# Particle Size Distribution Effects that Should be Considered when Performing Flotation Geometallurgical Testing

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## ABSTRACT

Flotation recovery in geometallurgical modelling is usually predicted based on relationships developed from batch flotation testing of different ore types. These batch flotation tests are usually performed at a constant set of operating conditions and at a fixed feed grind size P80. Usually the laboratory feed grind P80 is that which is expected to be produced in the full-scale flotation circuit for that particular ore type.

Flotation recovery will be a strong function of the valuable mineral particle size distribution, with lower recovery of ultra-fines due to poor flotation kinetics, optimum recovery for the intermediate sized particles and lower recovery for the coarser particles due to poorer liberation of this fraction. This valuable mineral particle size distribution is not characterised by the solids P80 parameter alone. The proportion of material in each class will also be a function of the slope of the size distribution and the degree of preferential grinding of the valuable mineral in comparison to the total mass of the ore.

This paper will demonstrate (through example) how these other size distribution parameters can change with a change in grinding technology, the size and operating conditions used in a grinding unit and the characteristics of the classification technology used in conjunction with the grinding unit. Case studies will be presented that demonstrate how this can result in these parameters being very different in laboratory testing to that produced in the full-scale comminution circuit. A rule of thumb that laboratory rod milling rather than ball milling produces a size distribution most similar to that of the full-scale grinding circuit will be challenged. These differences can result in a significant change in flotation recovery. It is therefore important to consider these parameters when developing a laboratory geometallurgical program, to enable more accurate prediction of the flotation recoveries that will be achieved in the full-scale process.

## INTRODUCTION

Geometallurgical testing that is performed to predict flotation performance achievable by an ore type in an existing flotation circuit or proposed flotation circuit design often involves small-scale batch laboratory flotation testing. These tests can range in complexity:

- A single-stage batch flotation test, which can be used to gain an indication/ranking of the expected recovery of different ore types, or used to derive model parameters for subsequent flotation modelling of performance in a multistage flow sheet.
- A rougher/cleaner test in which the concentrates are repeatedly floated to minimise entrainment and get a better insight into the final concentrate grade.
- A more complex lock cycle test procedure. Lock cycle batch flotation experiments provide a better indication of the expected grade and recovery achievable from multistage processing of an ore. These tests are performed such that they replicate, on small scale, the full-scale flotation flow sheet. To incorporate the effect of recycle streams, they are often performed multiple times with the recycled streams from a previous test added at the appropriate place in the subsequent test.

It is assumed in these types of testing that the selectivity and recovery achievable in these small-scale tests will be similar to that which will be achievable in the larger full-scale flotation machines. Therefore it is important to produce a pulp for

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testing which is as similar as possible to that which will feed the full-scale flotation process.

One variable that has a significant effect on flotation recovery is the size distribution of the valuable mineral. Particles of different size and degree of liberation float at different rates. Flotation recovery is often optimal for particles of an intermediate size (Figure 1), with coarser particles exhibiting slow flotation kinetics because of their size and poor liberation and fine particles exhibiting slow flotation kinetics because of poor flotation collision efficiency.

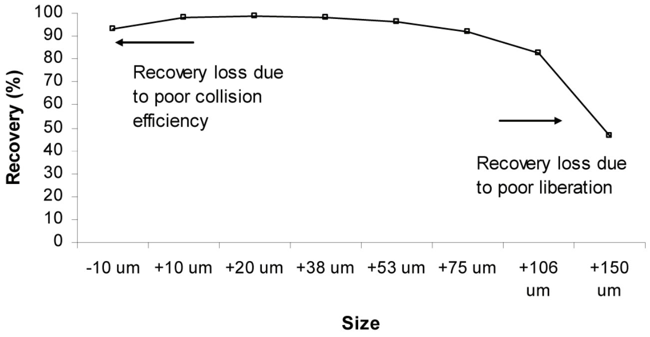



**FIG 1** - Classical recovery versus size relationship.

The overall recovery achieved by a process is dependent on the recovery of each individual size class ($R_i$) and the proportion of material in each size class ($m_i$):

$$R = \sum_{i-1}^{No\ of\ Sizes} mi\ Ri$$

Production of a sample for flotation in a design or geometallurgy batch flotation program often involves crushing or rolls crushing followed by ball or rod milling to achieve a target size P80. Often little attention is paid to other factors of the size distribution, namely its sharpness and whether the valuable mineral size distribution is very different to that of the total mass. These parameters, as will be demonstrated in this paper, can have a significant effect on the recovery achieved in the test.

