The Science of Comminution
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Chapter 1: Outokumpu Mills group Overview

Outokumpu (former Nordberg Mills) has, over many years, established itself as a leader in the development of innovative, efficient comminution equipment and related mineral processing technology.

Outokumpu’s mill group was created by combining the grinding mill business formerly known independently as Nordberg, Morgardshammar and Scanmec with the Outokumpu mill business. Outokumpu (former Nordberg Mills) was an early leader in the design and production of large rod and ball mills for the mining industry and has supplied some of the world’s largest unit machines.

Outokumpu rod mill commissioned in 1970, grinding coal for firing boilers at an installation in the state of Arizona, USA.

The engineers of the Morgardshammar Mining division have a long, proud history of service to the Scandinavian mining industry and are well-known for innovative design solutions. In addition to a broad range of rod, ball and pebble mills, Morgardshammar is also recognized as an international leader in the design and supply fully and semi-autogenous mills, particularly those with low aspect ratio sometimes known as “Scandinavian” or “South African” style mills.

Morgardshammar optimized the design of grinding mills using antifriction type main bearings for high reliability and load carrying capacity. Another Morgardshammar innovation was the adoption of nodular cast iron for use in producing end plates and girth gears. Nodular iron offers high strength and toughness, and its use offers the benefit of more reliable foundry techniques.

The Norwegian firm Scanmec also was known for introducing important innovations in mill design. Starting from the Late 1970s, Scanmec proved that the large mill units could be both economical and reliable. Following a strict requirement from the client, Scanmec designed and supplied the innovative grinding mill installed at the Sydvaranger Mine in Norway.

This large (6.5 x 9.2M) iron ore grinding mill was the first mill of any kind supplied with a wrap-around motor (ring motor) to the mining industry. Scanmec also developed the concept of employing shell mounted pad type bearings to reduce the peak stress levels which had previously proven to be problematic. Today, former Nordberg Mills, Morgardshammar and Scanmec are member business units of the Outokumpu Mills Group. As a result of that merge Outokumpu’s mill experience dates back more than a half century when grinding technology was in its infancy. Since then, through the delivery of nearly 1000 successful installations, the engineers and metallurgists of Outokumpu Mills Group have introduce many technological “firsts” which now are accepted as state-of-the-art industry practice. Outokumpu’s reliable grinding mills are available from, and fully supported by, its worldwide sales and service network.

Outokumpu’s highly skilled technical service crews are available to assemble grinding mills, even in some of the world’s most remote locations. A complete mill erection service is offered to assure proper assembly, alignment and commissioning in the shortest possible time period.

Grinding Mills frequently are on the critical path of the project, so time saved in mill delivery and erection has a direct, positive effect on mine cash flow.

Outokumpu’s mills are provided with excellent technical service support to assure proper mill assembly and operation.
Chapter 2: Mechanical Features

Outokumpu approaches the design of grinding mills from the perspective of the process engineer. Starting from the owner’s process objectives, Outokumpu incorporates mechanical features into the design which will assure optimal process performance and very high reliability. In selecting mechanical design elements, Outokumpu considers the owner’s business objectives, at the installation site, experience of the operators and the logistics of delivering the mill unit.

Outokumpu Mills has a long experience in supplying mills employing rolling element bearings. Thermal expansion of the mill along the axis of rotation is taken up by the pedestal and rocker mounting arrangement shown here.

For many applications in remote areas or under severe climate conditions, Outokumpu can offer the traditional Morgardshammar design for mills supported on trunnions with anti-friction mill bearings. These mills utilize a simple pedestal and rocker assembly to allow for axial thermal expansion of mill rotating assembly.

The anti-friction bearings allow for very high load carrying capacity over a relatively short trunnions surface. This means that the principal stress range on the mechanical rotating assembly is reduced. Anti-friction bearings are well suited to remote locations because they rely upon simple, low pressure lubrication arrangements when compared with other bearing alternatives. Standard Morgardshammar design incorporates nodular cast iron as the base material for the ring gear and endplate (head) castings. Nodular cast irons provides a strong durable material which minimizes the chance for casting defects because of its ease of pouring and simpler foundry technique relative to cast steel.

For very low capacity installations, Outokumpu offers a range of innovative hydraulic drive mills. The use of a hydraulic drive coupled through the mill trunnions provides a very low cost solution for variable speed mill operation. Since the mill and its power unit are supplied as a module, the mill can be supplied, erected and commissioned very quickly without the need for highly skilled technicians. Outside of traditional hard rock mines, these mills offer an attractive, economic alternative for limestone grinding, for stack scrubbing in power plants, and for producers of industrial minerals.

For high capacity applications, Outokumpu designs large mill units based on the principle of shell mounted supports. In this case, rotating assembly is supported on pad type bearings riding on journals machined on the outer diameter of the mill shell. With this kind of arrangement, the ends of the mill are unencumbered by the supports. This permits the size of the feed and discharge openings to be optimized for process conditions rather then the physical dimensions of the bearings. The resulting mill unit, while maintaining the required dimensions of the grinding chamber, is shorter overall. This in turns allows for smaller, flat profile foundations and less floor space consumed. These features result in savings of both time and cost for the complete installation.

Small mill units driven by an integral hydraulic motor are a reliable and economical solution for many low capacity installations. These mills are very simple to install and operate.
Mechanically, the use of shell mounted bearings fully minimizes the support span, keeping stress on the structure to the lowest possible level. Since the load is being carried at the mill diameter and support is provided directly at the ends of the grinding chamber, fabricated endplates can be employed since they are only lightly loaded.

By completely eliminating the heavy cast endplates, shell supported mills offer proven benefits of shorter and highly reliable delivery cycles. Outokumpu is driving the technology even further by designing fully fabricated mill units.

In today’s market environment, where metals and mineral prices are relatively volatile, it is important to be able to design, supply and commission mills in the shortest possible time.

Fully fabricated mills provide the purchaser with an alternative to longer delivery cycles dictated by designs which incorporate heavy castings. Risk of delay due to unexpected complications in the foundry process is totally eliminated. The fabrication process is easily monitored, and it is relatively simple to confirm the integrity of the completed components and assemblies. This is a particularly important consideration as the delivery of the grinding mills frequently sets the critical path of the project. During erection, the unique Outokumpu-Scanmec pad bearings allow for very rapid erection and alignment. The bearing modules incorporate integral hydraulic cylinders which when activated instantly provide proper alignment and load sharing between pads. After initial erection using this technique, a single simple lock ring on each pad is tightened to assure that proper alignment and load sharing are maintained during operation.

By using the most current finite element modeling techniques to analyze and optimize the mill structure, Outokumpu is able to supply mills which will provide reliable operation and very long service life. Stress in the mill structure is maintained within acceptable limits set forth in the most stringent international standards. The designed staff is also skilled in performing fatigue analysis and torsional analysis of the complete drive system. Outokumpu mills are fully designed using computer aided design (CAD) systems so that drawings can be provided in digital files to accelerate the total project design cycle. Outokumpu is an innovative technology leader offering state-of-the-art grinding solutions for a wide variety of process applications.

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The most important benefits of Outokumpu’s shell supported mills are clearly shown in this diagram. A 10 MW shell supported mill consumes the same foundation space as a 5 MW mill supported on traditional trunnion bearings. This feature provides significant savings in the cost of civil works associated with the mill installation, as well as time required for mill erection and commissioning.
Chapter 3: Grinding Fundamentals

Mineral Liberation

The object of comminution in mineral processing is usually to liberate the valuable mineral component from waste or gangue. The decision as to how fine a grind to adopt is crucial to process economics, and is the first decision to make in evaluating the process.

In recent times, the understanding and measurement of liberation has advanced rapidly with the introduction of Scanning Electron Microscopy techniques for individual particle imaging, coupled with automated image analysis methods to provide particle population statistics. Test work is still required to verify the liberation predictions.

Feed Preparation

The feed to a grinding mills is invariably crushed to a size appropriate to the grinding process selected. SAG mill feeds require little crushing, whereas ball mill feeds require a much finer crush.

Run-of-Mine Milling is rarely practiced, with the exception of ores from narrow seam or vein underground mines. Here the blasting methods used are such that there are few large (+300 mm) lumps in the feed. A coarse mill feed is desirable only where the grinding process used is fully autogenous.

The feed to Autogenous (AG) and Semi Autogenous (SAG) mills is an important variable in the performance of the grinding circuit. It can be significantly influenced by the crushing circuit and, in some cases, by the mining operation itself. Blasting practice has marked effect on mill capacity, particularly for the softer ores.

Autogenous mills use the feed material as the grinding media. The larger the particle the more energy can be imparted, and therefore the more impact breakage is likely. Thus, a coarse feed is important for AG milling. Typically material which is over 150 mm in diameter is required in the feed to ensure adequate impact breakage, and target F80s of around 200 mm are usually sought.

In SAG milling, steel grinding media is added to the mill. The size of the grinding media has a significant impact on the rate of the breakage, with a ball of, say, 125 mm equivalent in mass to a rock of approximately 180 mm. Therefore the feed required for SAG mills does not have to be as coarse as that for AG mills. There is benefit in milling efficiency in limiting the feed size to F80s around 110 mm.

When extremely competent ores are encountered, it can sometimes be beneficial to secondary crush most or all of the SAG mill feed in order to improve power efficiency.

Rod Mill feed is usually prepared using two stages of crushing, typically to ~25 mm.

When Ball Mills are used as the first stage of grinding, it is usual to prepare a feed to ~12 mm or less, with typical F80s of 8-10 mm. Three or four stages of crushing are normally required to get to this feed size. Coarser feeds tend to produce rejects (“scats”) unless a grate discharge arrangement is used to minimize or prevent the occurrence.

Dry Grinding

Unless the process that follows grinding demands a dry feed, it is usual to wet grind. When grinding to an equivalent specific surface area, wet grinding has a lower power demand than dry grinding. On the other hand, the wear on mill linings and grinding balls tends to be lower with dry grinding, due to the absence of corrosion effects.

Moist feed materials must be dried if the moisture content is high, usually by providing a heated air stream through the mill. The classification systems associated with dry grinding tend to be very space consuming, requiring transport devices and air filters in addition to the classifiers.

Dry autogenous mills have been in use for many years. Their design differs from wet autogenous mills in that the grates are replaced by an air swept classification system. Other system, such as those using mechanical elevators ahead of the classifier, are sometimes used.

Rod mills experience reduced capacity when dry grinding, as the material hindered by the rods when passing through the mill from the feed end to the discharge end. Central peripheral discharge can be provided to partially overcome this problem. Ball mills are more sensitive to clogging than rod mills, and very dry material is required. Again, the ball charge tends to swell in dry
grinding, limiting the charge volume too 28-35%. Often, two compartment mills are used, with large balls in the first chamber and smaller media in the second. An extra grate wall, frequently called a diaphragm, separates the two compartments. Heat dissipation in dry grinding is usually the biggest problem, causing difficulties with alignment and liner systems.

**Power Considerations**

Around half the energy used in most material processing plants is consumed in grinding. Usually, it is the single biggest operating cost item, and good energy utilization is critical to project economics. Sizing of grinding mills is mostly carried out by determining the energy required for the duty and selecting an appropriate unit to deliver the energy. Often, this can be done by laboratory testing. Two forms of testing are common: the Bond grindability tests and single particle tests such as the impact test or the drop weight test.

The Bond Grinding Indices are useful in predicting the power required for rod and ball mills. They can also be used by operators to assess the power efficiency of an existing circuit, as explained below. However, the Bond BWI test is not a good predictor of the behavior of AG/SAG mills, unless adapted using empirical factors.

In order to assess AG/SAG mill behavior, single particle tests have been devised which at the energy required to break the particle under impact conditions, and the relationship between the energy applied and the size distribution of the “daughter” products.

**Bond Grindability Tests**

Fred C. Bond, one of the pioneers of crushing and grinding, derived the following formula for calculation of the required energy to reduce particles from a feed 80% passing size (F80) to a product 80% passing size (P80).

\[
W = \frac{10 Wi}{(P80)^{0.5}} - \frac{10}{(F80)^{0.5}}
\]

where:
- \(W\) = work input in kWh/t
- \(Wi\) = Bond Work Index – (usually ball work index for mill sizing) in kWh/t

such that multiplying the new feed tones by \(W\) gives the power requirement in kW, providing that the Work Index, feed and product size are known.

The testing methods for determining Bond indices are described within the Ore Testing section which follows. The is carried out dry in batch mode, and it has been determined over the years that so-called Efficiency Factors should be applied to \(W\) to drive the corrected power requirement. Rowland (1982) defined these as follows:

**Efficiency Factors**

- **EF1 (Dry Grinding)**
  
  With most materials, for the same range of work, dry grinding requires 1.3 times as much power as wet grinding. In some special cases, this correction factor can be as low as 1.1 or great as 2.0. Contact your nearest Outokumpu office for assistance with special dry grinding applications.

