Review of Existing Eco-efficient Comminution Devices

N. W. Johnson

Abstract

In achieving the necessary liberation for separation of valuable minerals from ores, metallurgists have traditionally selected from the competing options for size reduction devices by making decisions based on various types of costs. While these costs remain important in selection of devices, consideration of additional factors which incorporate the eco-efficiency of the devices provides a somewhat changed basis for selection.

This review examines the broad reasons for different technical efficiencies for various types of size reduction devices. In particular, size reduction devices with confined bed breakage, with stirred mills of various rotational speeds and therefore various energy intensities, and other mills such as the Hicom Nutating Mill and the Kelsey Fine Autogenous Grinding Mill are discussed in terms of their operating principles and their implications for the eco-efficiency of the device. The review also considers some additional attributes of each device which are relevant from the viewpoint of eco-efficiency.

From the viewpoint of eco-efficiency, compact size reduction devices (lowered energy consumption in manufacture) that have high throughputs and lowered energy usage per unit of feed are expected to be linked together to provide improved size reduction systems. The need for grinding media and an external classification system, both of which represent consumers of energy, for each device is discussed. It was observed that the useful ranges of some of the size reduction equipment are yet to be determined, such as the upper limits for the high intensity mills.

Some grinding systems can provide a given level of liberation at a coarser product sizing than for other systems, sometimes in conjunction with producing a desirable, more compact size distribution. Such outcomes may arise from improved matching of the size reduction mechanism(s) with the texture of the ore, or from a contribution from non random breakage leading to one or more types of selective liberation. Alternative systems need to be compared from the viewpoint of the overall liberation level achieved for a given product sizing and the contributing mechanisms (if any) to selective liberation.

From the discussion, a number of the size reduction devices provided additional benefits during the size reduction step or downstream from the size reduction step. Examples are the absence of detrimental effects from media (inert or not required), the provision of shear in the size reduction step (potential surface cleaning of adhering particles and colloidal deposits), lowered resistance to breakage in downstream size reduction steps and some mechanical activation of the minerals of relevance in downstream steps. There was also flexibility with many of the devices for both wet and dry modes of operation.

INTRODUCTION

Over the last two decades there has been an increase in the range of size reduction equipment that personnel in the mineral industry can consider for the purpose of achieving the needed liberation values for new plants or for upgrades of existing plants. Because of the rapid change in the range, there is scope for new combinations of size reduction equipment. Further, the size range over which some equipment can be applied is open ended in some cases.

A further important aspect is the potential for including a more complete basis for comparison of options where the eco-efficiency of the equipment is also used in decision making. Traditionally, two candidates for a particular size reduction task would be compared on the basis of:

1. Capital cost
2. Installation cost
3. Operating cost
4. Maintenance cost
5. Availability

These factors are all very important and would remain important parts of any comparison. However, to consider the eco-efficiency of each item of equipment for comparison purposes, some aspects of the listed points would be examined differently and additional aspects would be considered. The following aspects can be noted:

1. Energy consumed in manufacturing the device and its associated equipment, and its energy efficiency in operation (including the associated equipment).
2. The application of point 1 to grinding media in particular.
3. The application of point 1 to an external classification system.
4. The liberation of the valuable mineral achieved at a given product sizing.
5. Reductions downstream in the resistance of the ore to further size reduction.
6. Improvements in other down-stream processing properties.

It must be noted that the chemical environment inside the size reduction device controls the preparation conditions for the minerals in an ore. These are important for processes such as froth flotation. These may be reducing or non reducing and the range of conditions that are possible in a particular size reduction device can be of importance. Traditionally, ball mills have used mild steel grinding media which produce reducing conditions in the ball mill but also result in adsorption of hydrophilic iron hydroxides on the surface of the ground minerals, due to loss of the media by corrosion.

It can be noted that removal of liberated gangue at coarse sizes and other procedures such as microwave pre-treatment can also affect the eco-efficiency of a size reduction system. However, aspects such as these are outside the scope of this review.

TECHNICAL ASPECTS OF SIZE REDUCTION

The purpose of size reduction in mineral processing is almost always to achieve levels of liberation for the suite of minerals which allow the required separation efficiencies to be reached. After discussion of the technical issues in size reduction, the connection between size reduction and the extent of liberation will be discussed again.

Fuerstenau et al. (1991) considered various scenarios for the comminution of brittle solids ie sulphide mineral systems. The authors stated that "optimal utilization of energy in the comminution of brittle solids, as measured for instance by the surface area produced per unit energy expended, is achieved when a single particle is broken slowly under pure compression (Schonert, 1967). The next most efficient method is by the compressive loading of a bed of particles in a piston-die arrangement. In this mode, comminution occurs primarily by very high localized interparticle stresses generated within the particle bed". The authors contrasted this scenario with the situation in conventional size reduction equipment such as a ball mill or other mills containing media, where a "carrier" is utilized for "transporting the energy to the solids". The authors indicated that "the energy consumption for a comparable degree of size reduction in the compression loading route is only one-half of that in ball milling (Schonert, 1988) and attributed this partly to inefficiencies associated with the use of media for supplying energy for size reduction. Relevant discussion on assessing and improving the energy efficiency of comminution processes can also be found in Fuerstenau and Vazquez-Favela (1997).

Some important recurring themes for discussion of each type of size reduction device are the magnitude of the stress levels reached and their distribution i.e. the extent to which these stress levels occur in a controlled, narrow range or occur in a wide range around the mean value. Tavares (2004) noted that "if the level of the applied stressing is excessively high, the particle will be broken finer than the required size" and that "if the level is too low, then insufficient stresses will be created within the particle during impact and the particles will not fracture, resulting in losses as heat". A narrow range of stresses is required at the appropriate magnitude and between these two extremes.

Another important theme is the extent of confinement of the particle to be broken i.e. the extent to which the particle can escape from the zone in which breakage may occur, thereby avoiding breakage. This aspect is linked to the frequency of successful breakage events.

In ball mill grinding of particles in a slurry via the impact breakage mechanism, stress is imparted to a particle as a result of a successful collision between a grinding ball with the particle, resulting in fracture of the particle into many fragments with a range of particle sizes. The fracturing of the particle is achieved through propagation of a system of cracks which affect the majority of the original particle. Such major fracturing events can only occur when a "successful" collision (ie physical contact and imparting of sufficient stress) has taken place.

A serious impediment to a successful collision (and a major fracture event) for quite small particles is the high probability of escape of the small particle from the collision zone for the particle and the grinding ball, due to the drag force of the water which must also leave the collision zone (Schonert, 1989). This escape of a particle from the expected collision site represents a major impediment to size reduction. In wet grinding of quite small particles in a ball mill, there is theoretically enough energy for a major fracturing event if a successful impact occurs (Schonert, 1989).

In short, in a normal wet ball mill environment seeking impact breakage, major fracturing events can occur. However, these major events occur very, very infrequently. As a result, a small amount of extra size reduction is achieved by an unacceptably high input of energy to the ball mill.

The strength of particles and therefore their resistance to size reduction increases as they become smaller. This arises because the probability of flaws in particles decreases as a they becomes smaller. At a coarse size, initial breakage events will utilize existing flaws and weakness. Such sites are less likely to exist in the fragments from several stages of breakage unless the size reduction device is designed to initiate more such flaws.

An inverse relationship has been demonstrated between the logarithm of the force per unit area for incipient fracture and the logarithm of particle diameter for several types of solid (Schonert, 1989; Schonert, 1986). For small particles as flaws are depleted, the observed strength approaches the high intrinsic strength of the solid. Hence, for size reduction of small particles, higher forces may be needed and repeated applications of the higher forces may be necessary for size reduction.

For very fine grinding, a suitable size reduction device requires two properties:

1. Suitable stress levels must be available.
2. Relatively high probabilities for successful application of the stress to the particles must exist.

The class of size reduction devices known as stirred mills provide forces which are 65% higher than a conventional grinding device (Schonert, 1989). More importantly, the relatively small media size in a stirred mill greatly increases the number of active, collision zones in a given volume of mill. The breakage mechanism in these zones is not one of major fracturing events as described earlier for the impact mechanism in a wet ball mill.