A sharper size distribution curve can enable a greater proportion of particles in the intermediate size range, therefore increasing flotation recovery. This parameter is often characterised as the slope of a log-log plot of the cumulative percentage passing versus size curve, which is referred to as the modulus of the distribution (Schuhmann, 1940) (Figure 2a).

Preferential breakage of the valuable mineral over that of the gangue minerals in a feed is usually beneficial as it will result in less grinding required to achieve a particular valuable mineral P80. The degree of preferential grinding can be assessed by assaying the product of grinding and plotting the cumulative percentage passing of the valuable mineral versus the cumulative percentage passing of the total mass (Figure 2b). The further the relationship to the right on this graph, the more preferential breakage that occurred during comminution.

As will be demonstrated in this paper, these size distribution parameters are sensitive to:

- the comminution technologies and variables associated with these technologies in both the laboratory and full-scale grinding circuits
- classification methods employed in the full-scale grinding circuits.

It will also be demonstrated how these size distribution parameters (independently of the overall P80) can have a significant effect on flotation recovery. It is therefore important to consider these parameters when developing a laboratory geometallurgical program to enable more accurate prediction of the flotation recoveries that will be achieved in the full-scale process.

## SIZE DISTRIBUTION PARAMETERS PRODUCED DURING GRINDING

### Shape of the distribution

Technologies for comminuting ore have evolved over the decades in the quest to reduce costs of production. Crushers and grinding machines have got bigger in size and circuits have changed from the classic crushing/rod mill/ball mill circuits to crushing/semi-autogenous grinding (SAG)/ball milling which enable the treatment of larger tonnage rates. In recent times, a number of plants are installing large high pressure grinding rolls (HPGR)/ball milling circuits because of its lower energy use per tonne of ore processed. Vertimills and stirred mills are replacing ball mills in fine grinding applications due to their improved energy efficiency in this particle size region.

Comminution occurs as a consequence of a series of individual breakage events and is usually modelled using some sort of particle population approach. Each particle class

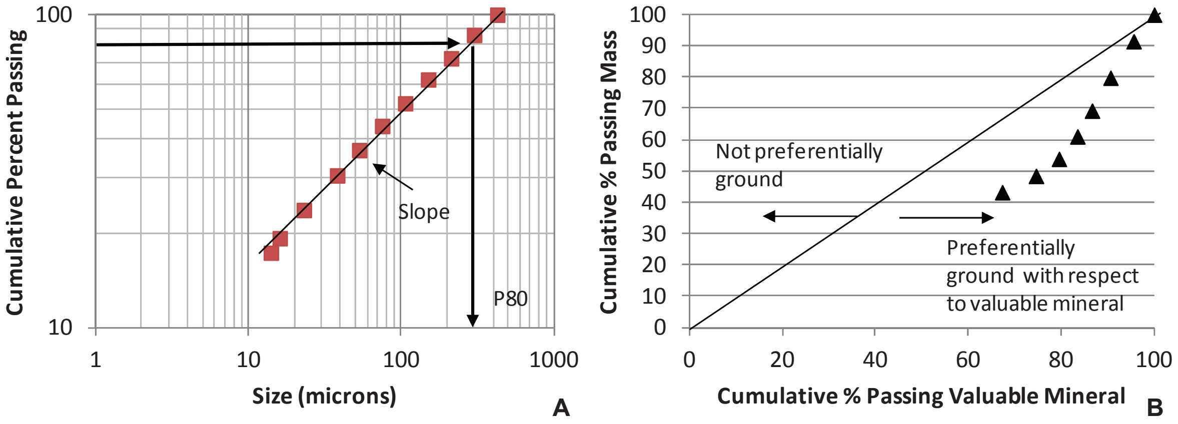



**FIG 2** - Characteristics of a feed size distribution (A) modulus/slope and P80 of the cumulative per cent passing versus size relationship and (B) cumulative per cent valuable mineral versus cumulative per cent solids.