- **EF2 (Open Circuit Grinding)**

  For ball milling EF2, is a function of the degree of control required on the circuit product. Open circuit inefficiency multipliers are shown below:

<table>
<thead>
<tr>
<th>Product Size</th>
<th>Control Reference % Passing</th>
<th>EF2</th>
</tr>
</thead>
<tbody>
<tr>
<td>50</td>
<td></td>
<td>1.035</td>
</tr>
<tr>
<td>60</td>
<td></td>
<td>1.05</td>
</tr>
<tr>
<td>70</td>
<td></td>
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<td></td>
<td>1.20</td>
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<td>90</td>
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<td>1.40</td>
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<tr>
<td>92</td>
<td></td>
<td>1.46</td>
</tr>
<tr>
<td>95</td>
<td></td>
<td>1.57</td>
</tr>
<tr>
<td>98</td>
<td></td>
<td>1.70</td>
</tr>
</tbody>
</table>

- **EF3 (diameter Efficiency Factor)**

  Using a base diameter of 2.44 m inside liners, the correction for other diameters is given by:

  \[
  EF3 = (2.44/D)^{0.2}
  \]

  Note: The maximum value applied for EF3 is 0.914 for practical design purposes.
EF4 (Oversized Feed Factor)

\[
EF4 = \frac{R_r + (BWI - 7) \cdot [(F_o - F_e)/F_o]}{R_r}
\]

where:

- \( R_r \) = Reduction Ratio, \( F_{60}/P_{80} \)
- \( BWI \) = ball mill work index in kWh/short ton
- \( F_o \) = optimum feed size = \( Zf \cdot (13 + RWI)^{0.5} \)
- \( Zf \) = a constant, where rod milling = 16000 and ball milling = 4000
- \( RWI \) = rod mill work index in kWh/short ton

The influence of \( R_r \) should be assessed with caution in the first stage of a two-stage circuit.

Do not use this for rod mill prepared feed to a ball mill and do not apply if \( EF4 < 1.0 \)

EF5 (Fineness of Grind)

\[
EF5 = \frac{[P_{80} + 10.3]/1.145 \cdot P_{80}}{}
\]

The factor is only applied when \( P_{80} < 75 \mu\text{m} \).

EF6 (High/Low Ratio of Reduction Rod Milling)

The equation used, unless \( R_r \) is between \( R_{ro} = -2 \) and +2, is:

\[
EF6 = 1 + \frac{(R_r - R_{ro})^2}{150}
\]

where:

- \( R_{ro} = 8 + \frac{34}{D} \)

\( D \) = inside liner diameter of rod mill (meters)
\( L \) = length of rods (meters) = Rod Mill EGL \cdot 0.15

EF7 (Low Ratio of Reduction Ball Milling Factor)

If the ratio of ball mill feed to product drops below 6, the EF7 correction factor comes into effect, the extent worsening with decreasing ratios. The formula for calculating this factor is:

\[
EF7 = \frac{2 \cdot (R_r \cdot 1.35) + 0.26}{2 \cdot (R_r \cdot 1.35)}
\]

Note: Do not apply an EF7 factor greater than 2.0 without conducting continuous test work. Contact Outokumpu for assistance with these cases.

EF8 (Rod Mill Feed)

When calculating rod mill power for rod milling only, an EF7 value of 1.4 is used when the feed is prepared by open circuit crushing and 1.2 in closed circuit.

For Rod/Ball circuits 1.2 is used for the rod milling stage only, if the feed is prepared in open circuit.

EF9 (Rubber Liners Factor)

The use of ball diameters up to 80 mm are best suited to rubber liners in terms of wear resistance. Steel liners are best suited to primary ball milling applications requiring larger than 80 mm balls and rod mills.

EF9 is applied to mills with rubber lifters, as they tend to be somewhat bulkier than the equivalent steel configuration, reducing available grinding space. Rubber liners also absorb a portion of the impact energy of the steel media, reducing efficiency. Outokumpu typically assigns an EF9 value of 1.07 for rubber lined mills.

Operating Work Index

The Bond equation (Equation 1.1) can be applied to operating circuits by knowing the specific power, \( W \), the \( F_{80} \) and \( P_{80} \) values. The so-called operating work index is then corrected for mill size and inefficiency factors to obtain \( W_{oc} \), the Corrected Operating Work Index. This can then be compared with the laboratory derived index for the ore being treated, \( W_i \), to measure the power efficiency of the circuit. At Outokumpu, a further correction to the analysis is applied by knowing the ratio of rod to ball mill Work Index, and using it in the efficiency factor assessment.

Useful information on the state of the grinding mill operation is obtained from this diagnosis which may lead to recommendations on changes to the liner configuration, ball charge makeup, or slurry density.
At its comminution test center and minerals laboratory in the USA, Outokumpu conducts a variety of tests to establish ore work index. The impact test apparatus is shown here.

Limitations of the Bond Test

The Bond method has several limitations, the main ones are as follows:

- It is designed to predict power in a wet grinding circuit operating at 250% circulating load. Moving away from this condition reduces the accuracy of the test.
- It does not predict the behavior of large rocks in grinding circuit where the mode of breakage is impact dominated.
- The definition of the Bond Work Index is the energy per unit mass required to reduce a particle from “infinite” size to 80% passing 100 um, serious discrepancies can occur. The closing screen in the test must reflect the size to which the particle is to be ground. At finer sizes of around 10 um, it is usually not possible to apply Bond predictors of power.
- In two-stage ball milling, EF4 must be limited to its purpose of correcting for oversize feed, rather than allowed to dominate the power equation. A value limited to 1.2 is suggested.
- Often, the shape of the size distribution generated by two-stage grinding operation differs significantly from the shape obtained by crushing a sample to perform a Bond test. The F80 may be the same, but the amount of fines at, say, F30 or F20 will be markedly different. A correction to the distribution must be applied before the bond method will predict the power requirement of the second stage. This particularly applies to SAG/Ball milling, but also two-stage ball milling and, to some extent to rod/ball milling. (see Fines Correction). The same is true for other unnatural or scalped feed distributions. These conditions require additional grinding energy based upon the variation from a more standard feed distribution.

Single Particle Tests

Impact breakage tests dominate the methods used for characterizing ores by single particle tests. Two methods are prevalent for hard ores: the Bond Impact Test and the Drop Weight Test, each described in the Test Methods section. The former is concerned with the onset of breakage, while the latter examines the daughter products generate under increasing impact energy applications. The methods are primarily aimed at establishing the power required in AG/SAG mills, but are adapted from crushing tests.

Bond Impact Test

Barratt (1986) proposed a method for predicting SAG power involving the use of a combination of Bond Work Indices over a range of sizes from F80 to a defined P80, applying a correction factor to resultant power, and deducting the ball milling component of the power.

\[
E_{(SAG)} = \left[10W_{1c}(S_{1c}) + 10W_{1r}(S_{1r})\right]K_r + 10W_{1b}(S_{1b})K_b\] \times 1.25 \cdot 10W_{1a}(S_{1a})
\]

where: \(E_{(SAG)}\) is the specific SAG mill power in kWh/t

\(W_{1c}, W_{1r}, W_{1b}\) are the Crushing, Rod and Ball mill Work Indices

\(S_{1c}, S_{1r}, S_{1b}\) are \((1/\sqrt{P} - 1/\sqrt{F})\) for the equivalent stage size ranges

Note that the ball mill product size fixed at 110 um

Kr is composite of EF factors for rod milling, excluding EF3

Kb is composite of EF factors for ball milling, excluding EF3

It was observed that the method can be used unless the CWI and RWI are significantly higher than the BWI, in which case SABC is indicated and \(E_{(sag)}\) can be discounted by 10% to arrive at a power efficient SABC design.

Siddall, et al., (1996) classified the responses obtained from impact testing the products of a tumbling drum and related them to correction factor, designated \(f_{(sag)}\) which is applied to the
Bond Ball Mill Work Index to predict the total power required to grind from $F_{80} = 150 \text{ mm}$ to $P_{80} = 75 \text{ um}$. The equations take form:

**EQUATION 1.3**

$$P_{(TOT)} = 10 \times W [1 - (1500000)^{1/7.5}]$$

By subtracting the ball mill power requirement and correcting for feed size, the SAG mill power can be predicted:

**EQUATION 1.4**

$$P_{(SAG)} = P_{(TOT)} - P_{(CR)} - P_{(BM)}$$

Where:
- $P_{(TOT)}$ is the total circuit power
- $P_{(CR)}$ is the correction for feed $F_{80}$ size
- $P_{(BM)}$ is the correction for the ball mill power

The methods have been found to predict the single stage grinding power required in a AG/SAG mill. In both methods, there is a reliance on either pilot plant data or database correlations in order to establish $T_{80}$, and hence the SAG mill power in two-stage grinding circuit.

**Drop Weight Test**

Recently, JK Tech has introduced a Drop Weight Test to replace the Pendulum Test. Both methods are aimed at obtaining “appearance” data – i.e., the daughter product characteristics from an impact breakage event. As explained in the Ore Testing section which follows, values for $a$ and $b$ are derived experimentally. They are related as follows:

**Equation 1.5**

$$t_{10} = A\{1 - e^{-b.Ecs}\}$$

Although there are some broad correlations between the JK appearance functions and other indices, the use of the parameters is usually restricted to the steady state mathematical model, SimMet, and hence beyond the scope of this publication.

**Mill Speed**

The method of expressing the speed of a grinding mill is usually as a percentage of the critical speed, $N_c$.

**Equation 1.6**

$$N_c = \frac{42.3}{\sqrt{D}}$$

In practice, normal mill speeds range from 60 to 90% of critical, with the choice dictated by operational and economic considerations. To a first approximation, power drawn by a mill is proportional to mill speed over the range of interest, suggesting that mills should be run as fast as possible. However, the useful work done by the grinding charge is related to the mode of breakage induced, which is turn influenced by the liner design and charge level. Higher mill speeds lead to higher wear of rods, balls and liners. Figure 1 illustrates the trajectory of the outer envelope of the charge at increasing speeds for the same ball size with two lifter designs. Studies such as this have produced the following general guidelines:

**AG Mills**

An impact mode of breakage is usually sought, and with no steel media in the mill it is possible to run speeds in the range 80 – 90% $N_c$.

**SAG Mills**

Typical operating speeds are around 75% $N_c$. Liner damage will occur if the balls are allowed to impact them directly, and SAG mills are commonly applied with variable speeds drives.

**Rod Mills**

Rod mills operate at a lower speed than ball mills to ensure that there is no cataracting of the rods. Typical speeds related to inside shell diameter are:
**Ball Mills**

Smaller mills can be run at high speeds up to 85% Nc, medium diameter mills at lower speeds – 70-72% Nc. There is an emerging trend of operating very large mills (> 5 m dia.) at higher speeds – typically 76% Nc – in an effort to overcome an inactive kidney problem. As general guide, Equation N° 1.7 can be used below 5m.

\[ \% \text{ Nc} = 83.8[D]^{-0.108} \]

**Linear and Speed Effects**

Figure 1 also shows the effect of differing lifters on the trajectory of balls in a ball mill for fine grinding, it is desirable to have the charge cascading, rather than cataracting. This is achieved by selecting a lower mill speed and/or using a wave liner profile. For breakage of larger feed particles, the grinding balls should strike the charge close to the toe. Higher lifter bars and mill speeds will assist. Outokumpu offers a liner profile analysis service for existing mills, and uses the analytical procedure when supplying new mills to ensure the correct application of the lining system is engineered into the mill. An overview of this system is explained in chapter 7.

**Fines Correction**

The product from the first stage of grinding, i.e., an AG mill, SAG mill or rod mill, usually has a different size distribution than that produced by crushing to prepare ball mill feed. Invariably, more fines are present in a ground ball mill feed. In order to correctly predict the ball mill size required in a secondary milling application, it is necessary to modify the mill feed size by removing finished product from it. Figure 2 shows size distributions for the two methods of feed preparation an clearly illustrates differing fines content at the same P80. Typically, a partition curve is applied to the SAG product at the size of the separation required to generate final product. The result is that only a fraction of the SAG product requires secondary grinding, and this daughter product exhibits a coarser size distribution than its parent.

**Classification and CLR aspects**

Efficient classification is an integral part of any closed grinding circuit. SAG mills generally operate in the range of 50 – 150% Circulating Load Ratio (CLR), while ball mills are normally operated at 250 – 350% CLR. CLR is best measured in the plant using mass flow to the cyclones. However, there is a standard method which uses the size distributions of the streams to derive a mass balance. Use these techniques to check that the mill is grinding the optimum tonnage by maintaining the target CLR.

Ensure that the classifier is performing well by analyzing its behavior on a regular basis.

Screens, classifiers and hydrocyclones can be used as classification devices.