Instead, fine progeny fragments are produced through chipping of the surface of the particle at local points with application of a normal stress, resulting in rounding of the particles and on-going size reduction of each original particle. The result of each breakage event is essentially a slightly changed original particle and some progeny particles that are much smaller than the original particle. This pattern of breakage is much different from the impact breakage mechanism.

The breakage mechanism in a stirred mill can be described by the term “attrition”. The key properties of this type of breakage event provided in a stirred mill are:

1. The stress levels must provide chipping of progeny particles from the surface of the initial particle.
2. Very frequent size reduction events can be provided via frequent unavoidable collisions between the particles to be ground and a large number of quite small grinding media moving at high speed.

The key aspect of the attrition mechanism in a stirred mill in comparison to the impact mechanism in a ball mill is that a comparatively large number of size reduction events is provided, with each event being quite small in magnitude. This appears to be more successful than providing an occasionally successful, large size reduction event.

The chemical environment in a size reduction device can be reducing or non reducing. The need for either condition depends on the nature of the ore and the type of downstream processing. In many traditional wet tumbling mills, steel grinding media have been employed for imparting the energy to the particles to be reduced in size. The media corrodes in this environment and the ferrous ions introduced into the aqueous phase form ferrous/ferric hydroxide deposits which adsorb randomly on the particles, making downstream processing much more difficult, particularly when fine and very fine regrinding is needed to achieve liberation. The use of steel grinding media is presumed to have arisen because of its cheapness, rather than a plan to obtain reducing conditions. If reducing conditions are required, an inert media and a suitable reagent (not steel) may be employed with better technical outcomes. Hence, the type of grinding environment needs to be considered in discussing the eco-efficient grinding devices.

Random or non random size reduction can occur in a size reduction device. Random size reduction alone leads to improvements in liberation when the progeny particles are produced from regions where the breakage pattern dictates that the progeny are smaller than grain size of the mineral. Veasey and Wills (1991) reviewed methods for improving mineral liberation.

Breakage at the boundaries between the valuable mineral and other minerals is typically visualized when non random size breakage is considered to be leading to selective liberation. However, this is an over-simplification. King and Schneider (1998) and others have adopted the approach that any form of non random breakage can be viewed as contributing to selective liberation (see also King, (2001) and summary in Table 1). A thorough analysis is needed to understand properly the source of any selective liberation and photographs showing apparent fracture planes along grain boundaries may mislead.

Table 1: Six separate non random fracture mechanisms during fracture of particles containing multiple minerals (all references to grade are volumetric grade)

Contributing Mechanism to Selective Liberation	General Description	Additional Comments
1. Selective breakage	Unequal brittleness exists for the minerals.	The selection function (rate of breakage) depends on both initial size and grade
2. Differential breakage	Breakage pattern depends on the initial grade also	The breakage function depends on the initial grade and not just initial size
3. Preferential breakage	Arises from crack branch-ing occurring more freq-ently in one mineral	Average grade changes with with progeny size
4. Phase-boundary fracture	Cracks propagate along mineral boundaries (elevated likelihood)	Interfacial area decreases (measure by image analysis)
5. Liberation by detachment	Grains of mineral of interest loosely bonded to other mineral grains	Similarities to 4. Grains of mineral of interest can detach giving no loss in interfacial area. Particles exist at size of inherent grain dimensions. Described by Gaudin (1939)
6. Boundary-region fracture	Preferential fracture of highly stressed area near mineral boundary leads to fine composites	Produce more smaller particles from mineral boundary area and less liberation exists in these small particles than for coarse progeny from elsewhere

King (2001) makes a number of points concerning the mechanisms in the table:

1. "No convincing evidence of significant phase boundary fracture has been reported in the literature."
2. Liberation by detachment is a "fairly rare phenomenon".
3. "Only the first three of these six non-random fracture effects have been modeled successfully and the last three require considerably more research."

If selective liberation contributes to achieving the target level of liberation, it will be achieved at a somewhat coarser product sizing than by random breakage in general. Eco-

efficiency will be improved. It must be cautioned that phase boundary breakage does not lead to enhanced liberation under some circumstances (King, 1994).
However, the seeking of enhanced liberation at a given product sizing by matching ore and machine properties is consistent with the objectives of eco-efficiency.

Some types of size reduction devices can produce microcracking in a given stage of size reduction. If this occurs, it can be expected that the following size reduction device will experience a lowered resistance to breakage. This outcome may also be consistent with an eco-efficient process. The weakening can be observed by grinding narrow size fractions of the product of interest in a laboratory ball mill or by testing the strength of individual particles in the product as used by Tavares (2005). By use of an impact load cell (a drop weight apparatus), the minimum amount of energy to fracture each particle can be determined. The particle stiffness can also be determined, where the particle stiffness reflects the internal damage (effect on the material microstructure).

Tavares (2004) described a method for decoupling the influences of differing product size distributions and different minimum fracture energies for the downstream size reduction of the products of different size reduction equipment.

In the following sections, the nature of various comminution devices will be discussed based on their general ability to contribute to eco-efficient operations.

HIGH PRESSURE GRINDING ROLLS

Background

It was noted earlier that size reduction devices which do not rely on media for transporting the energy possess advantages in efficiency of energy utilization. For size reduction of many particles at once by compression of the particle bed, advantages have been demonstrated at the small scale by the use of a piston-die arrangement operated in the batch mode in compression via the work of Prof. Schonert. The essential features of this equipment operated in batch mode have been retained in related equipment which can be operated in the continuous mode and at high throughputs. Prof. Schonert developed the key requirements for such a device (high pressure grinding rolls or high pressure rolls crusher) and these were patented by him. A practical manifestation of these requirements is a device with a set of high compression and counter-rotating cylindrical rolls with particles introduced at the top and with a slab or flake of product emerging from the rolls where the product requires deagglomeration.

Under the terms of the funding under which the research was conducted, Prof. Schonert was obliged to offer German industry the opportunity for development and manufacture of the actual size reduction device. Krupp-Polysius and KHD became the two companies who would potentially supply large throughput devices to the minerals industry. Later, Alpine also became a company supplying the device, limited to a different market (low throughput devices suitable for high value solids).

Fuerstenau et al., 1991 noted that “the choke-fed, pressurized roll mill invented by Schonert (1985) is perceived as a major breakthrough in improving the productivity and energy efficiency of the size reduction process from coarse (several millimeters) size feeds down to trans- and sub-sieve size products.” Otte (1988) provided an overview of the technology.

General Description

The high pressure grinding roll (also known as a roller press or high pressure roll crusher) is a completely different size reduction device from roll crushers which have been used in the mineral industry for over 100 years. Superficially, there is a resemblance in the layout and general appearance of roll crushers and high pressure grinding rolls. To review the high pressure grinding roll technology, its properties are initially compared and contrasted with the comparatively well known roll crusher technology.

Table 2: Comparison of roll crusher and high pressure grinding roll technologies

Characterisitic	Roll Crusher Technology	High Pressure Grinding Roll or High Pressure Roll Crusher Technology
Throughput	low	high
Roll speed	3 to 6 m/sec.	1.4 to 1.75 m/sec.
Length/diameter ratio of roll	1 to 2	0.18 to 0.6
Operational density of feed	0.25 to 0.6 t/m^3	2.3 to 2.5 t/m^3
Gap between rolls	fixed	adjustable
Grinding force	fixed	adjustable (1)

(1) = one roll fixed to mainframe and with one "floating" roll positioned by hydraulic rams with a specific grinding or pressing force

In a traditional rolls crusher, the feed particles are introduced essentially as individual particles into the fixed gap between two driven and rotating, relatively low diameter rolls. In contrast, in a high pressure grinding roll, the particles are introduced into the gap between two more slowly counter-rotating, larger diameter rolls, forming a bed of particles (see difference in operating density in table 2) whose thickness is determined by the gap between the rolls. The comminution occurs solely or largely within the bed of material as a result of the grinding force applied via a mechanism on one roll and which is transmitted via the bed. When the comminution does not occur entirely within the bed, effectively single particle breakage occurs before the particles are drawn into the compressed bed where the majority of the size reduction occurs through the mechanism of inter-particle crushing. In contrast, a rolls crusher utilizes single particle breakage.