has a probability of breakage which is a function of the size of the particle, its rate of breakage and its residence time in the device. Once selected for breakage, the size of the progeny of this breakage event is finer, the greater the energy applied to the particle and the weaker the ore breakage properties.

A sharp particle size distribution in the product of a unit will occur when the production of ultra-fines during grinding is minimised. This occurs when the probability of breakage of the coarser particles is much higher than the fines and when individual breakage events are at a relatively low energy, resulting in coarser progeny.

This is the reason why rod mill product distributions, for example, have a steeper slope than ball mill grind size distributions. In rod mills, the coarse particles act as bridges between the rods and therefore preferentially take the compressive forces over that of the fines. The breakage rate of coarse particles is therefore significantly higher than the fines. A substantial amount of work has been published in the literature comparing the size distributions produced by rod mills in comparison to ball mills (Crabtree *et al* (1964); Kinasevich *et al* (1964); Lawry and Quast, 1995a, Lawry and Quast, 1995b). Table 1 summarises the modulus of the Schuhmann relationship measured for different minerals measured by Crabtree *et al*, 1964 and Kinasevich *et al*, 1964, demonstrating that rod mills have higher modulus values than ball mills for a range of different materials.

**TABLE 1**

Comparison between the Schuhmann modulus of a rod mill and a ball mill product size distribution (after Crabtree *et al*, 1964 and Kinasevich *et al*, 1964).

| Mineral | Distribution modulus | |
|---|---|---|
| | Rod mill | Ball mill |
| Pyrite | 0.93 | 0.75 |
| Barium | 0.75 | 0.58 |
| Corundum | 0.61 | 0.65 |
| Quartz | 0.90 | 0.86 |
| Limestone | 0.61 | 0.61 |
| Calcite | 0.65 | 0.63 |
| Galena | 0.80 | 0.63 |

Because of the wider size distribution produced from ball milling, they are almost always operated in closed circuit with classification at full scale. Classification results in more selective breakage of the coarser sizes because the fines are removed preferentially from the circuit and therefore a steeper size distribution curve is produced. This was demonstrated by Armstrong (1960), who compared open and closed circuit ball milling and rod milling in the laboratory. He concluded that the size distribution from a laboratory rod mill gave a similar-shaped size distribution to that of a closed circuit laboratory ball mill. He also demonstrated how a laboratory rod mill gave a similar shape of size distribution to a 36 inch (0.8 m) Hardinge ball mill in closed circuit with a rake classifier treating the same ore. It seems to be this piece of work that is the origin for the commonly held perception that laboratory grinding should be performed in a rod mill to produce a similar shape of size distribution to the full-scale comminution circuit. Does this axiom still hold, however, as grinding and classification technologies used in comminution circuits have evolved?

Grinding mills have become much larger and cyclones are now widely used for classification. Current industrial scale ball mills, for instance, are usually about 4 to 5 m in diameter, which is much larger than the 0.8 m mill used in Armstrong's comparative study. Energies of breakage are therefore much larger and classification efficiency has likely changed. Cyclones, rather than rake classifiers, are largely used for classification, and fine screening technology has advanced to the point where it is now starting to be employed within conventional grinding circuits to increase efficiencies of separation. Jankovic and Valery (2012) have demonstrated in the laboratory how both the size of the recirculating load and the classification efficiency can have a significant effect on the sharpness of the product size distribution curve produced from milling (Figure 3).