**Optimization**

Mass balancing and screen analysis of the process streams, in conjunction with a proper power analysis, provide the data with which to optimize any grinding circuit. Usually, the circuit is surveyed over a short period of 10 to 60 minutes by carefully sampling all streams. Techniques such as operating work index calculations, f(SAG) determination or even a full modeling of the circuit can be applied to ensure efficient operation.
Figure 1  Effect of Mill Speed and Lifter / Liner Profile on Mill Operation

Figure 2  Size Distribution Illustrating Fines Correction
Chapter 4: Ore Testing

In order to quantify what type of grinding circuit is best suited to the ore and the size of mills required, it is necessary to subject samples to crushing and grinding test work. Depending on the type of milling circuit being considered, test work can range from simple determinations based on small sample of rock or core, up to comprehensive pilot scale testing requiring hundreds of tones of whole ore.

The most commonly used tests are:

- Bond grinding indices (rod, ball and abrasion)
- Unconfined compressive strength
- Impact crushing test
- Autogenous tumble test
- JK drop weight
- Pilot scale milling

Bond Grinding Indices

There are many test facilities around the world that can undertake these tests, which require standard Bond mills made to exacting specifications and have specific sample makeup needs. Outokumpu operates a fully equipped comminution test center capable of performing the full range of grinding tests.

The ball mill work index determination requires a small 10 kg of drillcore or rock, which is crushed to –3.35 mm (6# Tyler series). The test enables basic grinding power requirement in ball mills to be determined, from the feed 80% passing size (F80) to circuit 80% passing size (P80), using the classic Bond formula.

The rod mill work index test requires 20 kg of material, which is crushed to -12.7 mm (1/2") and tested in a standard Bond Rod mill. The sample is ground to –1.18 mm(14# Tyler) to emulate the duty of a primary rod mill in front of a secondary ball mill. The rod mill index derived from this test is used in conjunction with the ball mill work index to determine rod mill power demand, again using the Bond power equation. This test is also commonly available throughout the world.

The abrasion index test requires only 5 kg of material, which is crushed and screened to an exacting size range of + 12.7 -19.0 mm (+1/2" – ¾"). The test uses a small laboratory scale mill with a test paddle that is weighed before and after being rotated in contact with the dry test sample. The difference in weight is designated as the abrasion index, and is used in conjunction with Bond formulae to predict liner wear and media consumption in rod and ball mills, as well as in crusher lines.

Unconfined Compressive Strength (UCS)

This test has been developed to provide the opposite of tensile strength, in that the strength of a rock sample under compression by a single vertical force is determined. The test requires the use of a specialized compression device which applies an evenly controlled force to the rock until failure. Unfortunately, the test is undertaken in many different types of devises, with widely varying sample specifications, which makes cross-comparison of results difficult at times.

One international standard that is used widely is the ASTM2938-86, which conforms to a sample specimen machined into a cylinder featuring a length twice that of the specimen’s diameter, which is ideally 50 mm (2"). The test produces two outputs, the first being the mode of breakage, which can provided insight in to the nature of the rock. The second output is the actual UCS value, which is usually quoted in Mpa. The UCS value is used to guide crusher manufacturers in selection of appropriate equipment, and to assist the grinding consultant with assessing an ore’s competency.

Impact Crushing Test

These tests can take two forms. The first is the Standard Bond Crushing Test, which has a requirement of twenty pieces of rock or core of size + 50 – 75 mm (+2” – 3”). Pieces are placed in a twin pendulum device (Figure 3) and impacted to failure to produce an impact crushing strength, measured in kWh per tonne of ore.

Twenty specimens are tested to provide a measure of variability of results, as there is a tendency towards heterogeneity in rocks at larger sizes. The standard index is used primarily by crusher manufacturers to assign downrating factors for ore toughness in crusher selection.

The second test is somewhat more meaningful and features a larger sample of rock or drillcore. In a typical evaluation, around 120 kg of material is...
tumbled in the standard 1.83 m diameter x 0.3 m wide Bond autogenous media competency test mill for 500 revolutions at 26 rpm. This is done to eliminate imperfections in rocks and to effectively mimic seasoned pebbles in a mill charge. The drum product is screened to remove −19 mm material, and the remaining rocks are sorted into four to five sizes classes, depending on the size of core. These size classes are 19 x 25 mm, 25 x 38 mm, 38 x 51 mm, 50 x 75 mm, and 75 x 100 mm. Twenty (20) rocks in each size class are selected and subjected to standard Bond Impact Crushing Work Index determination. The test machine pendulums can be adjusted so that they strike the specimen normal to the vertical axis. The test provides the raw data required to drive an impact crushing profile, which is valuable tool in identifying the type of comminution circuit that an ore samples is best suited to (Figure 4).

**Autogenous Tumble Test**

When sample is available in the form of lump rock, the standard Autogenous Tumbling Test can be carried out, using 10 rocks in five sizes ranges between 102 and 165 mm. The rock is normally tumbled for 500 revolutions in a 6 ft x 1 ft drum and the product sized. The drum products provides raw data to enable the following evaluation to be made by the grinding consultant:

- interpretation of the product distribution against generic curves
- production of media in AG and SAG mills
- The amount of critical size build-up
- The tendency for ore to generates fines (-6 mm material
- overall amenability to autogenous milling

The test provides excellent insight into impact breakage and auto abrasion characteristics of ores, but is currently only performed in a few laboratories around the world.

**JK Drop Weight Test**

This test has been devised by the Julius Kruttschnitt Mineral Research Centre (JKMRC), and is used to derive impact breakage and abrasion parameters for use in their simulation package, JKSimMet. The method involves dropping a metal weight from a set height onto a test specimen and sizing the “daughter” products from the resultant rock failure (Figure 6). A number of specimens of varying sizes are tested to generate breakage curves from which the simulation parameters are calculated. The test is useful adjunct to the media competency tests described above in that once the type of circuit that is best suited to the ore is identified, the JKSimMet simulator can then be used as one means of verifying initial mill sizes.

**Other Tests**

**Macpherson Test**

The most notable method in use in the Americas is the test developed by Art Macpherson. It uses dry grinding on material typically crushed to −38 mm to ascertain the autogenous characteristics of an ore in a 450 mm diameter mill. The results are then compared to a Standard Bond tests, and an empirically scaled value for the amount of power that is theoretically required to grind the ore is determined. The test is normally used as a
precursor to pilot scale AG/SAG milling. The main area of concern with the test is the underlying assumption that ore at large lump sizes behaves in a similar manner to the small sizes used in the test at ~38 mm, which is not consistently correct. This is particularly so with tough siliceous ores (typical of the greenstone belts in Australia and parts of Africa). However, the test can serve as useful adjunct to the other tests discussed above in providing some insight into the autogenous characteristics when whole ore is not available for testing. It generates an estimate of the product size from an AG/SAG mill.

**Hopkinson Pressure Bar**

The JKMRC in Australia is developing this test to study the breakage properties of rock materials. The device consists of a 6.4 m long steel bar suspended horizontally (Figure 7), with a test rock specimen attached to one end. The rock is impacted and compressed by the rebound energy of the impact to a known distance to control the impact velocity. The speed of the impact bar at the point of impact is measured using an optical sensor, and converted into input energy for rock failure. The test is significantly different from the Drop Weight Test, which measures “daughter” products resulting from the Drop Weight Test, which measures “daughter” products resulting from rock failure, in that the threshold energy for impact breakage of a single rock particle can be determined. The test may find valuable application in assessing critical size problems in SAG mills and aid in crusher selection.

**Starky Test**

This test has been devised by Minnovex in Canada to predict SAG mill specific power requirements using only ~12.7 mm material. The test uses a small 300 mm dia x 100 mm long laboratory scale mill with a small ball charge of 25 mm balls to grind the 2 kg test sample. The objective is to establish the grinding time required to grind the ore to 80% passing 1.7 mm (10#), the closing screen size. This test claims to demonstrate a strong correlation between grinding time for ores and their corresponding SAG mill specific power draw. The test is considered to be still in its developmental stage, but it could prove to be an attractive alternative to tests requiring large sample size. The concern expressed about the Macpherson test also applies to this test, perhaps more so, as the Macpherson test at least has a very large database for comparison with actual operations, which provides the basis for calibration of the autogenous index against laboratory results.

**Pilot Scale Testing**

In the majority of “greenfields” projects, it is usually not possible to access whole ore in the early stages of the study; hence, the focus on drillcore testing. However, when whole ore is available from a current operation or from a development audit or shaft, and depending on the scale of the proposed operation, it is possible to undertake pilot scale SAG mill testing. As a rule of thumb, for circuit of less than 2 MTPA capacity, piloting is usually not justifiable, with the cost of such a venture usually incorporated into extra mill length and/or motor rating.

For simple AG or SAG mill piloting without online downstream piloting of other unit processes (such as flotation or solvent extraction) 100 – 150 tonnes of ore are required, with campaign duration being 10 to 15 working days in a test facility. For more complex arrangements, campaigns have been known to run over two months, with corresponding escalating costs. A number of excellent test facilities are available around the world to carry out a pilot test program.
Circuit Surveys

When the mill selection being considered is the result of an intended plant upgrade, it is highly desirable to obtain survey data in the form of mass balances and sizing data. This should be supplemented by process information such as milling rate, power draw and equipment configurations (operating ball charge, total mill charge volume, milling speed, cyclone parameters, etc.) This data can be used to provide input information for power based modeling, or for more sophisticated breakage rate based simulators.
Chapter 5: Rod Mills

Application information

Rod mills (Figure 8) are used in coarse grinding applications usually to prepare ball or pebble mill feeds. The availability of good quality rods is currently limited to sizes less than 6.08 m (20 ft) which restricts capacity. They originally enjoyed widespread use in many applications, but with the trend to higher capacities in single stage grinding lines, their use is generally limited to specialized situations such as:

- a very coarse product size is required, such as primary grinding application in bauxite digestion
- overgrinding is to be minimized, as rod mills provide more selective grinding than ball and SAG mills
- the ore is very tough, making it unsuitable for SAG milling without secondary feed crushing
- dry grinding applications such as for coke breeze in the iron ore industry
- as a preparatory step prior to magnetic cobbing

Feed size to rod mills can be anywhere from 80%-4 to 20 mm, although optimum efficiency is normally seen in the range 15 to 17 mm, with a reduction ratio of around 15:1.

Dry rod milling is generally discouraged due to poor material flow and charge swelling leading to rod breakage and tangling. It is power inefficient, and, except in a few specialized applications, is usually not considered.

Rod mills require more stringent L/D rations than balls mills, in the range 1.4 to 1.6, to prevent rod tangling. Rod length is generally restricted to 6.08 meters (20 ft), with 150 mm clearance being required to the ends of the mill. This in turn limits the maximum diameter offered to 4.5 m inside shell.

Unlike ball mills, rod charge volume is not restricted with increasing mill size, with volumes of 40 to 45% volume being common. When dry rod milling, lower charge levels of 30 to 35% are recommended.

<table>
<thead>
<tr>
<th>Mill Diameter (meters)</th>
<th>Recommended Milling Speed (rpm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>2.40</td>
<td>19.0</td>
</tr>
<tr>
<td>2.70</td>
<td>17.5</td>
</tr>
<tr>
<td>2.90</td>
<td>16.8</td>
</tr>
<tr>
<td>3.25</td>
<td>15.6</td>
</tr>
<tr>
<td>3.50</td>
<td>15.0</td>
</tr>
<tr>
<td>3.80</td>
<td>14.3</td>
</tr>
<tr>
<td>4.15</td>
<td>13.6</td>
</tr>
<tr>
<td>4.50</td>
<td>12.9</td>
</tr>
</tbody>
</table>

Source: Design and Installation of Comminution Circuits, Rowland, 1982: p. 405

Test Requirements

Bond rod and ball mill indices are required on typical material that will be processed. The minimum sample requirement for the tests is 25 kg of either whole rock or split NQ (50 mm dia) drillcore. Testing of multiple samples from different locations in the mine is recommended to determine the extent of hardness variability.

Typical Flowsheets

Typical flowsheets are provided in Figure 9 and include:

- open circuit single stage rod milling
- open circuit rod milling followed by closed circuit ball milling (or pebble milling)

Closed circuit rod milling is uncommon, unless there is a tight tolerance on oversized material in the product, in which case closing with a DSM screen is appropriate. Open circuit rod mills do not suffer the same penalties as ball mills, which incur 20% inefficiency without classification.
Rod mills are generally limited to a diameter of around 4.5 meters due to the constraints of the maximum rod length, and motor sizes of the order of 1500 kW (approx. 2000 hp). For this reason, rod mills are generally only used as primary grinding units in applications of up to approximately 1.8 MTPA capacity.