As result, despite the high rotational speed of the rolls crusher, the throughput is much lower than for the high pressure rolls crusher because the density of the solid between the rolls is much lower. High pressure grinding rolls produce in effect a "slab" of material, containing the individual particles from size reduction which "cling" to each other as a result of the grinding force applied in the technology. An essential step is the de-agglomeration of the individual "particles" in the "slab" which emerges beneath the rolls in the high pressure grinding roll.

The wear in rolls crushers is usually considered to be uneven across the rolls. In principle, more even wear may be possible with high pressure grinding rolls as the full width of the roll is intended to provide size reduction. In practice, this desirable outcome may be more difficult to achieve.

It is possible to visualize a cylinder containing particles without any force applied. This situation can be equated with the same feed particles entering the upper regions of a high pressure grinding roll with no applied grinding force.

It is then possible to imagine a neatly fitting piston being used to apply a downward force on the particles in the cylinder, at the same time compacting the bed and reducing the size the particles. The resulting compacted bed height can be equated to the thickness of the "slab" of material emerging from the gap between the rolls in the high pressure grinding roll technology.

For high pressure grinding rolls operating at increasing energy levels ie higher operating pressures or forces, it is observed that, beyond a critical energy input level, the breakage rates decrease markedly, until breakage ceases and the energy is mainly consumed in further compaction of the bed (Hanisch and Schubert, 1982; Schonert and Flugel, 1980). It was suggested by Tavares (2005) that another reported property of high pressure grinding rolls (the introduction of damage or micro-cracking) is likely to continue or even be accelerated under these conditions.

Maxton et al. (2003) noted that "the specific pressing force (i.e. total force divided by projected roller area) is typically in the range of 3-9 N/mm^2 depending on the application, while pressures of up to 1000 N/mm^2 have been measured within the operating gap.

The high pressure grinding roll technology was originally applied to the cement industry in Europe during the 1980's. In this industry, relatively soft materials are crushed. Subsequently, the technology has been applied to harder more resistant materials. More importantly, the technology has been progressively applied to more abrasive feeds including feeds from the mineral industry. There have been problems with some of these applications due to the abrasiveness of the materials causing wear rate problems with the surfaces of the high pressure grinding rolls.

In Australia, Argyle Diamonds has been the only user of high pressure grinding rolls (HPGR) up to 1993 (Maxton et al, 2003) within the mineral industry. These author reported "Argyle Diamonds introduced HPGR in a secondary/tertiary crushing duty during 1990 when RP1A01, the first of two Krupp Polysius roller presses (2.2 m in diameter by 1.0 m wide) was installed. The second vitually identical machine (RP1B01) was installed in 1994 to operate in parallel with RP1A01.

To date Argyle Diamonds is the only mine within Australia to utilize HPGR and the mine now has three roller presses in operation following the recent completion of the Recrush HPGR Project. The third roller press (RP3E01) employs relatively new technology - hard metal studded tyres - in a recrush (or quaternary crushing)".

The technology can be used for the equivalent of fine crushing, rod milling and ball milling. It can be used in "wet" and "dry" modes. The key aspect is that the material can form a cake in the compression zone and, if this is possible, size reduction to 50 μm is possible. This is most likely when the feed has a wide range of sizes. Further, the feed must not contain too much water. For "dry" feeds, this is not an issue. In contrast, if regrinding of a wet stream at 80% solid is considered, cake formation is unlikely. As a guide, the water content of the feed should not exceed 15% by volume if regrinding of a fine feed was contemplated. Povey (2005) discussed recently the use of the technology in a wet classification circuit.

The cake formed in the compression zone requires allowance for disintegration and a means must be allowed for this.

Patzelt et al. (2005) indicated that the technology is well established in the world diamond and iron ore industries. The outcomes from some major installations in other branches of the world mineral industry during 2006 will have an important bearing on attitudes towards this technology, with wear rates being the issue of greatest interest.

Details of Findings

The findings on high pressure grinding roll technology will be discussed under a number of headings:

1. Energy efficiency and applied force
2. Liberation/grain boundary breakage
3. Roll surface protection
4. Protection from tramp iron

Energy Efficiency and Applied Force

It is a common finding in the literature that this technology is more energy efficient than conventional equipment in reducing the size of a given feed to a particular product sizing. Schwarz and Seebach (2000) reported the technology required 30% to 50% of the energy of conventional ball mills for the same size reduction task on cement clinker. Maxton et al. (2003) stated that “the efficiency arises from the determinate and relatively uniform loading of the material” during compression between the rolls, in contrast to the additional inefficiencies in crushers and mills arising from the nature of the loading which is “random and highly variable”. While such comparisons appear straight-forward, there are complications in performing comparisons due to changes in the shape of the product size distributions. In general, the literature abounds with observations that high pressure grinding rolls prepare a product containing an elevated proportion of relatively fine particles due to the high inter-particle stresses ie a changed shape for the product size distribution.

Another aspect of the energy efficiency of systems containing high pressure grinding rolls is that the product from this technology displays less resistance to further size reduction. The literature indicates that the lowering of energy requirements in the subsequent stage is typically in the range 20% to 50%. Tavares (2005) reported the following findings:

1. Coarse particles in the feed are damaged and weakened preferentially. (Therefore the initial stages of subsequent size reduction will display the greatest energy savings.)

2. The weakening is greater at higher applied roll pressures.
3. The weakening is independent of the position of the coarse particles in the bed.

When the applied roll pressure is increased, it follows that the specific energy consumption for the high pressure grinding roll is increased for the feed being processed.

Norgate and Weller (1994) reported results on laboratory scale high pressure grinding rolls from some Australian ores. The authors reported that "the specific energy consumption of the rolls increases linearly as the applied grinding force increases, but this is not accompanied by a linear increase in the size reduction achieved. Instead the latter increases at a gradually reducing rate, which is shown to be best described by a power-law equation. It is also shown that increasing the applied grinding force reduces the Bond Work Index of the rolls product and hence the energy required for any subsequent comminution".

By comparison of the requirements for a single stage operated at higher pressures than for two stages at lower pressure in series for the same ore, Norgate and Weller (1994) found that two units in series consumed less total energy to achieve the same size reduction. However, because of the lower pressures employed for the series operation, less reduction in the downstream resistance to size reduction could be expected.

For closed circuit operation, Patzelt et al. (2005) indicated that the applied force "determines the circulating load and the energy consumption of the HPGR circuit, but has no influence on the product size distribution of a closed circuit". The authors also discussed the effect of applied force on throughput for hard and soft ores.

Maxton et al. (2003) indicated that this technology "was the only form of comminution known to Argyle Diamonds that could achieve the requisite reduction ratios whilst maintaining a relatively wide operating gap. The requirement for a wide operating gap is specific to the diamond processing industry to avoid the potential for diamond breakage during crushing."

Smit (2005) used copper grade recovery curves from triplicate laboratory batch tests to assess the effect of a range of applied forces and conventional size reduction technology. He demonstrated that a relatively low applied force gave the best position for the grade recovery curve. Further, the grade recovery curves for the range of applied pressures were all in a better position than for the conventional technology (ball mill).

Selective Liberation

Fandrich (1998) reviewed the available literature for confined bed breakage and concluded that conditions for assisting selective liberation may require operation in the appropriate range for energy inputs/stress rates. The appropriate supply of energy to cracks may promote fracture paths along weakened mineral boundaries rather than across the boundary. Fracture paths tend to be less influenced by microstructure at elevated stress rates.

Apling and Bwalya (1997) used release analysis as a measure of liberation, finding that the technology provided a given degree of liberation at a coarser product sizing. These authors then indicated that energy savings would result from this finding as less size reduction was needed to achieve the target of the process ie a given level of liberation.

Dunne et al., 1996 stressed the potential significance of the micro-cracking which occurs to an elevated extent at grain boundaries, thereby elevating liberation and/or solvent penetration for a given nominal product sizing. Von Michaelis (2005) summarized many instances for demonstration of improved behaviour of product (usually at small scale) from high pressure grinding rolls in the areas of heap leaching, bio-oxidation processes for sulphides, gold leaching and gravity separations. However, sometimes benefits are not seen for some ores indicating the ore specific nature of the benefits (Dunne et al., 1996).