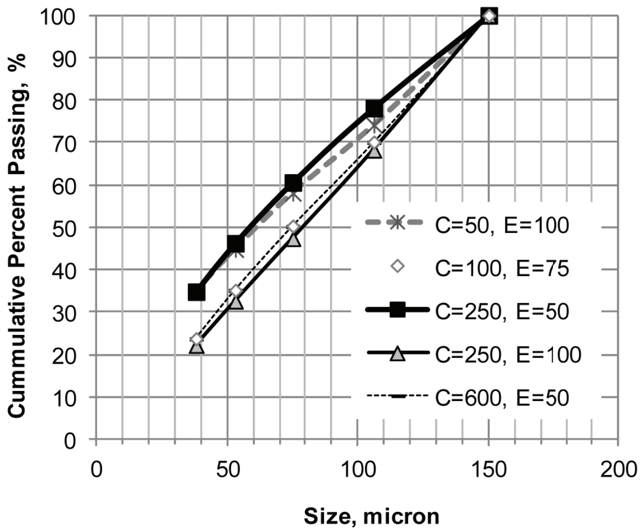



**FIG 3** - Effect of classification efficiency (E) and circulating load (C) on product size distribution (Jankovic and Valery, 2012).

Crusher/rod milling circuits have been superceded largely by SAG and autogenous grinding (AG) mills, which enable greater tonnage rates. SAG/AG mills are similar to ball mills in that breakage largely occurs due to the tumbling action of the charge and indiscriminate impact of balls/particles on the toe of the charge. The resulting product size distribution will vary as the breakage rates and transfer rates of particles through the mill vary as a consequence of the mill operating parameters (eg ball load, ball size, total mill load, operating speed, diameter, length and trommel diameter) and the hardness of the ore (Morrell, 2004). Product size distributions produced from SAG/AG circuits exhibit a wide variation in shape but are typically wider than that observed from a ball mill in closed circuit with a cyclone (Figure 4, referenced from Morrell, 2011).

Compressed breakage comminution devices (eg HPGR, vertical roller mills or VRM) are reported to produce sharper size distributions than conventional tumbling mills because in a compressed bed, the coarser particles take the load, shielding the finer particles and preferentially break (Hawkins, 2007). For example, Crosbie *et al* (2005) in laboratory and pilot plant work found that the VRM produced sharper product size distributions for the same ore than conventional tumbling mills (Figure 5a). Vizcarra (2010) found a sharper product size distribution produced from a piston and die compressed bed than in a hammer mill – a device that applies indiscriminate impact breakage to particles (Figure 5b).

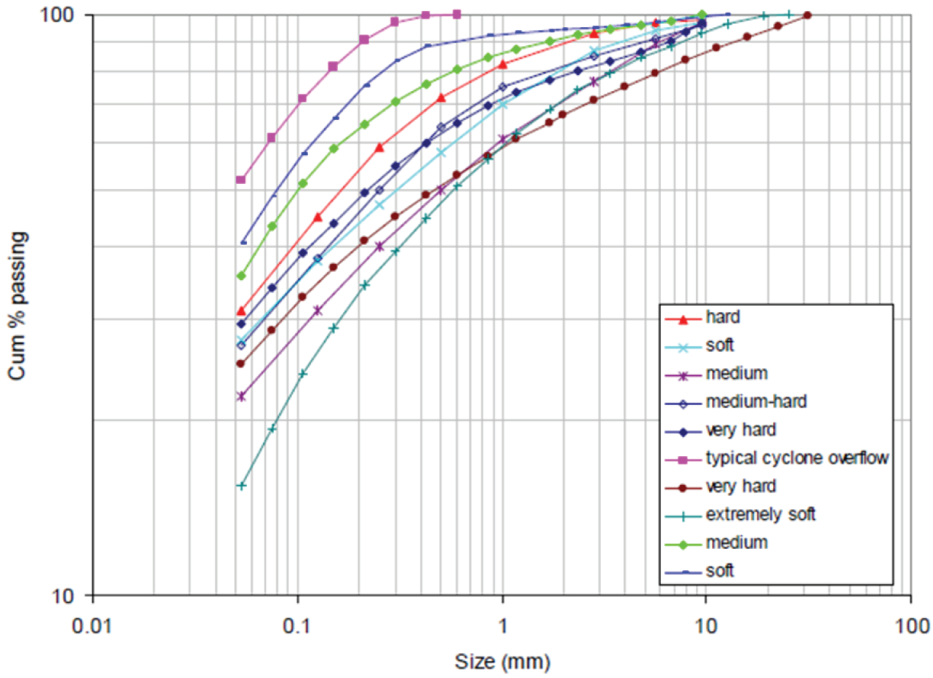
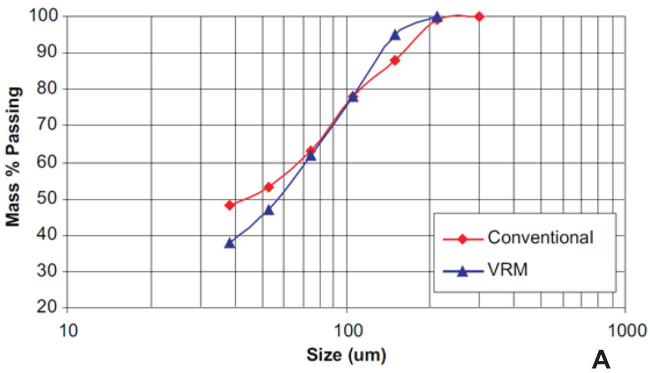