Specified mill power is determined by using a combination of the rod and the ball indices for the ore, incorporated into the standard Bond power formula. Actual pinion power draft for a given mill size is determined using the following formula, developed by Bond:

\[
\text{EQUATION 3.1} \\
\text{pinion power} = M_c \times \left[ 1.752 \times D^{0.34} \times (6.3 - 5.4 \times V_p) \times C_s \right] \\
\text{(KW)}
\]

Where:

\[
M_c = \text{x mass of grinding charge, incorporating rod SG, voids, % fill, and mill volume}
\]

\[
D = \text{inside liners mill diameter, in meters}
\]

\[
V_p = \text{fraction of mill volume loaded with rods}
\]

\[
C_s = \text{fraction of critical speed}
\]
Chapter 6: Ball And Pebble Mills

Application information

Ball mills are tumbling mills which operate with a charge of spherical steel grinding media. They are by far the most common type of grinding unit in use today.

Ball mills can be either of the overflow discharge or of the grate discharge type, with typical designs being presented below.

Grate discharge ball mills allow up to 16% more power draw for a given mill size than an equivalent overflow discharge mill. They do, however, demonstrate higher media consumption rates and an inability to purge broken balls from the mill which continue to draw power but do not contribute significantly to grinding. As a result, lower efficiency is experienced when using grate discharge mills, which tends to negate the increased power draw available. They are mainly used when there is a differential in specific gravity (SG) between the valuable product and the rock from which it is liberated, such as coarse gravity gold.

The grate provides a positive discharge mechanism for the heavy component which would otherwise build up in the mill (Figure 10).

Wet overflow ball mills (Figure 11) allow the product to exit the mill by overflowing the discharge trunnion. As mill size and capacity increase, so the ball load used decreases. A load level of 35% by volume is typical of larger mills. Unground rock and worn balls can exit the mill, and it is common to fit a segregation device such as a trommel or discharge screen to remove them from the slurry flow.

They can be applied as primary reduction devices receiving feed from secondary or tertiary crushing circuits with sizes up to 80% -25 mm for soft feed or as fine as 6 mm for very hard feed. Overflow ball mills differ from rod mills in that they can either generate a coarse product to be fed to a secondary grinding unit, or can grind to final product size as a single stage mill.

Operating speeds are approximately 72 to 77% of critical, which is dependent on the mill diameter. Guidelines for mill speeds at various mill sizes are given in Table 3. Higher speeds than those stated in the table are possible but liner wear will increase accordingly. Ball mills are generally preferred over other circuit types when:

- ore is of low abrasion index and hardness
- grinding in a single stage to final product size without undue slaming is desirable

<table>
<thead>
<tr>
<th>Mill Diameter (meter)</th>
<th>Recommended Milling Speed (rpm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>2.0</td>
<td>23.33</td>
</tr>
<tr>
<td>2.4</td>
<td>20.75</td>
</tr>
<tr>
<td>2.8</td>
<td>18.96</td>
</tr>
<tr>
<td>3.2</td>
<td>17.50</td>
</tr>
<tr>
<td>3.6</td>
<td>16.27</td>
</tr>
<tr>
<td>4.0</td>
<td>15.86</td>
</tr>
<tr>
<td>4.4</td>
<td>15.12</td>
</tr>
<tr>
<td>4.8</td>
<td>14.48</td>
</tr>
<tr>
<td>5.2</td>
<td>14.10</td>
</tr>
<tr>
<td>5.6</td>
<td>13.59</td>
</tr>
<tr>
<td>6.0</td>
<td>13.12</td>
</tr>
</tbody>
</table>

Ball Charge Regimes

Table 4 provides suggested initial charge make ups for various ball top sizes. The ball top size requirement is driven by a combination of feed size, mill diameter and speed, and ore specific gravity.
Test Requirements

Bond rod and ball mill indices are required on typical material that will be processed if the ball mill is intended for a primary milling duty, receiving feed of a size exceeding 80% -2 mm. For secondary milling duties, a ball mill work index determination will suffice. The minimum sample requirement for the rod and ball index tests is 25 kg of either –12.7 mm (1/2") crushed rock or split NQ (50mm dia.) drillcore. For the ball mill work index test alone, only 10 kg of material is required. Testing of multiple samples from different locations in the mine is recommended to determine the extent of hardness variability.

Typical Flowsheets

Typical flowsheets featuring ball mills as the main grinding unit are provided in Figure 12 and include:

- closed circuit single stage ball milling
- closed circuit series ball milling

Open circuit ball milling is uncommon, as such circuits incur a 20% inefficiency due to lack of control over the product top size. Ball mills normally operate in closed circuit with a classifier such as hydrocyclones or DSM screen.

Standard Sizes

Outokumpu has been building large ball mills for the last three decades, the largest being a 6.5 meters dia x 9.7 meters 9.65 MW unit supplied to a client in the CIS. The trend in the 1990s and beyond is toward even larger mills, with units of 6.7 meters diameter and power ratings in excess of 10 MW now commonly being specified. These larger mills present their own unique problems such as the requirement of twin pinion arrangements or gearless (ring motor) drives to provide sufficient power draw. The larger ball mills also require lower operating ball charges.

TABLE 4

<table>
<thead>
<tr>
<th>BALL MILL INITIAL CHARGE REGIMES</th>
<th>Proportion of Makeup (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ball Top Size (mm)</td>
<td>100 80 65 50 40 25</td>
</tr>
<tr>
<td>100</td>
<td>28 22 18 14 11 7</td>
</tr>
<tr>
<td>80</td>
<td>31 25 19 15 10</td>
</tr>
<tr>
<td>65</td>
<td>36 28 22 14</td>
</tr>
<tr>
<td>50</td>
<td>43 35 22</td>
</tr>
<tr>
<td>40</td>
<td>62 38</td>
</tr>
</tbody>
</table>

Ball mills usually operate with ball charges ranging from 30 to 40 % by volume. Smaller mills can handle 45 % charge without suffering from inefficiencies, but large diameters mills require lower ball charges with a minimum of 25 % being desirable to overcome the “inactive kidney” phenomenon, whereby the central region of the ball charge imparts little useful grinding action as a result of an excessive number of ball layers. This can be partially overcome by using a larger ball top size and speeding up the mill.
Shell supported ball mill supplied to Petchenka Nickel in Russia. This 9.6m mill was supplied complete with 10 MW ring motor drive.

Ball mill specific power requirement is determined using standard Bond formulae, incorporating the appropriate correction factors, (EF’s) described earlier. Actual pinion power draw for a given wet overflow ball mill size is determined using the following formula (developed by Bond):

Pinion power = \( M_c \times [4.879 \times D^{0.3} \times (3.2-3V_p) \times C_s(1-0.1/2^{9-10C_s})] + S_s \) (KW)

Where:

- \( M_c \) = mass of grinding charge, incorporating ball SG, voids, % fill, and mill volume
- \( D \) = inside liners mill diameter, in meters
- \( V_p \) = fraction of mill volume loaded with balls
- \( C_s \) = fraction of critical speed
- \( S_s \) = slump factor, in kilowatt hours per metric tonne of balls

Factors for grate discharge mills vary from 1.08 for high charge mills 1.16 for low charge mills. This factor is multiplied by the overflow pinion power to arrive at the grate discharge ball mill power.

**Pebble Mills**

Pebble mills use pebbles generated from the ore as grinding media. The pebble must meet certain criteria to be used in the manner such as the following: high impact strength, “round”, high SG, etc. These features enable the pebbles to act as efficient grinding media and minimize media and resist chipping and breakage. This allows the charge to have maximum grinding capability, instead of having chipped or flat pebbles which do not tumble and are basically wasted volume, hence power.

The higher the SG, the higher the power draw of the mill. This is comparing steel media with an SG of 7.8 with ore of SG 2.5 upwards.

Pebble mills can be power modeled in a similar way to ball mills, ensuring that the appropriate SG is taken into account.
Chapter 7: Fully and Semi Autogenous (AG and SAG) Mills

Conventional grinding options, consisting of multi-stage crushing followed by rod and ball mills, are fast becoming obsolete in the move toward lower metal grades and higher milling throughput requirements. Fully autogenous (AG) and semi autogenous (SAG) grinding mills are commonly applied in the high capacity circuits for processing ores with lower metal grades. Most AG/SAG mill circuits require only a single stage of feed preparation and in some instances run of-mine ore is fed directly to the mill.

Autogenous mills rely on the ore itself to serve as the grinding media and perform the size reduction work. AG mills treat competent ores that do not exhibit tendencies towards critical size build-up resulting in excessive coarse lump retention. In order to provide sufficient media, the feed size is usually much coarser than a SAG mill feed, and can even be run-of-mine, requiring scalping at a coarse size of 300 to 400 mm.

SAG mills rely on steel grinding media to supplement deficiencies in rock media in the mill charge resulting from poor rock integrity. SAG mills operate at various ball and total charge levels. This creates a specific charge density which dictates the power draw depending upon ore SG. Ball charge regimes are very different to that of balls mills, in that the top size can range from as low as 65 mm to as high as 150 mm in diameter. Ball charge volume ranges from as low as 1 – 3 %, typical of operations treating competent reef quartz, to as high as 15% volume for soft laterites. SAG mills utilize rock as well as steel grinding media to grind ore. The steel media supplement the large rock lumps required for impact breakage. Intermediate size classes are also required for efficient grinding by attrition and abrasion.

SAG mill sizing is based upon comprehensive test work, either laboratory or pilot scale. The test work gives results which indicate “ore character” and appropriate power modeling can be based upon similar ores. A specific circuit efficiency is allocated to the ore and a required power calculated based upon Bond’s equation.

Variable speed drives are becoming more common on AG/SAG mills especially in operations with variable ore characteristics. They impart additional operational flexibility which is valuable for circuit stability.

High Aspect vs Low Aspect Mills

Depending on the application, the mill configuration can vary greatly. There are two main configurations supplied by Outokumpu, these being referred to as high aspect and low aspect. These terms refer to the ratio of effective grinding length inside liners (EGL) to mill diameter inside liners, or L:D ratio. High aspect mills usually encompass the range 0.3 to 0.6, whereas low aspect mills can range from a “square” mill of 1:1 to ratios as high as 1:6.

Low aspect mills (Figure 13) commonly called “Scandinavian mills” or “South African style mills” are usually the preferred configuration for treating ores in single stage circuits, and for situations where a fine grind is required. The long grinding chamber of low aspect mills provides more grinding per pass through the mill resulting in manageable mill circulating loads. These mills are best suited to grinding low to medium competency ores such as the reef quartz ores of South Africa or the weathered epithermals of South East Asia’s “Rim of Fire.”

If the ore is competent, low aspect mills can encounter difficulty in achieving effective impact breakage, with abrasion grinding becoming dominant, which results in inefficiency. In this instance, the high aspect “North American” mills are used, both in single stage circuits and open circuit arrangements with secondary ball or pebble mills. The high diameter coupled with short grinding chamber length favors the good impact breakage and flushing of fine material out of the mill that is required to mill competent ores efficiently. A high aspect mill of the same volume as an equivalent low aspect mill will draw more power, so the trend is toward installing large high aspect mills to cater to the elevated duties of porphyry copper operations in South America and South East Asia.
Mill Sizing Consideration

AG and SAG mills currently face limitations which are electrical and mechanical in nature.

The largest mills designed to date are 12.2 meters in diameter with 20MW installed power. Even larger units of up to 13.4 meters diameter are now being considered. At these sizes, engineering tolerances are very small, as stresses imposed on mill shells, heads and drives are very high. Also, as the limits of current international foundry capabilities are pushed (because of the very large size and weight of the components), there is more risk associated with the delivery of such large mill units of fully fabricated design, using shell mounted bearings to reduce the peak stress levels.

The largest single pinion drive on offer currently is 6.75 MW, with plans to go to 8 MW in the next few years. Therefore, in order to provide the + 20 MW power demanded by the world’s largest mills, it is necessary to go to bigger and bigger gearless drives which increase the capital cost of the mill significantly.

Breakage Rate/Population Balance Model

Another method of modeling the grinding process within AG and SAG mills uses population balances of the various particle size fractions.

The population balance assumes the mill is in steady state and therefore the amount of material within a particular size fraction must be constant. That is, the discharge from the mill must equal the amount of material that has resulted from breakage of larger size fractions, less any of the contents selected for breakage into smaller size fractions.

Discharge rates from the mill depend on the mill contents and the classification of the particles through the grates and pebble ports. The ability of the contents to pass through the grate becomes a mass transfer problem.

The material selected for breakage depends on the mill contents and also the breakage rate of each fraction. Empirical correlations of the breakage rates with varying operating conditions and mill dimensions have been developed.

Material arising from the breakage of larger particles is estimated in terms of an appearance function. This defines the particle size distribution resulting from the breakage of an individual particle, through the combination of mechanisms including impact and abrasion.

Modeling requires the fitting of the breakage rates to plant sizing data around the mill. This can be a tedious task, especially if the data used is not of good quality.

Power Model

For SAG mills, mill power draw is not only a function of mill size but charge density as well. For ball and rod mills, the charge is predominantly steel, but for SAG mills this is a mixture of steel and rock media, as well as slurry.