For downstream leaching processes, Patzelt et al. (1995) stressed the importance of assessing individual size fractions of product from conventional size reduction and high pressure grinding rolls. For valid comparisons between methods, this overcomes the issue created by the increased content of fine particles often observed in the product from high pressure grinding rolls. Patzelt et al. (1995) reported higher gold recoveries from leaching of each size fraction when the feed was prepared by high pressure grinding rolls.

Clarke and Wills (1989) compared the separation possible for a cassiterite ore after size reduction in a rod mill and a high pressure grinding roll. A more compact size distribution was observed with the high pressure grinding roll and heavy liquid separation tests on the product indicated also an improved amenability for separation, indicative of an improvement in liberation.

Fandrich (1998) carefully assessed the relevant mechanisms for the selective liberation observed for a binary iron oxide ore, by determining the contribution of each of the six mechanisms listed in Table 1. The tests were conducted in a cylinder/piston device to provide highly controlled conditions for confined particle bed breakage, but conditions approximating those in a high pressure rolls crusher. He found that the two contributing mechanisms were preferential breakage and phase-boundary fracture for this combination of ore and conditions (mechanisms 3 and 4 in Table 1).

Concerning preferential breakage (mechanism 3) for the binary ore, the “iron oxide phase was breaking the more brittle silicate phase preferentially”, resulting in “greater proportions of silicate phase reporting to the finer size fractions”. Two known phenomena in confined bed breakage were used to explain the observed preferential breakage. This appeared to be a more important contributor than phase-boundary fracture (mechanism 4). The importance of such mechanisms should be determined for other ores in confined bed breakage.

Interfacial surface area measurements showed some net loss of particle interfacial area between feed and product, indicating phase-boundary fracture (mechanism 4) contributed. A slight dependency of phase-boundary fracture on specific energy absorption was observed.

Roll Surface Protection

An extremely important practical aspect for general acceptance of this technology in the main-stream mineral industry is the materials employed and the design of the surface of the rolls. In their standard original form in the cement industry, “tyres” which were designed to slide on to the rolls becoming the wear resistant surface were the normal practice. This technology was inadequate for relatively abrasive solid. After some experience with hard-facing of the rolls, an improved design provided a tungsten carbide studded roll surface which allowed the formation of an autogenous layer of the solid being processed. The solid being processed became wedged between the studs resulting in interaction between the solid to be ground and the wedged solid (of the same type), thereby lessening interaction between the feed solid and the metal, lowering its wear. The wear which occurred was to the surface of the studs as long as the autogenous layer was formed.

An important early trial of studded tyres was conducted in the crushing plant at Cyprus Sierrita starting 1995 in Arizona (Thompsen et al., 1996a and 1996b). The tests were discontinued after “the copper price dropped and expansion plans at Sierrita were scrapped” (von Michaelis, 2005).

Maxton et al. (2003) made two important points on the experience at Argyle Diamonds in Australia:

1. It is necessary to establish in test work that an autogenous layer of the solid being processed will form (and compact tightly) for a given feed and studded roll design.
2. There should be an increase in the specific throughput with a studded tyre as a result of increased “grip” at the nipping point, potentially allowing the treatment of a given throughput of a material in a smaller high pressure grinding roll.

Maxton et al. (2003) reported that a wear problem appeared mid-way through the life of the studded tyres, resulting from the absence of an autogenous layer over the last 15-20 mm at each end of the tyre. This weakness was addressed by the use of welded strips of hard facing but caused significant lowering of the availability. The authors also reported that the general wear pattern on the moveable roller was much “smoother” than on the fixed roller. The authors indicated that further development to address edge wear was needed for hard and abrasive ores.

The current status of developments in wear protection was described by Morley (2005). Gardula et al. (2005) described one commercially available wear protection system and a second commercial wear protection system was described by Maxton et al. (2005).

Protection from Tramp Iron

The accidental introduction of tramp iron can break some studs in the same region on the surface, preventing formation of the autogenous layer and causing premature failure of the equipment. Maxton et al. (2003) reported the use of three stages of metal detectors for prevention of tramp iron entering the compression zone in which the rolls were approximately 30mm apart in the machine being discussed.

STIRRED MILLS

Stirred mills which are in use in the mineral industry have a range of rotational speeds for the agitator. The highest speed version is the IsaMill and the lowest speed version is the Vertimill (closely related to the tower mill). These devices are discussed in following sections.

These mills employ the attrition mechanism for size reduction. The attrition mechanism is a minor mechanism in tumbling mills where the impact mechanism dominates for these mills operating under normal conditions and for typical feed and product sizes. If a tumbling mill is used with unusually low diameter media, the relative and absolute importance of the attrition mechanism may rise.

Experiments on ball mills have shown some advantages from the use of an additive which modifies the rheology at high percent solid values. Experiments have also been reported for stirred mills to which additives have been added. It must be noted that stirred mills of the higher speed types generally operate at percent solid values in the range 40% to 60% (by weight).

Kapur et al. (1996) found that addition of a suitable dispersant to the slurry can greatly lower or eliminate the yield stress of the slurry and permit operation at somewhat higher percent solid values than would be possible otherwise. This means that the amount of solid present in the grinding chamber is increased, enhancing the productivity or throughput of the device. Zheng et al. (1996) also reported efficiency benefits in stirred milling with small additions of polyacrylic acid.

For the higher energy stirred mills, there may be downstream effects due to mechanical (or mechanochemical) activation of sulphide minerals. This may assist some downstream processes eg leaching.

The stirred mills with high shear rates may also provide a means for removal of adhering particles or colloidal deposits. For example, the Metso Detritor has tip speeds in the region of 10m/sec. while those in the IsaMill approach 20m/sec.

The IsaMill

The IsaMill is a large horizontal high speed stirred mill designed originally for high throughputs, fine product sizings and efficient energy utilization. The first model used in industry had a grinding chamber of 3000 litres with a 1100 kW motor. A second larger model being used in industry has a grinding chamber of 10,000 litres with a 2600kW motor.

The need for development of the IsaMill was the inception of a new phase of process development for the McArthur River deposit starting in 1989, a zinc-lead-silver ore-body which was difficult to process because of the fine grain size of its minerals. It was demonstrated in 1990 at laboratory scale that regrinding to 80% passing 7µm was necessary to achieve a concentrate of sufficiently high quality for sale.

In 1990, the conventional regrinding options in the base metal industry (ball mills and tower mills) were evaluated for the regrinding duty. Although these mills handle high throughputs, the energy consumption per tonne was unacceptably high for such fine product sizings. It was found that the energy consumption per tonne was acceptable for high speed horizontal stirred mills which were used in other industries at low throughputs for grinding to the region of 1µm, sometimes in the batch mode.

The laboratory results in 1990 were confirmed in pilot plant scale tests in 1992 by trials of a Netzsch horizontal stirred mill (100 litre grinding chamber) which was over-sized for the task. At the time, no large industrial scale mill of the general type used in the pilot plant scale testing existed in the market place for use in a future McArthur River operation. From the pilot plant experience, it was recognized that the necessary mill would have to be modified heavily to be suitable for continuous operation in the minerals industry under plant operating conditions. A large mill was needed to minimize the number of units operating in parallel, noting that the largest Netzsch mill had a grinding chamber of 500 litres.

Hence, various prototypes and a final design for such a large mill (3000 litres) were developed and evaluated on suitable streams in the existing zinc-lead-silver plant at the Mount Isa site of the same company. During this development work between 1992 and 1994, the technology provided metallurgical benefits from regrinding of key streams in the existing plant and provided an avenue for further development of that circuit during that period and also afterwards. Hence, four IsaMills were able to be installed for the start-up of the McArthur River plant in 1995 while two continued in operation in the Mount Isa plant at that time. While the Mount Isa ore was fine grained, it was not as fine as the McArthur River ore. However, a portion of each unit of feed to the plant at Mount Isa did require unusually fine regrinding to maximize liberation of the valuable minerals and therefore maximize their recovery.

The first publications of the this work were for the McArthur River ore (Enderle et al., 1997) and for the Mount Isa ore (Johnson et al., 1998). There are many later publications on this mill, of which two are listed (Gao et al., 2002 and Pease et al., 2006).