**FIG 4** - Example of size distributions produced from semi-autogenous grinding / autogenous grinding circuits (Morrell, 2011).



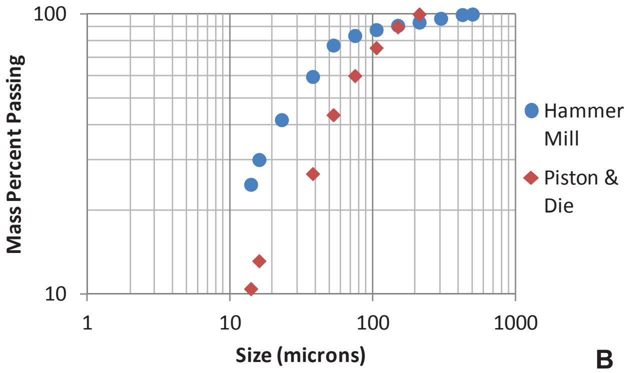




**FIG 5** - Examples of size distributions produced from compressed bed breakage and impact breakage: (A) vertical roller mill versus conventional tumbling mill for a nickel sulfide ore (Crosbie *et al*, 2005) and (B) piston and die breakage versus hammer mill for a copper sulfide ore (Vizcarra, 2010).

It is therefore concluded that product size distributions from operating plants are likely to vary widely depending on the grinding technology employed, the size and operating conditions of these technologies (eg diameter, ball load, speed, rolls pressure), the type of circuit employed (eg open versus closed circuit), the classification efficiency and the size of the recirculating loads. It's also likely that the shape of the distributions produced in the laboratory for geometallurgical testing will be different to that of the full-scale cells. This will be demonstrated later in the paper in two case studies.

## Degree of preferential grinding

It is usual that the different minerals in an ore will grind to a different degree of fineness during comminution. Sulfide minerals (eg chalcopyrite, sphalerite, galena, nickel sulfides) often are finer after breakage than their host rock. This presumably occurs because either they are softer (require less energy to break) and/or their progeny after a breakage event is finer.

As outlined in the introduction, the degree of preferential grinding that occurs can be assessed by comparing the percentage passing of the valuable mineral versus the percentage passing of the total mass (Figure 2b). What has been observed by a number of researchers is that the shape of this relationship is often independent of the degree of grinding or product P80 (Bazin *et al*, 1994; Runge *et al*, 2007). Figure 6 shows a typical result observed when ball milling the same ore for different times in the laboratory.