Power draw theory is based upon a charge load in the equilibrium, and relates to its center of gravity. Figure 14 describes this relationship visually. The following formula relates these values to the actual power draw:

\[ N = c \times W \times R_g \times n \]

Where:

- \( N \) = the gross mill power, in kW
- \( c \) = is a constant, nominally 1/1200 for SAG, AG and pebble mills
- \( W \) = the weight in kg of the charge
- \( R_g \) = the distance in meters of the center of gravity from the mill center
- \( n \) = is the speed of the mill in rpm

The power model is such that the mill power is more sensitive to diameters than length. Diameters affect power draw exponentially, whereas the relationship between length and power is linear. Therefore, incremental changes in diameter provide step changes in power draw. The trade-off is that the capital cost of the mill also climbs steeply with increasing diameter partly due to manufacturing methods and also as a result of increasing surface area and greater load on the mill structure.
Outokumpu utilizes unique software that enables the mills to be accurately sized from a given power draw, for both AG and SAG applications.

Application information

AG and SAG circuits are now widely accepted for the following reasons:

- the process is well understood, and its behavior can be predicted
- it is cost and power effective versus multi-line crushing
- lower maintenance requirements than a fine crushing circuit
- high availability
- well suited to high-clay or sticky ores
- very high throughput in a single line

SAG or AG milling usually offers superior net returns on a project when compared to more than two lines of conventional crushing or other grinding solutions, but there are exceptions:

- slimes generation is problematic to downstream processes such as flotation
- the ore is so competent that the mills retains a disproportionate amount of lump in the mill that cannot be dealt with by a ball charge or pebble extraction and crushing
- the ore is very non-abrasive, resulting in low media and liner consumption rates in all equipment

Outokumpu offers very large cone crushers and the unique Water flush crusher which can be applied with conventional ball mills to address many of these difficult ores.

ABC and SABC

Ore characterization in the form of autogenous and impact crushing tests will reveal an ore’s tendencies toward critical size build-up of tough pebbles (30 to 80 mm size range) that inevitably results in poor grinding efficiency. To combat this problem, it is common to install pebble relief ports or slots, of width 50 to 90 mm. Depending on the outcome of testing. Critical size pebbles can then be extracted, screened and conveyed to a pebble crusher, which is a more effective means of imparting efficient breakage. Crushed material at –10 to 15 mm is usually recycled back to mill feed for subsequent grinding by the ball or rock charge. The resulting circuit, comprising a semi-autogenous mill, ball and crusher is known as SABC, while the equivalent autogenous circuit is designated ABC (Figure 15). As a guideline, if the impact work index of the critical size material peaks at around 1.5 time the ball mill work index, a crusher will show economic benefit in reducing $f_{SAG}$, sometimes to unity.

Consideration needs to be given to providing affective means of removing steel scats and tramp metallics from the pebble crusher feed, avoiding damage to crusher liners. This consists of a combination of a belt magnet, magnetics sensor after the magnet and a diversion gate to allow automatic bypass of the crusher in the event that metal on the belt is sensed.

Shell supported ball mills supplied by Outokumpu Mills A/S. Use of shell mounted pad bearings allows the feed and discharge arrangements to be configured to suit process parameters.
Test Requirements

In order to determine which type of mill to install whether it be high or low aspect, SAG or AG, a comprehensive ore testing program is required. Test requirements are detailed in the section on Ore Testing, but specific attention needs to be given to determining an ore's impact response, autogenous characteristics and grindability.

Standard Sizes

Outokumpu has a long history of excellence in the supply of low aspect SAG mills, but has in recent years moved into the manufacture of high aspect mills to meet the growing demand in the area.
Chapter 8: Process Control

The Need for Process Control

The grinding process is invariably used to prepare ore for further downstream processing, usually in the form of flotation or leaching. These downstream processes can be grind size dependent, with recoveries of the valuable mineral drastically affected if the optimum grind size is not achieved. This and throughput have obvious implications to the economics of the project.

Process control normally operates on three levels:

1. Stabilizing Control
2. Supervisory Control
3. Optimizing Control

Stabilizing control is the base level control actions which are used to stabilize the grinding circuit and overcome fluctuations in the input variables. Disturbances within the process are ever present, and the interactions of these disturbances can adversely affect the throughput and/or the final product size.

Variable Measurement

To be able to control a variable, it is necessary to be able to measure the variable or at the very least deduce its value from other measurements.

The most common measurements taken around the grinding circuit are:

1. Hopper level
2. Mill weight
3. Feed rate
4. Water and slurry flow rates
5. Slurry densities
6. Cyclone pressure
7. Product particle size
8. Mill power
9. Mill sound patterns
10. Slurry viscosity

Outokumpu has developed the Proscon Control System to control not only the grinding circuit, also for Flotation Circuit and Hydrometallurgy applications.

Hopper levels are usually measured using an ultrasonic device. These are non intrusive and can be used for a variety of fluids. Other continuous level indicators commonly in use are capacitance probes, differential pressure cells, and bubble tubes.

Mill weight is commonly measured with a load cell positioned under the mill bearings. It can also be inferred from a reading of the bearing pressure.

Ore feed rates are usually measured by a weightometer, or belt weigher. This takes the form of a conveyor passing over a cantilevered load cell.
Magnetic flow meters are the most common method used to measure both water and slurry flow rates. However these meters are not suitable for slurries which magnetic in nature.

Slurry density is measured by a nucleonic density gauge, which functions on the attenuation of a radioactive source by the solids in the slurry stream.

The PSI 200, Particle Size Indicator, with physical measurement.

Slurry viscosity may be measured and controlled for some ores where it is known that viscosity can be a problem. Several companies market on-line viscometers.

The remaining instruments are reasonably standard transducers which are readily available in the marketplace.

The basic strategy is similar for all circuits, and is described for the SABC circuit.

**Control Strategy**

The control loops described are shown in Figures 17, 18, and 19.

The objective of the control strategy is to achieve the desired product $P_{80}$ size, at a throughput exceeding the minimum value, with minimum disturbance to the process.

The feed end water addition to the SAG mill is measured and controlled to a set point that is ratio from the feed tonnage signal output from the weightometer. The water flow set point is calculated from the ratio, which is either an operator input or an output from the viscosity controller. If the viscosity increases, the ratio is adjusted to increase water addition to the SAG mill.

**SAG Discharge End Water Addition**

The discharge end water addition to the SAG mill is measured and maintained to the set-point which is output from the cyclone feed density controller. The density meter measures the pulp density of the cyclone feed stream and the water addition rate is adjusted to control the density to the operator input set-point. This set-point may be automatically adjusted by the output of the viscosity controller in the future.

This variable is important in achieving the desired final product size and density.

**Ball Mill Recirculating Load**

Both the density and flow of the cyclone feed stream are measured, and the solids mass flow calculated. This is compared to the feed tonnage to give the circulating load around the ball mill circuit. This controller has an operator input set-point, with the output going to a logic block for the mill feed rate control.

This variable is important in achieving the desired final product size, which is measured via the particle size analyzer.
SAG Mill Load Control

The mill feed rate is measured by a conventional belt weightometer. The output of the controller is sent to the apron feeder speed controller. The signal is also used to calculate the circulating load, as described in previously.

The set point for the feed rate controller comes from a logic block, receiving inputs of circulating load, mill weight, and mill speed. The logic block will assess these variables and take actions as described below:

- If the SAG mill load (as detected by the load cell) is rising, increase speed if possible.
- If the speed is at maximum and the load is still rising, back off the feed rate.
- If the ball mill circulating load is high, divert some of the load back to the SAG mill.
- If the tonnage is still limited by the ball mill capacity, and the SAG load is falling, cut the speed of the SAG mill.
- If the SAG load is low and the ball mill circulating load is not at the maximum, increase the feed rate.

This logic is presented in tabular form as Table 5.

The strategy should be used in conjunction with short term control of the mill dilution water and the product particle size control.

In summary, the system will increase feed tonnage until the constraints of either mill weight or recirculating load are met.

Mill Power Indication

Both the SAG mill and the ball mill will provide power indicators as inputs to the control system.

Cyclone Pressure

The cyclone feed pressure indication signal will be integrated with the control system. This variable is important for efficient circuit classification.

SAG Mill Speed Control

The control strategy for SAG mill speed is incorporated in the SAG mill load control algorithm.

**TABLE 5**

<table>
<thead>
<tr>
<th>MILL LOAD</th>
<th>CIRCULATING LOAD</th>
<th>MILL SPEED</th>
<th>ACTION</th>
</tr>
</thead>
<tbody>
<tr>
<td>Increasing OK not at maximum</td>
<td>Increase Mill Speed</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Increasing OK maximum</td>
<td>Decrease Feed Rate</td>
<td></td>
<td></td>
</tr>
<tr>
<td>OK High</td>
<td>-</td>
<td>Divert U/F to SAG</td>
<td></td>
</tr>
<tr>
<td>Decreasing High not at maximum</td>
<td>Decrease Mill Speed</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Decreasing OK</td>
<td>-</td>
<td>Increase Feed Rate</td>
<td></td>
</tr>
</tbody>
</table>

Pebble Crusher – Steel Removal Logic

If a pebble crusher is used in the SAG mill circuit, steel scats and other forms of tramp material must be removed to provide reliable crusher operation. The logic for the steel removal from the trommel oversize stream is described below:

- If the conveyor trips, a diversion gate directs the trommel oversize material to an emergency dump.
- Steel is removed by either a magnetic head pulley or an over belt magnet.
- A metal detector, before the crusher, indicates if any steel has passed through the collection system. If so, a diverter gate bypasses the pebble crusher and the feed is returned after a suitable time period has elapsed.
- The crusher diverter gate has a manual override, so that on-line maintenance can be performed on the crusher, without significant impact on mill downtime.

Sound Pattern Indication

A microphone and sound pressure transmitter are used for monitoring the sound pattern from the SAG mill. Initially this is used as an indication only, until a history can be established and
subsequent incorporation into the mill control logic is achieved.

**Supervisory Control**

Supervisory control developed to allow more advanced control methods to be implemented. An example of this includes the calculation of the circulating load around the ball mill and the adjustment set points to front end controllers. The supervisory system allows data collection and retrieval via electronic transfer so that the metallurgist can analyze plant data.

Included in the supervisory system is the single input single output controller with operator input set-point, as well as more intricate loops such as feed forward control, interlocking, sequence control, and drive control. Expert systems can reside in the supervisory system, but would normally form part of the high level control of the plant.

**High Level Control**

High level control algorithms include predictive models, expert systems, and optimizing control. This type of control acts on a much higher time scale than the stabilizing control; minutes and hours compared to milliseconds and seconds.

The use of models to predict the most effective control action has gained popularity as computing power has increased. These models use economic data and process variables to determine the optimum operating conditions for the plant. Tools of this nature are extremely powerful and can contribute significantly to the bottom line of the business.
Grinding Mills

### Rod Mill Dimensions & Weights

![Elevation View](image1)

![Plan View](image2)

**NOTE:** Drive can be located on either end of mill.

---

| I.D. Shell | I.D. Liners | Critical Speed | Net Compartment Length | C + B + F | D | E | F | G | H | J | K | L | M |
|------------|-------------|----------------|------------------------|----------|---|---|---|---|---|---|---|---|---|---|---|
| 10'-6"     | 9'-6"       | 24.9           | 12'-8" to 16'         | 3-0'     | 1-8" | 3-7" | 2-63/4" | 2-71/2" | 13-0" | 1-7" | 2-11 1/2" | 2-4" |
| 11'-6"     | 10'-6"      | 23.7           | 13'-16"               | 3-0'     | 1-10 1/2" | 3-7" | 2-61/2" | 3-7" | 13-0" | 1-7" | 2-11 1/2" | 5-4" |
| 12'-6"     | 11'-6"      | 22.6           | 14'-16"               | 3-0'     | 1-10 1/2" | 3-7" | 2-61/2" | 3-7" | 14-0" | 1-7" | 2-11 1/2" | 5-4" |
| 13'-6"     | 12'-6"      | 21.7           | 15'-18"               | 3-0'     | 1-10 1/2" | 3-7" | 2-6 1/2" | 3-7" | 14-0" | 1-7" | 2-11 1/2" | 5-4" |
| 14'-6"     | 13'-6"      | 21.7           | 16'-21"               | 3-0'     | 2-1" | 3-11" | 2-6 1/2" | 4-1" | 14-0" | 1-7" | 2-11 1/2" | 5-4" |
| 15'-6"     | 14'-6"      | 20.9           | 16'-19"               | 3-0'     | 2-10 1/2" | 3-7" | 2-6 1/2" | 3-7" | 15-0" | 1-7" | 2-11 1/2" | 5-5" |
| 14'-6"     | 13'-6"      | 20.9           | 19'-22"               | 3-0'     | 2-1" | 3-11" | 2-6 1/2" | 4-1" | 15-0" | 1-7" | 2-11 1/2" | 5-5" |
| 15'-6"     | 14'-6"      | 20.1           | 18'-21"               | 3-0'     | 2-10 1/2" | 3-7" | 2-6 1/2" | 4-1" | 15-0" | 1-7" | 2-11 1/2" | 6-0" |
| 15'-6"     | 14'-6"      | 20.1           | 21'-24"               | 3-0'     | 2-4" | 4-0 1/2" | 2-6 1/2" | 4-3" | 15-0" | 1-7" | 3-5 1/2" | 7-1" |
Determining Motor Horsepower Requirement

The power required to grind a material from a given feed size to a given product size can be estimated from the following equation:

\[ W = \frac{1.34}{\left( \frac{10Wi}{\sqrt{P}} \right) \left( \frac{10}{\sqrt{F}} \right)} \]

Where \( W \) = power consumption in hp-hrs/short ton for a wet-grinding, closed circuit operation with \( P \) greater than 70 micrometers. Multiplying \( W \) by the new feed rate (in STPH) will give the power requirement at the mill pinion shaft including bearing and gear losses. Motor losses and other drive component losses, such as reducer and clutches, are not included.