Various types of grinding media (sand, slag, heavy medium plant reject and a size fraction screened from a semi-autogenous grinding circuit product) were able to be used. The media is typically in the range 2 to 4 mm in diameter. Gao et al. (2000) compared the properties of two media, copper reverbatory furnace slag (CRFS) (sg of 3.7) and heavy medium plant rejects (HMPR) (sg of 2.6). The lower density type was more energy efficient than the higher density type but produced less quantity of finished product per unit of time. The energy efficiency was also dependent on the size of the media for both types. The grinding media is added with the feed slurry to the mill.

One area requiring heavy development was the grinding media retention system inside the mill. A system utilizing only hydraulic classification in a centrifugal field was developed, eliminating the use of internal screens of any type. A second area was the matching of materials to the ore properties to minimize wear of internal components.

The following paragraph is reproduced from Enderle et al., 1997 to provide some key findings. "As part of the initial work on the McArthur River ore, the mechanism of grinding in the batch laboratory stirred mill was checked by using fundamental testing methods (short tests on individual fractions). These showed that, as expected, the attrition mechanism was dominant (Andreatidis, 1995). Part of the evidence was provided by analysis of the shape of the fragments of various tests (Scarlett, 1990). Further, it was shown that the use of the stirred mill for the complex textures of the McArthur River ore produced, for some minerals, an increase in the level of liberation for a given nominal product sizing in comparison to normal size reduction methods (Andreatidis, 1995). For some minerals in the McArthur River ore, a higher level of liberation in the 5 to 10 micron fraction was observed for stirred milling compared with "normal" regrinding at the laboratory scale (Andreatidis, 1995 and Enderle et al., 1997). For one mineral, this higher level of liberation was associated with a greater proportion existing in the 5 to 10 micron fraction for a nominal product sizing and both factors therefore increased the overall level of liberation (Andreatidis, 1995). Such benefits were not observed for regrinding another ore (not from McArthur River) with less complex textures".

Some mechanical activation (increased defect density detected in the mineral crystal lattice) of minerals prior to leaching in the recently developed Albion Process which employs IsaMill technology and leaching at atmospheric pressure is believed to be a relevant factor in this process (Hourn, 2006). The ability of the IsaMill to grind to very fine product sizes is also relevant in the Albion Process. For example, copper minerals such as chalcopyrite with a high sulphur to copper ratio require size reduction prior to this process to approximately 12µm to ensure a passivating layer of sulphur is not created from leaching of copper ions, preventing further reaction.

Metso Stirred Media Detritor

The history of the Metso Detritor was covered by Smith (1999) and Burgess et al. (2001). These authors along with Davey (2002) discussed its initial applications in concentrators processing sulphide minerals for which its first application was in the Elura zinc-lead-silver concentrator in 1998 (New South Wales, Australia). This experience also served as a trial for an installation within the same company at the new Century Zinc concentrator (Queensland, Australia) where three were employed for a “normal” regrinding duty, and where fifteen were installed for an ultra fine regrinding duty where the target product sizing was between 6 and 7μm (80% passing size). This new plant was commissioned during 1999. A Metso Detritor was also employed at the Thalanga copper operation (Queensland, Australia) in a regrinding duty (indicative feed and product sizings of 60 and 40μm) at the end of 2000.

The Metso Detritor is classed as a vertical stirred mill with a tip speed for the impeller arms of approximately 10 m/sec (Burgess et al., 2001). These authors indicated that the stirring action resulted from “two sets of impeller arms, six arms to a set, positioned in the lower portion of the detritor”. The largest unit available is rated at 355kW, with a smaller unit rated at 185kW. The grinding chamber is octagonal in cross section and can be fitted with discharge screens at the top of the chamber or beneath the chamber.

Overall, Burgess et al. (2001) preferred the discharge screens (apertures typically 300 μm) to be located on the top of the chamber. It was commented that failure of a discharge screen was observed via an increase in grinding media loss/consumption. With the discharge screen in sound condition, its opening determined the size at which worn grinding media was able to leave the grinding chamber.

Smith (1999) indicated that the most efficient media diameter will generally be within the range 0.5-3.0 mm and that the optimal size can be determined by test work. Burgess et al. (2001) reported the use of “round” beach sand, between 1 and 2 mm in diameter, at Elura. Smith (1999) noted a media consumption rate at Elura of approximately 0.5 kg/t and that its delivered cost was 10% of that for conventional, manufactured grinding media. For the media at Elura, Burgess et al. (2001) indicated the importance of the particle hardness and shape, in addition to its diameter. It was indicated that most grinding media tested achieved the desired grinding performance. However, other issues were important as coarse media caused excessive wear to impeller arms, discharge screens and liners, while incompetent media and small media gave high losses through the discharge screens.

The feed to the detritor had typically a pulp density of 1.56 to 1.66 depending on the application. The detritor was operated in open circuit in the described cases. In some instances, the feed was cycloned and the detritor processed the cyclone underflow, with the cyclone overflow joining the ground cyclone underflow on an open circuit basis. In other instances, the feed stream went directly to the detritor (without prior classification).

Smith (1999) commented that previous ultra fine grinding experience with industrial minerals appeared to be applicable to the high density minerals in sulphide ores.

Vertical Stirred Mills

The mills in this category have been in use for the longest period in the mineral industry. Tower mills were first used for grinding limestone in Japan between 1950 and 1960 and were first used in the mineral industry around 1980. Metso are an important supplier of this product known as the Vertimill. It is believed that Kubota in Japan continues to supply this type of mill. Other vertical mills of this general type include the products from ANI-Metprotech (originating from South Africa) and the Sala agitated mill (SAM), originating from and mainly used in Europe. Both of these products have been in the market place for at least 15 years. For the ANI-Metprotech and Sala mills, stirring devices on the vertical shaft are arms or pins, as opposed to the helical screw in a tower mill.

These mills represent the “mature” end of the stirred mill range that has been applied in the mineral industry. Most attention will be given to the Vertimill (or tower mill) because the majority of installations have been of this type. The feed to these mills would normally be smaller than 2mm, and often would be much smaller than this size.

Sepulveda and Fletcher (1984) described the charge motion in a tower mill imparted by the rotating vertical screw-shaped impeller. “The screw impeller rotates in such a way as to move the grinding media upward through the central portion of the grinding compartment and downward through the outer portion. Grinding media such as steel balls, ceramic pebbles, or natural pebbles, one inch in diameter or less, are charged in the grinding chamber covering the screw flight agitator. Material to be ground is fed into the top, centre, or bottom of the main body depending on the application and size of the feed. As in other stirred ball mills, grinding occurs by a combination of impingement and rubbing among the grinding balls and grinding balls against screw and mill liners”.

Other authors (Napier-Munn et al., 1996) have commented for these mills that “there is no free-fall motion of grinding media and hence impact breakage does not occur. The movement of the stirrer through the ball bed, and the sliding/rolling motion that this imparts to the charge itself, provides a solely attrition breakage environment”.

When tower mills were introduced to the mineral industry, there was a tendency for the grinding balls to be too large for the duty, decreasing the efficiency of energy usage, and an important issue was also the wear rate of the leading edge of the rotating helical screw and down time associated with the resulting periodic maintenance.

Many installed tower mills retained an internal “coarse” classifier which was a feature used in limestone grinding, their initial industrial application. In wet grinding, as slurry over-flowed the top of the grinding chamber, the coarsest particles which could be transported upwards by the slurry were removed in the “coarse” classifier and returned to the mill for regrinding. External cyclones are also employed for closed circuit classification in the wet systems typically used in the minerals industry. Dry systems are also employed in other industries.

The tower mills were employed initially in wet regrinding applications where energy savings in comparison to wet ball mills were typically in the range of 30% to 50%. Tower mills have also been installed as tertiary grinding devices. Examples are the Red Dog (Kral, 1992) and Hellyer (Richmond, 1993) concentrators in which these mills followed a SAG mill-ball mill sequence. The incentives for this use of tower mills was again energy efficiencies in comparison to ball mills and a smaller "footprint" leading to lowered capital and installation costs.

The largest tower mills have motors of 1000kW and this size has been the largest available for some time. With tower mills, their specific energy consumptions increased when product sizes (80% passing) for sulphide mineral middlings below 10 to 15µm were sought using normal conditions for testing these ores in this type of mill. This outcome resulted in the testing and adoption of other more intense types of stirred mills for applications with products in this range. An important issue is the relative merits of tower mills and these other higher speed mills for product sizings the "normal" regrinding range.