The authors have observed, however, that this relationship can change significantly between different grinding technologies treating the same ore or when the mode of breakage within a device significantly changes. Figures 7 and 8 show two examples of this phenomenon. The first example (Figure 7) is a recalculation of data presented by Crosbie *et al* (2005) which compared grinding the same ore in a rod mill and a VRM in the laboratory. The VRM preferentially grinds the material whereas in the rod mill little preferential grinding occurs. In a second example (Figure 8a), data presented by Palm *et al*, 2010 has been recalculated and shows that running a rod mill dry gave much more preferential grinding than running a rod mill with water added (percentage solids not specified). Interestingly, wet milling gave a sharper size distribution (Figure 8b) which is indicative of a significant change in breakage rates. The use of a HPGR or cone crusher to precrush the ore does not change the relationship.

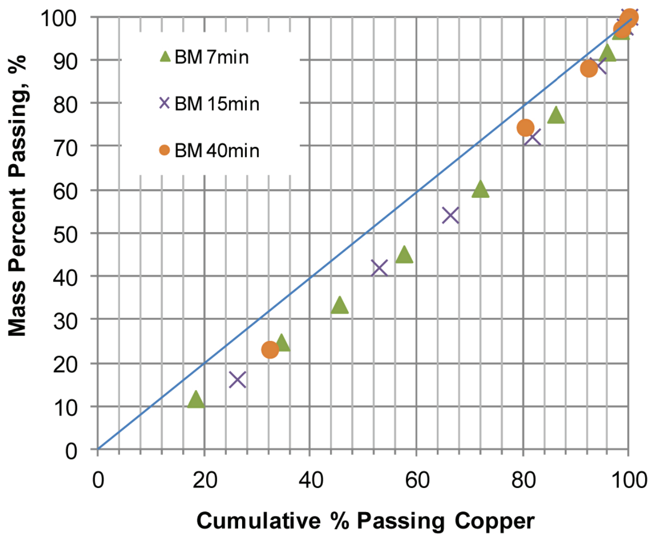



**FIG 6** - Cumulative percentage passing mass versus cumulative percentage passing copper achieved by ball milling a copper ore in the laboratory for different grind times.

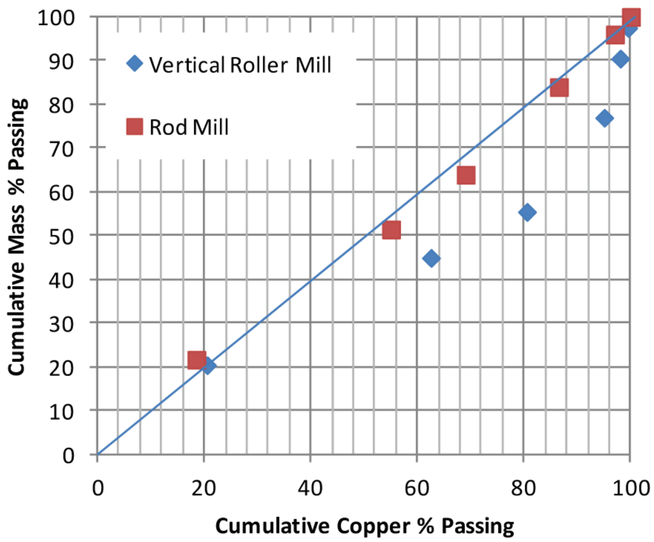



**FIG 7** - Cumulative percentage passing mass versus cumulative percentage passing copper achieved by rod milling and vertical roller milling a copper sulfide ore (after Crosbie *et al*, 2005).

Preferential grinding is also thought to be exacerbated in a full-scale circuit when classification is performed using cyclones. Cyclone cut size is affected by solids density and sulfide minerals often have a higher solids density than the host rock resulting in a finer cut size. This results in finer grinding of the sulfide mineral in comparison to the host rock.

Often when geometallurgical testing is performed, the ore is ground to a particular P80 and it is assumed that this will result in the same P80 of the valuable mineral in the full-scale cells. Differences in the mode of breakage and use of cycloning for classification at full scale may mean that this will likely not be the case. This will be demonstrated by example in the following section.