\( Wi \) = work index, which is the power in kwh/short ton required to reduce a material from theoretically infinite size to 80% passing 100 micrometers. Figure 1 is a table of overage work indices for various material.

\( P \) = size in micrometers of the screen opening which 80% of the circuit product will pass. See Figures 3A and 3B for typical open and closed circuit product analyses.

\( F \) = size in micrometers of the screen opening which 80% of the feed will pass. See figure 2 typical cone crusher product analyses.

In using the above equation, the following points must be considered:

A. The power consumption per short ton, \( W \), will only be correct for the specified size reduction when wet grinding in closed circuit. If the grinding circuit is changed then the power consumption also changes. Letting \( G = \) the gross hp-hrs. per short ton, the power consumption for other circuits becomes:

1. Wet grinding, open circuit, product topsize not limited: \( G = W \). (Most rod mill application fall into this category)

2. Wet grinding, open circuit, product Topsize limited: \( G = W \) to 1.25 \( W \).

3. Dry grinding, closed circuit: \( G = 1.30 \ W \).

4. Dry grinding, open circuit, product topsize not limited: \( G = 1.30 \ W \).

5. Dry grinding, open circuit, product topsize limited: \( G = 1.30 \ W \) to 2.0 \( W \).

6. If \( P \) is less than 80% passing 70 micrometers, power consumption:

\[ G = \frac{W (P + 10.3)}{1.145P} \]
Open circuit grinding to a given surface area requires no more power than closed circuit grinding to the same surface area provided there is no objection to the natural topsize. If topsize must be limited in open circuit, power requirements rise drastically as allowable topsize is reduced and particle size distribution tends toward the finer sizes.

B. The values of P and F must be based on materials having a natural particle size distribution.

C. The work index, Wi, should be obtained from plant data or test results, where the feed and product size distributions are as close as possible to those under study.

The most reliable work index values are those obtained from long term operating data. If this is not available, standard grindability tests can be run to provide approximate values.

The work index, Wi, will vary considerably for materials that appear to be very similar. The work index will also have a considerable variation across one ore body or deposit.

Rod and ball mill grindability test results should only be applied to their respective methods of grinding.

### Average Work Indices

<table>
<thead>
<tr>
<th>Material</th>
<th>Work Index Wi</th>
<th>Specific Gravity S</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bauxite</td>
<td>9.5</td>
<td>2.4</td>
</tr>
<tr>
<td>Cement Clinker</td>
<td>13.1</td>
<td>3.1</td>
</tr>
<tr>
<td>Coal</td>
<td>11.3</td>
<td>1.6</td>
</tr>
<tr>
<td>Coke</td>
<td>21.0</td>
<td>1.5</td>
</tr>
<tr>
<td>Coke Petroleum</td>
<td>74.5</td>
<td>1.7</td>
</tr>
<tr>
<td>Copper Ore</td>
<td>13.0</td>
<td>3.0</td>
</tr>
<tr>
<td>Flourspar</td>
<td>10.0</td>
<td>3.0</td>
</tr>
<tr>
<td>Gold Ore</td>
<td>14.8</td>
<td>2.9</td>
</tr>
<tr>
<td>Iron Ore</td>
<td>13.5</td>
<td>3.6</td>
</tr>
<tr>
<td>Hematite</td>
<td>12.9</td>
<td>3.8</td>
</tr>
<tr>
<td>Magnetite</td>
<td>11.5</td>
<td>3.9</td>
</tr>
<tr>
<td>Taconite</td>
<td>15.2</td>
<td>3.5</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Material</th>
<th>Work Index Wi</th>
<th>Specific Gravity S</th>
</tr>
</thead>
<tbody>
<tr>
<td>Lead Ore</td>
<td>11.3</td>
<td>3.4</td>
</tr>
<tr>
<td>Lead-Zinc Ore</td>
<td>11.4</td>
<td>3.4</td>
</tr>
<tr>
<td>Lignite (Coal)</td>
<td>13.4</td>
<td>1.4</td>
</tr>
<tr>
<td>Limestone</td>
<td>10.3</td>
<td>2.7</td>
</tr>
<tr>
<td>Molybdenum Ore</td>
<td>12.8</td>
<td>2.7</td>
</tr>
<tr>
<td>Nickel Ore</td>
<td>12.0</td>
<td>3.3</td>
</tr>
<tr>
<td>Phosphate Rock</td>
<td>10.5</td>
<td>2.7</td>
</tr>
<tr>
<td>Silver Ore</td>
<td>17.0</td>
<td>2.7</td>
</tr>
<tr>
<td>Tin Ore</td>
<td>11.2</td>
<td>3.9</td>
</tr>
<tr>
<td>Titanium Ore</td>
<td>11.9</td>
<td>4.2</td>
</tr>
<tr>
<td>Uranium Ore</td>
<td>17.0</td>
<td>2.7</td>
</tr>
<tr>
<td>Zinc Ore</td>
<td>12.4</td>
<td>3.7</td>
</tr>
</tbody>
</table>

NOTE: Average Work Indices should only be used for initial feasibility and estimates. Tests must be performed on representative samples of the material prior to final equipment selection as the work indices for a specific material have a wide variation.

Figure 1
Approximate Mill Feed Size
80% Passing From Nordberg Cone Crusher

<table>
<thead>
<tr>
<th>Topsize of Mill Feed Inches</th>
<th>Open Circuit Crushing 80% Passing Size Micrometers*</th>
<th>Closed Circuit Crushing 80% Passing Size Micrometers*</th>
</tr>
</thead>
<tbody>
<tr>
<td>2</td>
<td>32.300</td>
<td></td>
</tr>
<tr>
<td>1(\frac{3}{4})</td>
<td>28.400</td>
<td></td>
</tr>
<tr>
<td>1(\frac{1}{2})</td>
<td>24.000</td>
<td></td>
</tr>
<tr>
<td>1(\frac{1}{4})</td>
<td>20.200</td>
<td></td>
</tr>
<tr>
<td>1</td>
<td>16.300</td>
<td></td>
</tr>
<tr>
<td>7(\frac{8}{25})</td>
<td>14.250</td>
<td>12.200</td>
</tr>
<tr>
<td>3(\frac{1}{8})</td>
<td>12.200</td>
<td></td>
</tr>
<tr>
<td>5(\frac{1}{2})</td>
<td>10.150</td>
<td>10.150</td>
</tr>
<tr>
<td>3(\frac{1}{2})</td>
<td>8.100</td>
<td>8.100</td>
</tr>
<tr>
<td>4</td>
<td>6.100</td>
<td>3.800</td>
</tr>
<tr>
<td>6</td>
<td>3.300</td>
<td></td>
</tr>
<tr>
<td>6 Mesh</td>
<td></td>
<td>2.400</td>
</tr>
</tbody>
</table>

NOTE: These average table values will vary with the method of feeding, selection of crushe cavity, the weight, cleanliness and moisture content of the material and its fracture pattern. Accurate values should be established by actual testing.

* 1 Micrometer = 10\(^{-6}\) Meter
1 Inch = 25.400 Micrometers

Figure 2

Typical Grinding Mill Product Analysis
Open Circuit Operation

Figure 3B

These are considered average analyses and are based on a normal distribution. Variation can be expected due to varying material and operating characteristics.
Figure 4

Wet Grinding Ball Mill - Closed Circuit
Wet Grinding Rod Mill - Open or Closed Circuit
Horsepower per Ton related to WI and Particle Size
The graph in Figure 4 simplifies calculation of the power requirement once the work index is known. This graph is for ball mills wet grinding in closed circuit, and rod mills wet grinding in either closed or open circuit.

**Example 1**

A wet grinding ball mill in closed circuit is to be fed 100 STPH of a material with a work index of 15 and a size distribution of 80% passing ¼ inch. The required product size distribution is to be 80% passing 100 mesh. The power requirement is determined as follows:

1. On Figure 4 read the value of work expended in going from infinite size to 80% passing ¼ inch on the 15 work index plot (2.50 hp-hrs/short ton).
2. Read the value of the work expended in going from infinite size to 80% passing 100 mesh (16.5 hp-hrs/short ton).
3. Subtract the work already done on the feed from the total work required to produce the product (16.5 – 2.5 = 14.0 hp-hrs/short ton).
4. Multiply the new feed rate in STPH by the work per short ton (100 x 14.0 = 1400 hp).

**Example 2**

A dry grinding ball mill in closed circuit is to be fed 90 STPH of a material with a work index of the 13 and a feed topsize of ½ inch. The feed was prepared in a cone crusher. The required product size is 100% passing 200 mesh. The power requirement is determined as follows:

1. From Figure 3 we can estimate that 80% of the feed passes 8,100 micrometers.
2. From Figure 2B we can estimate that 80% of the product passes 52 micrometers.
3. The power consumption W is calculated from the formula:

\[ W = 1.34 \left( \frac{(10)(13) - (10)(13)}{\sqrt{52} \sqrt{8100}} \right) \]

4. Since W is for wet grinding in closed circuit with P greater than 70 micrometers the gross power consumption G is determined as follows:

\[ G = 22.23(1.3) \left( \frac{52 + 10.3}{1.145(52)} \right) \]

\[ G = 22.23(1.3)(1.046) = 30.23 \text{ hp - hrs/ST} \]

5. Multiply the new feed rate in STPH by G (30.23 x 90 = 2721 hp).

Note: Figure 4 could have been used to determine W. Also, if available F and P should be determined from screen analyses rather than Figures 2 and 3.

**Matching millsize to horsepower requirement**

The power requirement calculated above is the power that must be applied at the mill drive in order to grind the tonnage of feed from one size distribution to a finer size distribution. The following shows how the size of mill required to draw this power is calculated:

Figures 5 represents a section of a mill in operation. The power input required to maintain this condition is theoretically:

\[ Hp = \left( \frac{W}{33,000} \right)(C \ Sin a) 2m (N) \]

Where \( W \) = weight of charge

\( C \) = distance of center of gravity of charge from center of mill in feet

\( a \) = dynamic angle of repose of the charge

\( N \) = mill speed in rpm

If data from a similar installation is available, the value of the angle \( a \) can be found and the power demand of mills with various diameters at the same speed can be calculated.

The value of the angle \( a \) varies with the type of discharge, percent of critical speed, and grinding condition.
Thus direct comparison can only be made between mills with similar types of discharge. If various types of discharge are to be used, the following factors must be applied for mills of the same size and speed.

1. dry diaphragm = 1.0
2. wet diaphragm = 0.9
3. wet overflow = 0.8

In order to use the preceding equation, it is necessary to have considerable data on existing installations.

Therefore, this approach has been simplified as follows:

Five conditions determine the horsepower drawn by a mill –

1) Diameter
2) Length
3) % loading
4) Speed
5) Mill type

These conditions have been built into factors which are given in Figure 6. The approximate horsepower of a mill at the pinion shaft can be calculated from the following equation:

\[ HP = A \times B \times C \times L \]

Where:

A = factor for diameter inside the shell liners.

B = factor for mill type and charge volume (% loading)

C = factor for mill speed expressed as a percentage of mill critical speed.

L = length in feet of grinding chamber measured between head liners at the junction of the shell and head liners.

Some grinding mill manufacturers specify diameter inside shell liners, whereas Outokumpu grinding mills are specified by inside shell diameter. (In most cases subtract 6” to obtain diameter inside liners.) Likewise, a similar confusion surrounds the length of a mill. Therefore, when specifying mill dimensions it is extremely important to note where the dimensions are measured as there is no size convention for grinding mills.

The "B factors shown in Figure 6 are for steel grinding media. When other types of media or pebble milling are used, the "B factors must be modified in accordance with note 2.

**Example #3**

In example #1 it was determined that a 1400 hp wet grinding ball mill was required to grind 100 STPH of material with a work index of 15.0 from 80% passing ¼ inch to 80% passing 100 mesh in closed circuit.

What is the size of an overflow discharge ball mill for this application?