HICOM NUTATING MILL

The history of the Hicom mill has been described by Hoyer and Morgan (1996) who described its grinding chamber as a "truncated cone with a rounded base, open at the top and with a grate discharge at the bottom". The truncated cone sits in essentially a vertical position with its smallest diameter at the top (the feed end) and its large diameter at the bottom. Further, the nutating action of the chamber was described as "similar to a swirling conical flask in the wrist: the top wobbles and the bottom describes a circle". The definition of a nutating action is a periodic oscillation superimposed on a circular motion. The Hicom prototypes and the first commercial mills were designed, built and tested by MD Research in Sydney, as part of the Warman Group activities.

The Hicom Nutating Mill's motion is conceptually similar to a centrifugal mill (Lloyd et al., 1982) but it is a more practical version of the centrifugal mill by overcoming the discharge difficulties possible with centrifugal mills. Unlike the centrifugal mill, the diameter and forces vary in the vertical axis and there is no critical speed. Hoyer et al. (2001) indicated that "the mill load tumbles in a manner somewhat similar to a ball mill, though in a horizontal plane rather than a vertical plane".

The Hicom mill will provide size reduction into the region where the product has an 80% passing size between 2 and 10µm because of the very high forces provided in the mill, for both soft and hard materials including metalliferous ores.

The grinding action results from the combination of the grinding chamber geometry and the previously described high speed nutating motion. This results in the mill contents being accelerated repeatedly in an acceleration field of the order of 50 times stronger than gravity. With a maximum power input of 110kW and a grinding chamber volume of 60 litres, the power input per volume of grinding chamber is the highest of the mills considered in this review to this point and is of the order of 1850 kW/m^3. A version has been built with the same power input (110kW) and a smaller chamber (Hoyer et al., 2001). The high forces and high energy intensity inside the mill result in high grinding rates, 50 to 100 tines higher than for a conventional tumbling mill such as a ball mill.

The means for feeding the mill and the discharge system were described by Hoyer et al., (2001). "The feed drops vertically into the grinding chamber and is ejected via eight discharge ports around the periphery. These ports may be open holes of around 40mm diameter, or they may be covered with slotted grates". The Hicom mill design has a stationary feed throat which allows a simple feed system eg from a conventional feed belt or equivalent (Hoyer and Morgan, 1996).

Some type of external pre-classification system may be required to direct the size reduction at the required particles. There is no classification system within the mill, noting that discharge can occur via a grate.

This type of mill can operate with a charge of steel balls or ceramic media. Alternatively, an autogenous charge of ore can be employed with feed sizes of the autogenous "media" up to 80mm being acceptable. The mill can be operated both wet and dry and in batch and continuous modes.

The mill is extremely compact with a small footprint and small foundations making it suitable for use where space is scarce eg underground, on a floating processing plant or in enclosed plants as a result of very cold climates.

The Hicom mill has been used extensively commercially for industrial minerals and, to a lesser extent, with diamondiferous materials. Energy savings in comparison to ball milling were 31% to 70% lower for calcium carbonate in a closed circuit dry grinding system (Braun et al., 2002) for a feed of 94µm being ground to a product between 26 and 6.4µm (all sizes expressed as 98% passing sizes).

By operation of the Hicom mill in the autogenous mode (no media), it has been demonstrated that it can be used for size reduction of critical size material (20 to 60 mm in diameter) from autogenous and semi-autogenous mills to a much finer size than via a crusher, with overall energy savings for the system (Hoyer, 1996). It is believed that there are presently no installations of this application in industry. With operation in the autogenous mode, it is being used commercially in the diamond industry in South Africa to selectively grind non valuable components (such as shells and other flat particles) in suitable streams without affecting the diamonds, thereby reducing the stream volume (after removal by screening of the selectively ground non valuable components) by up to 60% before further costly processing steps. For a diamond bearing kimberlite from the Merlin deposit in the Northern Territory, a 25kW Hicom pilot plant mill demonstrated liberation of diamonds from kimberlite in the presence of barren dolomite and chert, the latter two being rejected as large lumps via 40mm discharge ports and the kimberlite being largely reduced to minus 4mm with liberation of the diamonds without damage (Hoyer and Lee, 1997).

It has been demonstrated to act as an attritioning device for ilmenite from which alumina-silicate coatings were lowered (Hoyer et al., 2001). The high energies in the mill allow mechanical activation effects (formation of special alloys) (Hoyer and Morgan, 1996).

KELSEY FINE AUTOGENOUS GRINDING MILL

This technology was developed in Australia by the originator of the Kelsey Centrifugal Jig. Its development appears to have been based on a desire to apply centrifugal force to size reduction, using past experience from the jig development and a perceived need for more options in the spectrum from attritioning to size reduction. A description of the device has been prepared (Jones and Robinson, 2005).

A rotating cylindrical grinding chamber spinning around its vertical axis applies stress to a particle bed formed at the periphery of the grinding chamber from particles continuously fed into the device. The rotational speed and the diameter of the grinding chamber determine the force applied in compressing the particle bed.

The feed slurry or dry solid are fed downwards through a fixed vertical feed pipe. An impeller in the feed inlet engages and accelerates the feed into the compressed particle bed which, as mentioned previously, lines the periphery of the grinding chamber. Alternatively, it is also possible for the device to be fed from beneath.

The authors describe an “inter-particle energy transfer environment” in the bed and from recently introduced particles which are accelerated into the bed and which leads to “shear and abrasion fracture”. This grinding environment allows size reduction to proceed without grinding media (and the various systems associated with media) and allows size reduction of material to product sizes in the region from 5 to 15µm (80% passing sizes), and probably finer.

This new mill appears to be, in effect, a continuous, industrial version of laboratory material assessment devices which direct single particles at high velocity at a fixed target, for assessment of their resistance to size reduction under these conditions. The stresses are applied rapidly in such a laboratory device. In the Kelsey mill, each particle is directed at high velocity to a target which is produced from particles already introduced into the device and retained within the mill.

DISCUSSION

The real purpose of size reduction is usually to obtain the required level of liberation. While random breakage leads to liberation, non random breakage from fortuitous circumstances or from appropriate selection of size reduction equipment and its operating settings, along with knowledge of the texture for a given ore, may lead to the required level of liberation at a somewhat coarser product sizing than for the pure random breakage case. Such outcomes improve the eco-efficiency of the overall process.

It can be noted from Table 1 there are six mechanisms which contribute to selective liberation, three of which are “breakage” based and the other three are connected with unusual and non random types of fracture at or near mineral boundaries. It is not common for the liberation process (including the size reduction aspects) to be analyzed in terms of the contributions from all the six possible mechanisms.

There appears to be a lot to be gained from improving understanding of the high pressure grinding rolls in these terms for ores with a wide range of textures. Further, in the case of stirred mills which foster the attrition size reduction mechanism, understanding of the interaction of this breakage mechanism with ores having a wide range of textures is warranted.

A recent paper from South Africa (Steyn et al., 1995) showed that, for the ore textures in one platinum ore, it was beneficial to employ staged grinding with flotation stages after each grinding step. The grinding stages were ball milling, followed by a Metso Detritor, and followed by an IsaMill. For this analysis, emphasis was placed on maximization of recovery of the platinum group elements. This example shows the importance of selection of the most appropriate circuit type which must be matched to the ore texture. The authors commented that “ore types most likely to benefit from staged grinding and flotation circuits have valuable particle grain sizes that vary significantly, or where these grain sizes are very fine”.

It is well established that confined bed breakage as exemplified by the high pressure grinding roll provides efficiencies in energy utilization for feeds up to several millimeters in diameter and products as fine as the sub-sieve region ie the fine crushing, rod milling and ball milling regions. More experience may be needed to define the upper and lower boundaries of this machine which can be operated in both the wet and dry modes. This type of machine also provides energy reductions in down-stream size reduction. This technology appears to require some reduction in edge wear. If achieved, the remaining impediment to its greater use in the mineral industry should be eliminated.

It is clear that the stirred mills allow size reduction to occur with acceptably low energy inputs in the fine and very fine grinding regions. The extent to which these mills remain energy competitive in the normal regrinding and grinding regions is a topic of current trials. It can be said that media commensurate with the feed size will be required and that large reduction ratios should not be sought, in order to keep the media size commensurate with the size of the material to be ground.