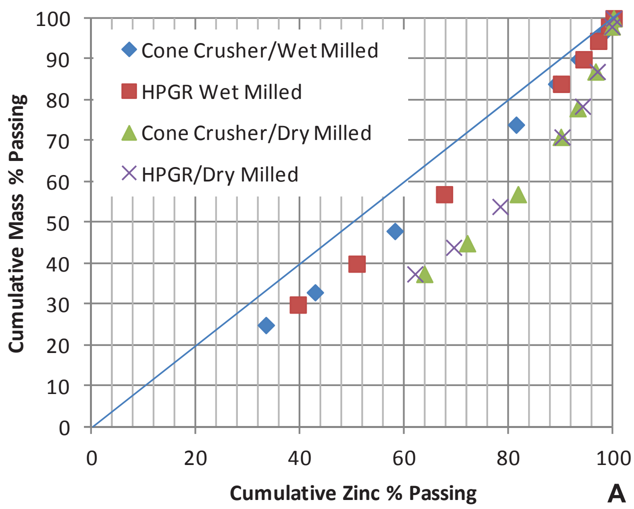
## LABORATORY VERSUS PLANT SIZE DISTRIBUTIONS

In this section two case studies will be used to demonstrate the differences that can be observed between laboratory and full-scale size distribution parameters.

### Case study 1 – copper ore milled in a rod mill/closed circuit ball mill

In case study 1, a survey was performed of the crushing, grinding and flotation circuit of a concentrator treating a copper sulfide ore. Figure 9 shows the comminution flow sheet. Feed to flotation (ie product of grinding) was sized and the size fractions assayed for copper.

Prior to surveying, drums of ore were also collected from the run-of-mine area and shipped to a laboratory for breakage and flotation characterisation. The sample was passed through a jaw crusher followed by a rolls crusher until it was -3 mm. It was then mixed using cone and quartering techniques and then split into 1 kg samples using a rotary splitter. Samples of 1 kg were milled for different times in both laboratory rod and ball mills and then sized and the size fractions of the ball mill tests were assayed. Rod mill tests were performed in a mill with a diameter of 20 cm with an 11 kg rod charge (50 per cent 26 mm/50 per cent 19 mm rods) at 50 per cent solids rotating at 55 rpm. Ball mill tests were performed in a mill with a diameter of 20 cm with an 11 kg ball charge (15 × 40 mm diameter, 38 × 28 mm diameter, 152 × 18 mm diameter) at 65 per cent solids rotating at 86 rpm.



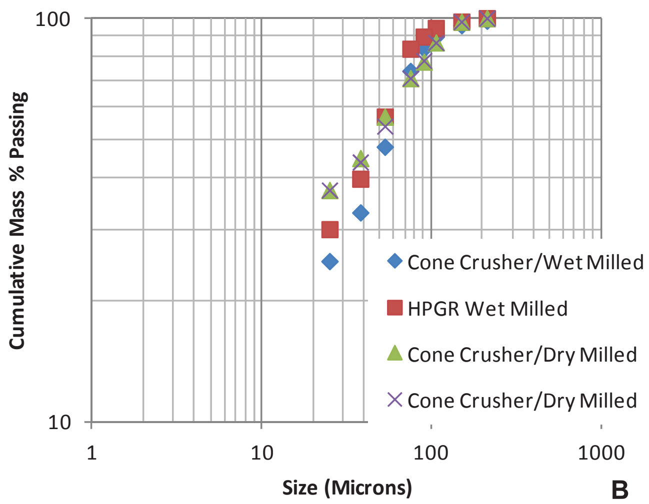




**FIG 8** - (A) Cumulative percentage passing mass versus cumulative percentage passing zinc and (B) cumulative percentage passing versus size achieved when wet and dry rod milling a zinc sulfide ore (after Palm *et al*, 2005).