1. \[ HP = A \times B \times C \times L \]

2. The effective diameter of a mill is the diameter inside lining or net diameter (in most cases, diameter inside shell minus 6”). Since no diameter has been specified, investigate a range to find the most suitable. From Figure 6 the values of A are:

<table>
<thead>
<tr>
<th>Diameter inside shell</th>
<th>Net diameter</th>
<th>Factor A</th>
</tr>
</thead>
<tbody>
<tr>
<td>11’0”</td>
<td>10’6”</td>
<td>63.5</td>
</tr>
<tr>
<td>12’0”</td>
<td>11’6”</td>
<td>79.3</td>
</tr>
<tr>
<td>13’0”</td>
<td>12’6”</td>
<td>97.5</td>
</tr>
<tr>
<td>14’0”</td>
<td>13’6”</td>
<td>118.5</td>
</tr>
</tbody>
</table>

3. Most overflow discharge mills operate with a charge that occupies 35% to 45% of the mill volume. An average value would be 40%. The value of B from Figure 6 for a wet overflow ball mill is 5.02.

4. Speed was not specified, so a range of speeds should be investigated to find the most suitable. From Figure 6 the values of C are:

<table>
<thead>
<tr>
<th>Percent Critical Speed</th>
<th>Factor C</th>
</tr>
</thead>
<tbody>
<tr>
<td>68%</td>
<td>0.1583</td>
</tr>
<tr>
<td>72%</td>
<td>0.1724</td>
</tr>
<tr>
<td>76%</td>
<td>0.1878</td>
</tr>
</tbody>
</table>
5. Solving for the length of a mill required to draw 1400 hp with a 10’6” net diameter, 40% charge and 68% of critical speed, we have:

\[
L = \frac{1400}{63.5 \times 5.02 \times 1583} = 27.7 \text{ ft.}
\]

6. Find the length to diameter ratio for the above mill: \(L/D = 27.7 / 10.5 = 2.64 \text{ to } 1\)

Note that the grinding chamber length and net diameter are used to calculate the \(L/D\) ratio.

7. In a similar manner the following table can be developed for the various diameters and speeds being investigated:

8. Choose the particular combination of diameter and length which seems best suited to your application. The following points should be considered in making this selection:

   A) Length to diameter ratio – rod mill ratios normally fall between 1.2 to 1 and 1.6 to 1; ball mill ratios normally fall between 1 to 1 and 5 to 1.

   B) Speed selection – the slower the speed, the less wear on media and liners; the faster the speed, the lower the capital cost.

The mill sizes obtained from preceding calculations will be a **first approximation** only. Many other variables, such as viscosity, ore density, liner design, etc. can have an effect on the grinding efficiency and mill power draw. While the size you select will be accurate enough for preliminary planning, Outokumpu should be consulted before the final size selection is made. Outokumpu’s engineering staff will determine your requirements with the speed and precision which computer calculations make possible. There is no cost or obligation for this service.

<table>
<thead>
<tr>
<th>Dia. Inside Shell</th>
<th>Net Dia. D</th>
<th>68% CS</th>
<th>72% OCS</th>
<th>76% OCS</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>L</td>
<td>L/D</td>
<td>L</td>
<td>L/D</td>
</tr>
<tr>
<td>11’0”</td>
<td>10’6”</td>
<td>27.7’</td>
<td>25.5’</td>
<td>23.4’</td>
</tr>
<tr>
<td>12’0”</td>
<td>11’6”</td>
<td>22.2’</td>
<td>20.4’</td>
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</tr>
<tr>
<td>13’0”</td>
<td>12’6”</td>
<td>18.1’</td>
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<td>15.2’</td>
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<tr>
<td>14’0”</td>
<td>13’6”</td>
<td>14.9’</td>
<td>13.7’</td>
<td>12.5’</td>
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</tbody>
</table>
Figure 4

Figure 5
HP = A x B x C x L
Factors for Calculation of Rod and Ball Mill Horsepower

<table>
<thead>
<tr>
<th>Diameter Inside Shell Liners</th>
<th>Factor A</th>
<th>% Critical Speed</th>
<th>Factor C</th>
<th>Diameter inside Shell Liners</th>
<th>Factor A</th>
<th>% Critical Speed</th>
<th>Factor C</th>
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<tr>
<td>90&quot;</td>
<td>43.1</td>
<td>62</td>
<td>0.1400</td>
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<td>79.3</td>
<td>62</td>
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<td>0.1460</td>
<td>100&quot;</td>
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<td>64</td>
<td>0.1657</td>
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<td>106&quot;</td>
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<td>71</td>
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<td>100&quot;</td>
<td>79.3</td>
<td>82</td>
<td></td>
</tr>
<tr>
<td>196&quot;</td>
<td>297.8</td>
<td>83</td>
<td>0.2166</td>
<td>100&quot;</td>
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<td>83</td>
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<tr>
<td>200&quot;</td>
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<tr>
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<td>100&quot;</td>
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<td>87</td>
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**Mill Type and Loading**

<table>
<thead>
<tr>
<th>Factor B (2)</th>
<th>Ball Mills</th>
<th>Rod Mills</th>
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<tbody>
<tr>
<td>% Loading</td>
<td>Dry Diameter</td>
<td>Wet Diameter</td>
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<td>20</td>
<td>4.30</td>
<td>3.87</td>
</tr>
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<td>22</td>
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<td>26</td>
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<td>5.84</td>
</tr>
<tr>
<td>50</td>
<td>6.50</td>
<td>5.85</td>
</tr>
</tbody>
</table>

*Net diameter: in most cases diameter inside shell minus 6".*

*Based on steel grinding media. For other types of media and pebble milling, B factor must be adjusted by the ratio of the actual charge density, in lbs, per cubic foot, to 315 or B x charge density 315.

Figure 6
Selection of Optimum Size Grinding Media

In any given rod or ball mill application, sizing of the grinding media involves two major opposing factors:

1. As the size (diameter) of the grinding media **increases**, the pressure between surfaces increases, making it possible to break up larger particles.

2. As the size (diameter) of the grinding media **decreases**, the surface area available for grinding small particles increases, resulting in an increased grinding capacity.

**Table of Initial Charges**

<table>
<thead>
<tr>
<th>Rod Diameter (Inches)</th>
<th>Rod Size Distribution for Startup Charge (% Weight)</th>
<th>Rod Topsize in Inches (M)</th>
</tr>
</thead>
<tbody>
<tr>
<td>5</td>
<td>19</td>
<td>4½</td>
</tr>
<tr>
<td>4½</td>
<td>17</td>
<td>4</td>
</tr>
<tr>
<td>4</td>
<td>16</td>
<td>3½</td>
</tr>
<tr>
<td>3½</td>
<td>15</td>
<td>3</td>
</tr>
<tr>
<td>3</td>
<td>13</td>
<td>2½</td>
</tr>
<tr>
<td>2½</td>
<td>10</td>
<td>2</td>
</tr>
<tr>
<td>2</td>
<td>10</td>
<td>1½</td>
</tr>
</tbody>
</table>

**Figure 7A**

<table>
<thead>
<tr>
<th>Ball Diameter (Inches)</th>
<th>Ball Size Distribution for Startup Charge (% Weight)</th>
<th>Ball Topsize in Inches (M)</th>
</tr>
</thead>
<tbody>
<tr>
<td>5</td>
<td>5.0</td>
<td>4½</td>
</tr>
<tr>
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<td>4.0</td>
<td>4</td>
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<td>1½</td>
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<tr>
<td>1½</td>
<td>1½</td>
<td>1</td>
</tr>
</tbody>
</table>

**Figure 7B**

Several other secondary factors also enter into the selection of media size:

1. The harder the ore, the larger the media required to break a particle of given size.

2. The larger the mill diameter, the smaller the media required to break a particle of given size.

3. The faster the mill speed, the smaller the media required to break a particle of given size.

The above factors are all taken into consideration in the following equation, which can be used to calculate the maximum size media required in a given mill to grind a given material:

\[
M = \sqrt[3]{\frac{F \times Wi}{K \times C_s}} \times \sqrt{\frac{S}{D}}
\]

- \(M\) = diameter of topsize media in inches.
- \(F\) = size in micrometers of the screen opening which 80% of the new feed passes.
- \(Wi\) = Work Index
- \(C_s\) = percent of critical speed.
- \(S\) = specific gravity of feed.
- \(D\) = diameter of mill inside liners in feet.
- \(K\) = constant, the value of which is 200 for ball mills and 300 for rod mills.

The above equation gives the maximum size media required for a particular application. In the majority of applications, this also be the size of media added to the mill to make up for wear during operation. After a long period of operation, the size distribution of the media will be a complete range from maximum size down to very small size. This distribution of sizes is referred to as a seasoned or equilibrium charge and will be found to be the most efficient charge for nearly all applications.

In Order to produce a seasoned charge as soon as possible after a mill has been started up, the initial charge should be graded with respect to diameter. A list of graded charges for various topsizes is given in figure 7.

**Rods**

The rods used as grinding media are hot rolled high carbon or alloy steel. The rods are machine straightened and cut to prevent cracks from forming at the ends which can cause breakage in an operating mill. The quality selected should be sufficiently hard to give good wearing properties but not hard enough to cause excessive breakage. The sizes used vary from 5” down to 1½” diameter. The length of rod used in a given mill will be between 6” and 9” shorter than the...
distance between new head liners measured at the juncture of the head and shell liners. Charging of rods into the larger mills is done mechanically. Charging should be done on a regular basis to maintain the optimum power draw.

When new rods are placed in a mill, they will be in line contact and the void space will be approximately 22%. Thus new stacked rods weigh approximately 382 lbs./cu. ft. As previously stated, the rod charge becomes swollen with ore particles and the bulk density of the total charge can appear to be much lower if proper procedures are not followed. Broken rods also increase the void space and result in decreased efficiency. Figure 8 is a table of grinding rod data.

**Grinding Rod Data**

<table>
<thead>
<tr>
<th>Rod Size (Inches)</th>
<th>Wt./Ft. (Lbs.)</th>
<th>Ft./Ton*</th>
<th>Surface Area (sq.ft./ton)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1½</td>
<td>6.0</td>
<td>333.3</td>
<td>131</td>
</tr>
<tr>
<td>2</td>
<td>10.8</td>
<td>185.2</td>
<td>98</td>
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<tr>
<td>2½</td>
<td>16.8</td>
<td>119.1</td>
<td>78</td>
</tr>
<tr>
<td>3</td>
<td>24.0</td>
<td>83.3</td>
<td>65</td>
</tr>
<tr>
<td>3½</td>
<td>32.8</td>
<td>61.0</td>
<td>56</td>
</tr>
<tr>
<td>4</td>
<td>42.8</td>
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<td>49</td>
</tr>
<tr>
<td>5</td>
<td>67.0</td>
<td>29.9</td>
<td>39</td>
</tr>
</tbody>
</table>

*This column enables a quick calculation of the number of rods required. Assume 25 tons of 3½” diameter x 19’6” rods are required. No. of rods = 61 x 25 = 78 rods

Figure 8

When new balls are placed in a mill, they will be in point contact with one another and the void space will be approximately 280 lbs./cu. ft. Figure 9 is a table of grinding ball data.

**Grinding Ball Data**

<table>
<thead>
<tr>
<th>Ball Diameter (Inches)</th>
<th>Wt.Each (Lbs.)</th>
<th>No./Ton</th>
<th>Surface Area (sq.ft./ton)</th>
</tr>
</thead>
<tbody>
<tr>
<td>¾</td>
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<tr>
<td>10</td>
<td>18.544</td>
<td>108</td>
<td>58.8</td>
</tr>
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</table>

Figure 9

Consumption of media in wet grinding results from two major sources. The first is abrasion of the media surface by contact with the material being ground, with the liners and with other media. The second is by corrosion of the freshly exposed media surface. Of the two sources, the second appears to make the major contribution to media consumption. Consumption of media in dry grinding results only from abrasion and is therefore much lower than media consumption in wet grinding. In some cases, coating of the media by material in dry grinding reduces media consumption to very low levels. Figure 10 is a table of expected levels of media consumption.

**Consumption of Grinding Media**

<table>
<thead>
<tr>
<th>Media Type</th>
<th>DRY Range</th>
<th>DRY Average</th>
<th>WET Range</th>
<th>WET Average</th>
</tr>
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<tbody>
<tr>
<td>Rod Mills</td>
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<td>0.3</td>
<td>0.5 - 2.0</td>
<td>0.9</td>
</tr>
<tr>
<td>Primary Ball Mills</td>
<td>0.1 - 0.7</td>
<td>0.3</td>
<td>0.5 - 2.0</td>
<td>0.9</td>
</tr>
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<td>Secondary Ball Mills</td>
<td>0.1 - 0.5</td>
<td>0.15</td>
<td>0.3 - 1.5</td>
<td>0.8</td>
</tr>
</tbody>
</table>

Figure 10

Balls

The balls used as grinding media are usually of either forged steel or cast Ni-Hard. The choice of ball composition will depend upon the material to be ground, the ball size required and the proximity of suppliers. Balls should be as nearly spherical as possible with no excessive forging or casting ridges. Charging should be done as frequently as possible to maintain optimum power demand.