In discussion of high pressure grinding rolls and their incorporation in comminution systems, Morley (2005) concluded that "properly designed HPGR-based circuits offer the potential of significant savings in comminution energy requirements and overall operating costs when compared to SAG-based circuits". He advocated that its application should be extended in the fine direction, "culminating in the generation of final ground product by air classification of HPGR product and the elimination of ball milling". It was noted that such circuits are used for other purposes (dry ground limestone for desulphurization) in Europe.

It is known in general that stress levels which have an appropriate mean value for the material being ground and a relatively small variance lead to less wastage of energy (value too low causing heat and other losses) and unnecessary production of small particles (value too high). Stirred mills utilize the attrition mechanism for size reduction in which frequent, small size reduction events are used, as opposed to infrequent, large size reduction events which dominate in a classical ball mill operation.

Inert media are often used in the stirred mills, providing a "clean" grinding environment. There appears to be a trade-off between the extent to which the media is manufactured or pre-prepared and the overall efficiency of size reduction in which eco-efficiency considerations exist. When the stirred mill is fully enclosed, the only entry point for oxygen under normal operation is dissolved oxygen in the pulp and these circumstances may provide an oxygen depleted pulp immediately after exiting the grinding chamber.

Stirred mills can be operated with a classification system (usually in open circuit). The IsaMill (Pease, 2006) does not necessarily require an external classification system. This can eliminate the need for a separate classification system and represents an aspect for inclusion in eco-efficiency comparisons. A recent paper suggests how Metso Detritors may be employed in series to decrease the need for external classification (Davey, 2006).

A recent paper (Nesset et al., 2006) has demonstrated that a range of small scale stirred and tumbling mills can be shown to have similar energy requirements if they are operated with appropriately sized, small grinding media. Such modes of operation for ball mills which presumably make attrition dominant have not normally been used. However, given the low energy intensity for such tumbling mills, it is likely that, on an eco-efficiency basis, the small physical size of the stirred mills would mean that such mills would be more eco-efficient, even for comparisons where the kWh/tonne values were similar for a given ore. It can be noted that the largest stirred mill in terms of installed power is the largest IsaMill (2600kW).

Stirred mills such as the IsaMill can provide sufficient energy levels to provide some extent of mechanical activation in downstream processing. This has been demonstrated for the higher energy input per unit volume Hicom Nutating Mill.

The Hicom Nutating Mill and the Kelsey Fine Autogenous Grinding Mill represent different classes of mills in which different compromises have been taken as discussed in the two preceding sections. The small physical size of these mills also may mean that eco-efficiency gains may be demonstrated with these mills by matching the properties of these devices with a range of materials and a range of tasks. A range of industrial applications for the Hicom Nutating Mill was discussed earlier. Its attributes need to be determined along the lines discussed in paragraph 3 of this section for other classes of mills. The Kelsey Fine Autogenous Mill which is at the beginning of its exposure to industry could benefit from a similar analysis which highlights their attributes.

SUMMARY

From the viewpoint of eco-efficiency, compact size reduction devices (lowered energy consumption in manufacture) that have high throughputs and lowered energy usage per unit of feed are expected to be linked together to provide improved size reduction systems. Examples of such equipment were reviewed, along with the need for grinding media and an external classification system, both of which represent consumers of energy.

The useful ranges of some of the size reduction equipment are yet to be determined, such as the upper limits for the high intensity stirred mills.

The achievement of the required liberation level is the ultimate objective of a size reduction system in most cases. Some grinding systems can provide a given level of liberation at a coarser product sizing than for other systems, sometimes in conjunction with producing a desirable, more compact size distribution. Such outcomes are very desirable from the viewpoint of eco-efficiency. They may arise from improved matching of the size reduction mechanism(s) with the texture of the ore, or from a contribution from non random breakage leading to one or more types of selective liberation. Alternative systems need to be compared from the viewpoint of the overall liberation level achieved for a given product sizing and the contributing mechanisms (if any) to selective liberation.

From the discussion, a number of the size reduction devices provided additional benefits during the size reduction step or downstream from the size reduction step. Examples during size reduction are the absence of iron hydroxide deposits derived from steel grinding media (no media used/inert media used) and the provision of shear in the size reduction step (potential surface cleaning of adhering particles and colloidal deposits). Downstream examples are lowered resistance to breakage in following size reduction steps and some mechanical activation of the minerals of relevance for the downstream steps.

There was also flexibility with many of the devices for both wet and dry modes of operation.

REFERENCES

Andreatidis, J.P. 1995. Breakage mechanisms and resulting mineral liberation in a bead mill, *M. Eng. Sc. Thesis,* Dept. of Min. and Metall. Eng., University of Queensland.

Apling, A. and Bwalya, M. 1997. Evaluating high pressure milling for liberation enhancement and energy saving, *Minerals Engineering,* Vol. 10, No. 9, 1013-1022.

Braun, R.M., Kolacz J. and Hoyer, D.I. 2002. Fine dry comminution of calcium carbonate in a Hicom mill with an Inprosys air classifier, *Minerals Engineering,* 15: 123-129.

Burgess, F., McGuire I. and Willoughby, R. 2001. Operation of Sand Mill Detritors at Pasminco Operations, *Fine Particles Processing and Tailings Summit,* 2-4 July, Perth, Australia.

Clarke, A.J. and Wills, B.A. 1989. Enhancement of cassiterite liberation by high pressure roller comminution, *Minerals Engineering,* Vol. 2, No. 2, 259-262.

Davey, G. 2002. Ultrafine and fine grinding using the METSO Stirred Media Detritor (SMD), *Proceedings 34th Annual Meeting of the Canadian Mineral Processors,* Canadian Institute of Mining Metallurgy and Petroleum, Montreal, 153-169.

Davey, G. 2006. Fine grinding applications using the Metso Vertimill grinding mill and the Metso Stirred Media Detritor (SMD) in gold processing, *Proceedings 38th Annual Meeting of the Canadian Mineral Processors,* Canadian Institute of Mining Metallurgy and Petroleum, Ottawa, 153-169.

Dunne, R., Goulsbra, A. and Dunlop. I. 1996. High pressure grinding rolls and the effect on liberation: Comparative test results, *Randol Gold Forum 96,* Olympic Valley, CA.

Enderle, U., Woodall, P., Duffy, M. and Johnson, N.W. 1997. Stirred mill technology for regrinding McArthur River and Mount Isa zinc/lead ores, *XX International Mineral Processing Congress,* Aachen, Germany, September 21-26, Vol. 2, 71-78.

Fandrich, R.G. 1998. Mineral liberation through confined particle bed breakage, PhD Thesis, The University of Queensland.

Fuerstenau, D.W., Shukla, A. and Kapur, P.C., 1991 Energy consumption and product size distributions in choke-fed, high-compression roll mills, *Int. J. Miner. Proc.,* 32, 59-79.

Fuerstenau, D.W. and Vazquez-Favela, J. 1997. On assessing and enhancing the energy efficiency of comminution processes, *Minerals and Metallurgical Processing,* Feb., 41-48.

Gao, M., Young, M. and Allum, P. 2002. IsaMill fine grinding technology and its industrial applications at Mount Isa Mines, *Proceedings 34th Annual Meeting of the Canadian Mineral Processors,* Canadian Institute of Mining Metallurgy and Petroleum, Montreal, 171-188.

Gao, M., Young, M., Cronin, B. and Harbort, G. 2000. IsaMill medium competency and its effect on milling performance, *SME Annual Conference,* Salt Lake City, Utah, 28 Feb. - 1 Mar.

Gardula, A., Broeckmann, C. and Paz, A. 2005. Hexadur – HIP cladded wear protection system for high pressure comminution of minerals, *Randol Innovative Metallurgy Forum,* Perth, August 21-24.

Gaudin, A.M. 1939. Principles of Mineral Dressing, McGraw Hill.

Hanisch, J. and Schubert, H. 1982. Zum einfluss der beanspruchungs-geometrie auf das zerkleinerungaverhalten von nicht allseitig begrenzten kornschichten, *Neue Bergbautech,* 12: 646-649.

