panels. Screen oversize feeds the ball mill, and the undersize is sent to the leach feed thickener. The ball mill has 6.51-m-diameter inside liners and an EGL of 10.21 m (22 ft × 33.5 ft shell) and is fitted with twin 4.25-MW variable-speed, geared pinion drives. Variable speed was chosen for the ball mill to minimize starting power requirements and is proving a useful control tool when screen classification is employed.

For most grinding circuits, hydrocyclones are used because of their simplicity and suitability for a range of applications. Typically, the size of the hydrocyclone is selected so that it can achieve the target cut size and final grind size at the desired cyclone overflow density. Cyclone diameter and capacity should be selected targeting a total number of operating cyclones between 4 and 12 to ensure process stability under a range of operating conditions. On this basis, there tends to be typical capacity ranges for cyclone diameters. These can obviously overlap if less than 4 cyclones or high overflow densities are targeted (very low circulating loads) or more than 12 cyclones and low cyclone overflow densities (high circulating loads). However, in a broad context, Table 5 indicates a capacity range for a cyclone diameter with lower envelope typical for finer grind sizes and the upper, coarser grind sizes (P_{80} ranging from 200 to 70 mesh).

The importance of the selection of the diameter and number of cyclones operated should not be underestimated, as this defines the slurry density exiting the circuit as cyclone overflow and also the circulating load for a given cut size. Operating a circuit within an optimal circulating load range is important in achieving efficient operation and optimal power draw from the grinding mills. If a circulating load is too low, overgrinding of fines will occur, flattening the shape of the product distribution. This results in higher power consumption for a target P_{80}. If the circulating load is too high, slurry pooling can occur, reducing power draw. This predominantly affects SAG mills and grate discharge ball mills but also affects overflow ball mills as well. Too high a flow through an overflow ball will result in ball ejection, coarse particle short-circuiting, and reduced power draw. Very low residence times and high slurry velocity negatively impact grinding efficiency.

The significance of classification efficiency will vary from project to project based on the nature of valuable minerals and the downstream extraction or separation processes. For example, the extraction of gold by cyanidation in many instances benefits from the preferential recirculation and overgrinding of high-density host sulfides minerals, increasing the liberation and exposure of fine gold inclusions. Gold extraction also does not suffer the reduction in efficiency at fine sizes typically observed for sulfide flotation. By contrast, overgrinding of high specific gravity sulfide minerals ahead of flotation is detrimental to flotation recoveries and more likely to be a factor in the selection of the classification process.

The optimal circulating load for a grinding circuit is not easily defined, although the circuit design and operating conditions should avoid exceeding the volumetric capacity of the tumbling mill. The penalties include operational performance issues associated with high slurry filling levels in grate discharge mills, or high superficial velocities and low residence times in ball mills. The effective management of these conditions during the design process was discussed earlier in this handbook. As a rule of thumb, a coarser grind or a low specific energy consumption grinding duty will be associated with a low circulating load in practice. Conversely, a fine grind or higher specific energy grinding duty will tend to operate at high circulating loads. Coarse grind size or low specific energy grinding duties can often operate effectively at circulating loads as low as 150%, while fine grinding or high specific energy applications can often operate at circulating loads greater than 600%.

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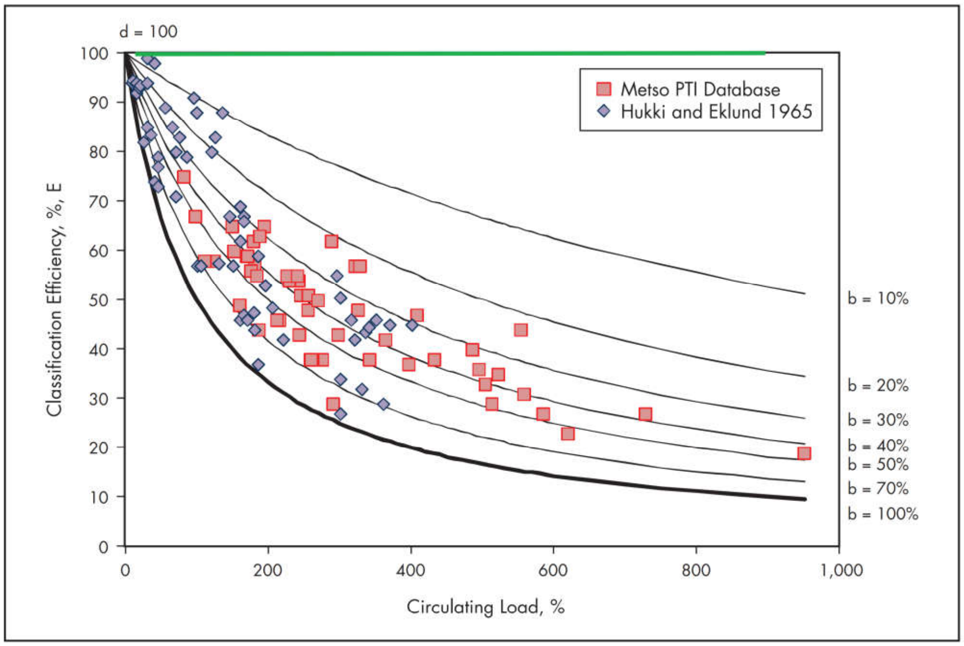

Source: Jankovic and Valery 2012a

Figure 9 Relationship between classification efficiency and recirculating load

Cyclone models are available in the literature and can be used in combination with cyclone vendor capacity charts to establish initial hydrocyclone performance and design criteria. Models such as those developed by Plitt and Nagaswararao (Napier-Munn et al. 1996) are particularly useful for this purpose. Most hydrocyclone vendors have in-house simulation software that can and should be used to support final cyclone selection.