When new balls are placed in a mill, they will be in point contact with one another and the void space will be approximately 280 lbs./cu. ft. Figure 9 is a table of grinding ball data.

When a mill has been operated for a reasonable length of time and the charge has reached an equilibrium condition, the size distribution of the product and circulating loads should be examined to determine if a change in the media make-up size would improve efficiency.

**Pebbles**

If rock, flint or ceramic pebbles, etc., are to be used as grinding media, the media size used should be such that a rounded pebble would have the same weight as a steel ball that would be required for the same feed and mill conditions. In the case of rock pebbles, this gives a first approximation only, and rock pebble grinding should be thoroughly pilot tested for any given material. In most pebble milling operations there is no need for a graded initial charge as media consumption is relatively rapid, and equilibrium size distribution is achieved in a relatively short time.

The charge volume (level) of a grinding mill is usually expressed as a percentage of the volume within the liners that is filled with grinding media. When the mill is stationary, the charge volume can be obtained by measuring the diameter inside the liners and the distance from the top of the mill inside the liners to the top of the charge. Measurement should be made at several points
down the mill length to obtain the average charge volume. The percentage loading or charge volume can then be read off the graph in Figure 11, or can be approximated from the following equation:

\[
\% \text{ Loading} = 113 - \frac{126 \ H}{D}
\]

Where \( H \) is the distance from top of the mill inside the liners to top of charge and \( D \) is the mill diameter inside the liners.

Figure 11 can also be used to determine the volume of the charge. The area of charge can be determined from the curve \( A/D^2 \) or the following equation:

\[
A/D^2 = \frac{\% \text{ loading} \times \pi}{400}
\]

Multiplying the area by the length will give the cubic feet of mill volume occupied by the charge. The weight of the grinding media in the mill or required to charge the mill can easily be calculated from the bulk density of the media.

If the bulk density of the grinding media is to be determined experimentally, care should be taken to insure that the smallest dimension of the container is at least 25 times larger than the largest piece of media. Using too small a container will cause errors due to increased voids at the sides of the container.

A mill’s maximum power draw is obtained when the charge occupies 50% of the mill volume. However, when the charge volume is increased from 45% to 50% the power draw increase is approximately 1%. Therefore, in most cases, the increase in media consumption would be greater than the production increase. As a result, mills are seldom run with charge levels greater than 45%.

Normal design operation is from 35 to 42% charge level for ball mills.

In rod mills, the charge is swollen by particles of feed which separate the rods. If the mill is shut down immediately after the feed is shut off, the charge level will be greater than if the mill had been “ground out” prior to shutdown. Because of this rod mills are normally operated with 32% to 40% charge by volume. In operation, this becomes a 40% to 50% charge. However, the charge bulk density is considerably lower than that of stacked rods.

Ball mill charges become measurably swollen only when there is a buildup of large unground material in the ball mill or when the density of the pulp in a wet mill is extremely high. Although these conditions are seldom encountered, it is recommended that ball mills be ground out prior to shutdown for measurement of the charge level.
<table>
<thead>
<tr>
<th>Term</th>
<th>Description</th>
<th>Typical Units</th>
</tr>
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<tbody>
<tr>
<td><strong>Circuits</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>ABC</td>
<td>Autogenous primary mill and secondary ball mill with in-circuit pebble crushing</td>
<td></td>
</tr>
<tr>
<td>AC</td>
<td>Autogenous primary mill and pebble crusher</td>
<td></td>
</tr>
<tr>
<td>AG</td>
<td>Autogenous primary mill</td>
<td></td>
</tr>
<tr>
<td>LPC</td>
<td>Lump mill / Pebble mill / Crusher (for pebble) circuit</td>
<td></td>
</tr>
<tr>
<td>SAG-Ball</td>
<td>Series milling with a SAG primary mill and secondary ball mill. Can feature primary or secondary feed crushing.</td>
<td></td>
</tr>
<tr>
<td>SABC</td>
<td>As for SAG-Ball milling with a pebble crusher handling trommel / screen oversize</td>
<td></td>
</tr>
<tr>
<td>SAC</td>
<td>Single stage SAG mill with pebble crusher</td>
<td></td>
</tr>
<tr>
<td><strong>Indices</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ai</td>
<td>Bond Abrasion Index</td>
<td>g</td>
</tr>
<tr>
<td>BWI</td>
<td>Bond Ball Mill Work Index</td>
<td>kWh/t</td>
</tr>
<tr>
<td>RWI</td>
<td>Bond Rod Mill Work Index</td>
<td>kWh/t</td>
</tr>
<tr>
<td>IWI</td>
<td>Impact Work Index</td>
<td>kWh/t</td>
</tr>
<tr>
<td>CWI</td>
<td>Crushing Work Index</td>
<td>kWh/t</td>
</tr>
<tr>
<td>CLR</td>
<td>Circulating Load Ratio ${ \text{tph of cyclone underflow}/\text{tph of circuit product} } \times 100%$</td>
<td>%</td>
</tr>
<tr>
<td>d&lt;sub&gt;50&lt;/sub&gt;</td>
<td>Particle size at which there is equal chance of particle reporting to overflow or underflow from a classifier</td>
<td>μm</td>
</tr>
<tr>
<td>d&lt;sub&gt;60&lt;/sub&gt;</td>
<td>Particle size at which there is equal chance of particle reporting to overflow or underflow which has been corrected for separation with water</td>
<td>μm</td>
</tr>
<tr>
<td><strong>Power</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Specific Power</td>
<td>Net power per tonne milled consumed by an individual grinding unit.</td>
<td>kWh/t</td>
</tr>
<tr>
<td>Drive Efficiency</td>
<td>The fraction of the power which is delivered to the pinion compared to what is delivered to the motor. Nordberg assumes 2.5% loss through the motor and 5% loss to the pinion.</td>
<td>%</td>
</tr>
<tr>
<td><strong>Testing</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Autogenous Media Index</td>
<td>An index which relates the feed and product size with the power draw during the test to calculate a standard index.</td>
<td></td>
</tr>
<tr>
<td>UCS</td>
<td>Unconfined Compressive Strength, usually complies to ASTM 2938-86</td>
<td>MPa</td>
</tr>
<tr>
<td>AXC</td>
<td>UCS mode of failure - Axial Cleavage</td>
<td></td>
</tr>
<tr>
<td>CAT</td>
<td>UCS mode of failure - Cataclasis (Explosive Shattering)</td>
<td></td>
</tr>
<tr>
<td>SHR</td>
<td>UCS mode of failure - Shearing</td>
<td></td>
</tr>
<tr>
<td>Term</td>
<td>Description</td>
<td>Typical Units</td>
</tr>
<tr>
<td>--------------</td>
<td>-----------------------------------------------------------------------------</td>
<td>---------------</td>
</tr>
<tr>
<td><strong>Testing (continued)</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>JKMRC</td>
<td>Julius Kruttschnitt Mineral Research Center, Brisbane, Australia</td>
<td></td>
</tr>
<tr>
<td>MQ Core</td>
<td>Core diameter = approx. 100 mm</td>
<td>mm</td>
</tr>
<tr>
<td>PQ Core</td>
<td>Core diameter = 85 mm</td>
<td>mm</td>
</tr>
<tr>
<td>HQ Core</td>
<td>Core diameter = 63.5 mm</td>
<td>mm</td>
</tr>
<tr>
<td>NQ Core</td>
<td>Core diameter = 47.6 mm</td>
<td>mm</td>
</tr>
<tr>
<td>BQ Core</td>
<td>Core diameter = 36.4 mm</td>
<td>mm</td>
</tr>
<tr>
<td><strong>Terms</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>F_{50}</td>
<td>Measure of efficiency applied to the normalized Bond Power from 150 mm to 75 (\mu)m</td>
<td>mm</td>
</tr>
<tr>
<td>EF</td>
<td>Measure of ball/rod mill inefficiency (efficiency factors)</td>
<td></td>
</tr>
<tr>
<td>F_{80}</td>
<td>Feed 80% passing size</td>
<td>mm</td>
</tr>
<tr>
<td>T_{90}</td>
<td>Primary mill product 80% passing size</td>
<td>(\mu)m</td>
</tr>
<tr>
<td>P_{80}</td>
<td>Circuit product 80% passing size, usually cyclone overflow</td>
<td>(\mu)m</td>
</tr>
<tr>
<td>Aspect Ratio</td>
<td>Describes the L:D ratio of the mill. Low aspect is usually in the range 1.00 to 1.70, high aspect 0.3 to 0.6.</td>
<td></td>
</tr>
<tr>
<td>Critical Speed</td>
<td>Speed at which the charge centrifuges</td>
<td>Nc</td>
</tr>
<tr>
<td>EGL</td>
<td>Effective Grinding Length - inside liner mill chamber length at mill periphery</td>
<td>m</td>
</tr>
<tr>
<td>L:D Ratio</td>
<td>EGL length to inside shell diameter ratio</td>
<td></td>
</tr>
<tr>
<td>Mill Charge Angle</td>
<td>The angle of the mill charge toe as measured from vertical degrees</td>
<td></td>
</tr>
<tr>
<td><strong>Mill Lining</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>A:B Ratio</td>
<td>Ratio of distance between lifters to lifter height above liner</td>
<td>degrees</td>
</tr>
<tr>
<td>Angle of Impact</td>
<td>The angle from the vertical at which the media strikes the mill</td>
<td>degrees</td>
</tr>
<tr>
<td>Effective Liner Width</td>
<td>Distance between lifters calculated from mill diameter and number of rows of lifters</td>
<td>mm</td>
</tr>
<tr>
<td>Equilibrium Angle</td>
<td>The angle from the horizontal at which the media leaves the liner wall</td>
<td>degrees</td>
</tr>
<tr>
<td>Departure Angle</td>
<td>The angle from the horizontal at which the media leaves the lifter</td>
<td>degrees</td>
</tr>
<tr>
<td>% open area</td>
<td>This is the percent of the grate face which contains grates/ports, etc.</td>
<td>degrees</td>
</tr>
</tbody>
</table>
The above dimensions are approximations. Nordberg will furnish detailed dimensioned drawings for construction purposes.
DATA PARA SELECCIÓN DE MOLINOS

Compañía: _____________________________________________
Dirección: _____________________________________________
Teléfono: _____________________________________________
Fax: _____________________________________________
E-Mail: _____________________________________________
Nombre de la Persona de Contacto: ____________________________
Cargo: _____________________________________________
Fecha: _____________________________________________

Sirvase llenar en la máxima extensión posible el siguiente cuestionario y dejar en blanco los datos que no se encuentren disponibles

Si ya existe un molino en operación en una etapa de molienda, similar al del nuevo molino a ser instalado en un futuro, preste especial atención a las preguntas No. 23 y siguientes.

1. Material a ser procesado: ____________________________________
2. Características del Material: ____________________________________
3. % Humedad en la Alimentación: _______________________________
4. Work Index (Wi): _____________________________________________
5. Gravedad Específica de la Roca: _______________________________
6. Densidad Aparente (Bulk) de la alimentación: ____________________
7. Capacidad (ton solidas / h): _______________________________
8. Tamaño de Alimentación (Max.size / F80): _______________________
9. Tamaño del Producto (P80): _______________________________
10. Tipo de Circuito: Húmedo __ Seco ___ Abierto ___ Cerrado ___
11. Tipo de descarga necesaria: __ Overflow, __ Malla en Trommel __ Grate
12. Localización de la planta: ______________________________________
13. Altitud (msnm): _____________________________________________
14. Temperatura Ambiente: _________ (F / C)
15. Velocidad Crítica: __________________________
16. Potencia (hp/kW): __________________________
17. Tipo y Material de revestimiento preferido: _______________________
19. Instalación: Bajo Techo ____ Al aire libre____
20. Transmisión Preferida __________________________(Acople, Embrague, motor de baja/alta velocidad, etc.)
21. Sistema Eléctrico: ________ Volts, _____ Hertz, _______ Phase
22. Existe ya un molino en operación? (Si/No) : _______

Si ya existe un molino, favor especifique los siguientes datos:

23. Molino existente : __________________
24. Dimensiones del molino : ________________
25. Tipo de Molino : ____________
26. Tipo de descarga : ___________
27. Potencia del motor instalado : __________
28. Tipo de circuito: ___ Abierto, ___ Cerrado, ___ En Seco, ___ En Húmedo
29. Indique para el molino existente Tamaño de Alimentación (Max. size / F80):-
   ________________
30. Indique para el molino existente Tamaño de Producto (P80) : __________
31. Indique para el molino existente la potencia consumida y/o potencia consumida por
   ton: __________

Fin del cuestionario

Sirvase devolver este cuestionario a Outokumpu Técnica Perú SAC
FAX: (51 1 ) 221 2633

Gracias por su tiempo