Hourn, M. 2006. Private communication.

Hoyer, D.I. 1996. High-intensity autogenous reduction of critical size material, *SAG 1996,* Vancouver, 1034-1048.

Hoyer, D.I. and Lee, D.C. 1997. High-intensity autogenous liberation of diamonds from kimberlite in the Hicom mill, *Minerals Engineering,* Vol. 10, No. 3, 265-273.

Hoyer, D.I. and Morgan, R.E. 1996. High-intensity size reduction in the Hicom mill, Chemeca 96, Sydney, 30 Sept. - 2 Oct.

Hoyer, D.I., Mulvihill, B.M. and Young, G.J. 2001. Hicom attrittioning to remove alumina-silicate coatings on ilmenite sand grains, *Minerals Engineering,* Vol. 14, No. 7, 723-731.

Johnson, N.W., Gao, M., Young, M. and Cronin, B. 1998. Application of the ISAMILL (a horizontal stirred mill) to the Lead-Zinc Concentrator (Mount Isa Mines Ltd) and the mining cycle, *Proc. AusIMM '98 – The Mining Cycle,* AusIMM Annual Conference, Mount Isa, April, 291-297.

Jones, T.A. and Robinson, C.N. 2005. Application of centrifugal force to the gravity recovery and fine grinding of gold sulphide minerals, *Randol Innovative Metallurgy Forum,* Perth, Aug. 21-24.

King, R.P. 1994. Linear stochastic models for mineral liberation, *Powder Technology,* 81: 217-234.

King, R.P. 2001. Modelling & Simulation of Mineral Processing Systems, Butterworth Heinemann.

King, R.P. and Schneider, C.L. 1998. Mineral liberation and the batch comminution equation, *Minerals Engineering,* Vol. 11, No. 12, 1143-1160.

Kral, D. 1992. Red Dog: Cominco's arctic experience pays off again, *Mining Engineering,* Vol. 44, No. 1, 43-49.

Kapur, P.C., Healy, T.W., Scales, P.J., Boger, D.V. and Wilson, D. 1996. Role of dispersants in kinetics and energetics of stirred ball mill grinding, *Int. J. Miner. Process.,* 47: 141-152.

Lloyd, P.J., Bradley, A.A., Hinde, A.L., Stanton K.H. and Schymura, G. K. 1982. A full-scale centrifugal mill, *J. S. Afr. Inst. of Min. and Metall.,* June, 149-156.

Maxton, D., Morley, C. and Bearman, R. 2003. A quantification of the benefits of high pressure rolls crushing in an operating environment, *Minerals Engineering,* 16: 827-838.

Maxton, D., Firsch, S. and van der Meer, F., 2005. KHD Humboldt Wedag high pressure grinding rolls developments for minerals applications, *Randol Innovative Metallurgy Forum,* Perth, August 21-24.

Morley, C. 2005. The case for high pressure grinding rolls, *Randol Innovative Metallurgy Forum,* Perth, August 21-24.

Napier-Munn, T.J., Morrell, S., Morrison, R.D. and Kojovic, T. 1996. Mineral Comminution Circuits – Their Operation and Optimisation, JKMRC Monograph Series in Mining and Mineral Processing 2, The University of Queensland.

Nesset, J.E., Radziszewski, P., Hardie, C. and Leroux, D. 2006. Assessing the performance and efficiency of very fine grinding technologies, *Proceedings 38th Annual Canadian Mineral Processors Operators Conference,* Canadian Institute of Mining Metallurgy and Petroleum, Ottawa.

Norgate, T.E. and Weller, K.R. 1994, Selection and operation of high pressure grinding rolls circuits for minimum energy consumption, *Minerals Engineering,* Vol. 7, No. 10, 1253-1267.

Otte, O. 1988. Polycom high pressure grinding principles and industrial application, *Third Mill Operators' Conference,* AusIMM, Cobar, 131-136.

Patzelt, N., Klymowsky, R. I, Knecht, J. and Burchardt, E. 2005. HPGRs for hard rock applications, *Randol Innovative Metallurgy Forum,* Perth, August, 21-24.

Patzelt, N., Knecht H., and Baum W. 1995. Case made for high-pressure roll-grinding in gold plants, *Mining Engineering,* June, 524-529.

Pease, J.D., Curry, D.C., Barnes, K.E., Young, M.F. and Rule, C. 2006. Transforming flow sheet design with inert grinding – the IsaMill, *Proeedings. 38th Annual Meeting of the Canadian Mineral Processors,* Canadian Institute of Mining Metallury and Petroleum, Ottawa, 231-249.

Richmond, G.D. 1993. Zinc-lead ore concentration by Aberfoyle Limited, Hellyer, Tas., in *The Sir Maurice Mawby Memorial Volume Second Edition (Vol. 1),* eds J.T. Woodcock and J.K. Hamilton, The Australasian Institute of Mining and Metallurgy, Melbourne, 530-535.

Povey, B. 2005, HPGR in a wet classification circuit, *Presentation at Randol Innovative Metallurgy Forum,* Perth, August 21-24.

Scarlett, B. 1990. Particle size and shape – the motor for comminution research, *Proceedings of 7th European Symposium on Comminution,* 79-88.

Schonert, K. 1967. Modellrechnungen zur Druckzerkleinerung, *Aufbereit. Tech.*, 8: 1-11.

Schonert, K. 1985. Zur Auslegung von Gutbett-Walzenmuhlen, *Zem. Kalk Gibs*, 38: 728-730.

Schonert, K. 1986. Advances in the physical fundamentals of comminution, *Advances in Mineral Processing (Arbiter Symposium),* AIME, 19-32.

Schonert, K. 1988. A first survey of grinding with high-compression roller mills, *Int. J. Miner. Process.*, 22: 401-412.

Schonert, K. 1989. Aspects of very fine grinding, *Challenges in Mineral Processing,* AIME, 155-172.

Schonert, K. and Flugel, F. 1980. Zerkleinerung sproder Minerale im hochkomprimierten Gutbett, *Prepr. European Symposium of Particle Technology,* Vol. A, Dechema, Frankfurt, 82-95.

Schwarz, S. and Seebach, M. 1990. Optimization of grinding in high pressure grinding rolls and downstream ball mills, *7th European Symposium on Comminution,* Ljubljana, vol. 2, pp. 777-788 (preprints).

Sepulveda, J.L. and Fletcher, P.C. 1984. On the regrind of molybdenite concentrates with a tower mill, *SME Annual Meeting,* Los Angeles, Calif., 26 Feb. – 1 March.

Smith, C. 1999. Svedala Stirred Media Detritor, in *IIR Conference Crushing and Grinding '99,* March 29-30, 1999.

Smit, I. 2005. Bench scale ore characterization using the high pressure grinding roll, Presentation at Randol HPGR Workshop, Perth, August.

Steyn, J., Knopjes, B., Goodall, C. and Harris, M. 2005. An evaluation of the effect of multiple grinding and flotation stages on flotation performance of a platinum-bearing ore, *Centenary of Flotation Symposium,* SME-AusIMM, Brisbane, 6-9 June, 1037-1043.

Tavares, L.M. 2004. Optimum routes for particle breakage by impact, *Powder Technology,* 142: 81-91.

Tavares, L.M. 2005. Particle weakening in high-pressure roll grinding, *Minerals Engineering*, 18: 651-657.

Thompsen, L., Patzelt, N. and Knecht, J. 1996. High pressure grinding for copper ore at Cyprus Sierrita, SME Annual Meeting, Phoenix, Arizona, March 11-14.

Thompsen, L., Patzelt, N. and Knecht, J. 1996. High-pressure grinding for copper at Cyprus Sierrita, *Mining Engineering*, Sept., 23-26.

Veasey, T.J. and Wills, B.A., 1991. Review of methods of improving mineral liberation, *Minerals Engineering,* Vol. 4, Nos. 7-11, 747-752.

Von Michaelis, M. 2005. Real and potential metallurgical benefits of HPGR in hard rock ore processing, *Randol Innovative Metallurgy Forum,* Perth, August.

Zheng, J., Harris, C.C. and Somasundaran, P. 1996. The effect of additives on stirred media milling of limestone, *SME Annual Meeting,* Phoenix, Arizona, March 11-14.