The interdependence of recirculating load and classification efficiency has been well demonstrated by Jankovic and Valery (2012a), as shown in Figure 9. These authors demonstrated that increased recirculating loads are associated with more moderate increasing in grinding circuit capacity than has been proposed by other studies because of the reduced classification efficiency of product size material that is often associated with higher recirculating loads. Figure 9 illustrates the moderate gains in grinding circuit efficiency with increased recirculating load that could be expected from a hydrocyclone system operating with typical efficiency (40% product material recovery to coarse product). As Figure 10 demonstrates, when classification efficiency is higher (15%–20% product size reporting to the coarse product with screening options), the impact of higher recirculating loads on circuit efficiency is much greater.

Jankovic and Valery (2012b) state that up to 15%–25% of the total grinding circuit energy consumption can be attributed to inefficient classification in conventional grinding circuits, highlighting the potential value that can be realized by a well-designed and operated classification system.

SAMPLING

The often-quoted adage that you cannot control what you do not measure is certainly true for grinding circuits, where control of a process that consumes most of the operation's power consumption is a significant ongoing concern. Grinding circuits, particularly tumbling mills, do not require a significant amount of manual sampling for process control and metallurgical accounting purposes, yet there are some streams that need to be accessible for safe and reliable sample collection. The optimization of any grinding circuit requires the periodic sampling of all major process streams, and the coarse particle size distributions and high slurry flow rates encountered around primary and secondary grinding circuits demand close adherence to the principles of cross-stream sampling. These requirements are often poorly addressed in grinding circuit design, leading to invariable compromises in sampling practices and consequent limitation of the plant operator's ability to fully optimize the performance of the circuit.

It is necessary to have some access to the mill discharge stream for wet grinding mills operating in closed circuit, so that the mill operating density can be manually measured and controlled. Access to the final grinding circuit product, often cyclone overflow, is essential for confirming pulp density and achieving grind size control. Where the grinding circuit product is required for metallurgical accounting purposes, the effective sampling of this stream will be carefully considered. In instances where such a sample is not deemed necessary for metallurgical accounting, accessibility for sampling can be considerably diminished. In all grinding circuits, the ability to manually sample the grinding circuit product for performance monitoring is an essential requirement, irrespective of whether automated sampling systems are in place for metallurgical accounting purposes.

For grinding circuit optimization surveys, accessibility to a larger number of sampling points will be necessary. These will generally include the feed and product streams from any individual milling, screening, or classification stage. Data will be mass balanced by size fraction requiring large samples to be

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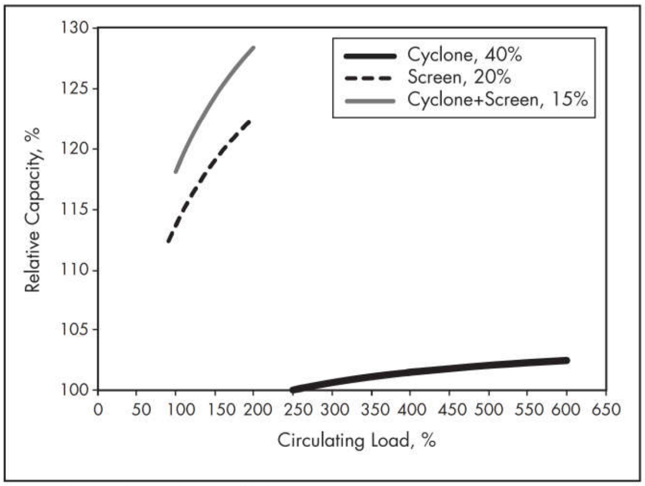

Source: Jankovic and Valery 2012a

Figure 10 Relative circuit capacity as a function of classification mechanism, fines recovery, and recirculating load

collected for statistically representative information on coarse streams. Any conveyors to be sampled require sufficient belt length to be accessible for sampling to allow the required sample mass to be collected. Slurry streams must be accessible for cross-stream sampling of the entire flowing stream. Further fundamental requirements are that any sample point must be accessible in a manner that conforms to acceptable safe work practices, and that sample handling and removal can be readily managed with minimal potential for manual handling issues. In some instances, such as wet screen undersize, a perfect sampling solution can be difficult to achieve in the design stage; however, a minimum requirement should always be to provide some practical means of access to the slurry stream for sampling.

More specific guidance on sampling requirements for grinding circuits is included in Chapter 3.10, "Grinding Circuit Performance Optimization," and these requirements should be reviewed during the design stage of the project.

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