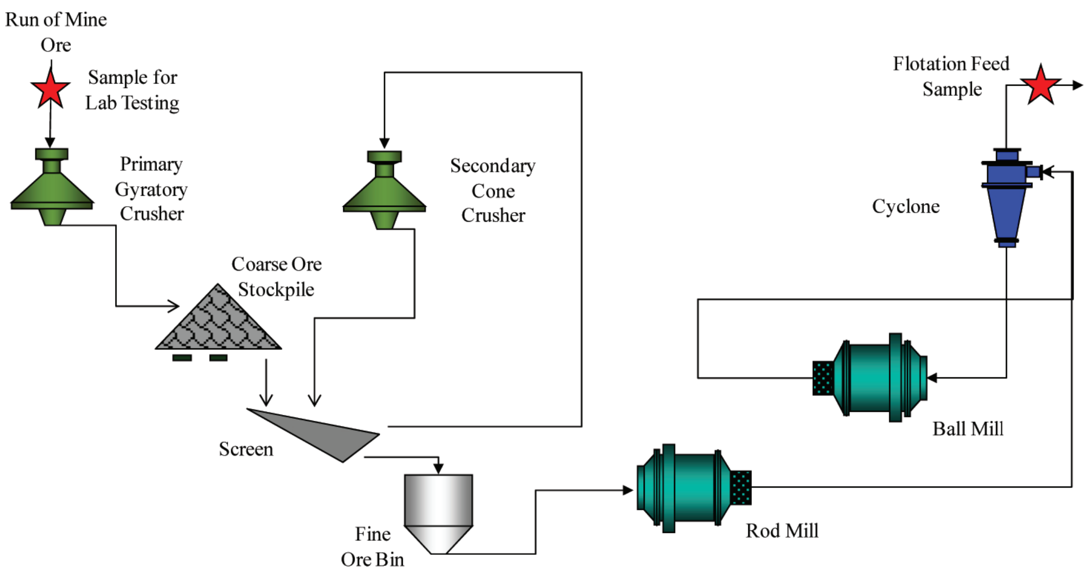



**FIG 9** - Plant comminution flow sheet for case study 1 denoting sampling points.

Figure 10 shows a comparison between the shape of size distributions produced from the laboratory rod and ball mill tests (Figure 10a) and the measured size distribution of the plant feed stream (Figure 10b).

As expected, the laboratory rod mill produces a sharper size distribution than the laboratory ball mill, producing much less -38 µm material. The plant size distribution after grinding exhibits the widest size distribution and the shape is more similar to that produced from the laboratory ball mill than the rod. This is a contradiction to the 'rule of thumb' that laboratory rod milling will produce a size distribution more similar to the full scale than a laboratory ball mill. This is particular significant because the circuit in question is a rod mill/ball mill circuit and not a SAG/ball mill circuit where the plant size distribution is expected to be wider.

Figure 11 shows that the degree of preferential grinding in the laboratory ball mill tests is similar to that observed in the plant feed data. Unfortunately the rod mill products were not assayed but it is suspected that these tests would have resulted in little preferential grinding (as shown in Figure 7), which would not have well represented the plant data.

It is therefore concluded in this case that laboratory ball milling would be the most appropriate method to grind the ore to replicate the characteristics of the full-scale plant grinding product.

## Case study 2 – Copper ore milled in a ball mill/ cyclone circuit

Case study 2 involves comparing typical plant feed size distribution parameters to that obtained from rod milling a sample collected from a circuit prior to ball milling. Figure 12 shows the characteristics of the milling circuit denoting the point at which the sample was collected in the circuit and the plant feed stream to which the laboratory results will be compared.

The laboratory information has been compiled by re-analysing data from test work performed at the Julius Kruttschnitt Mineral Research Centre (Tang, Bradshaw and Vos, 2013). In this work, a 100 kg sample of the crusher product (P80 = 3.35 mm) was collected from the circuit and shipped to the laboratory where it was mixed and then split into 1 kg samples by a rotary splitter. Rod mill tests were performed at 35 per cent solids with 15 stainless steel rods. Grinding time was altered to produce two different grind sizes (P80 = 150 and 106 µm). Copper assays were performed which enabled the copper distribution in the rod mill products to be determined.

The plant collects monthly composite samples of the feed after grinding which is routinely sized with the size fractions assayed for copper. The authors have analysed this data over a period of two years and found that, although there was some variation with P80 in the feed to the circuit, the general

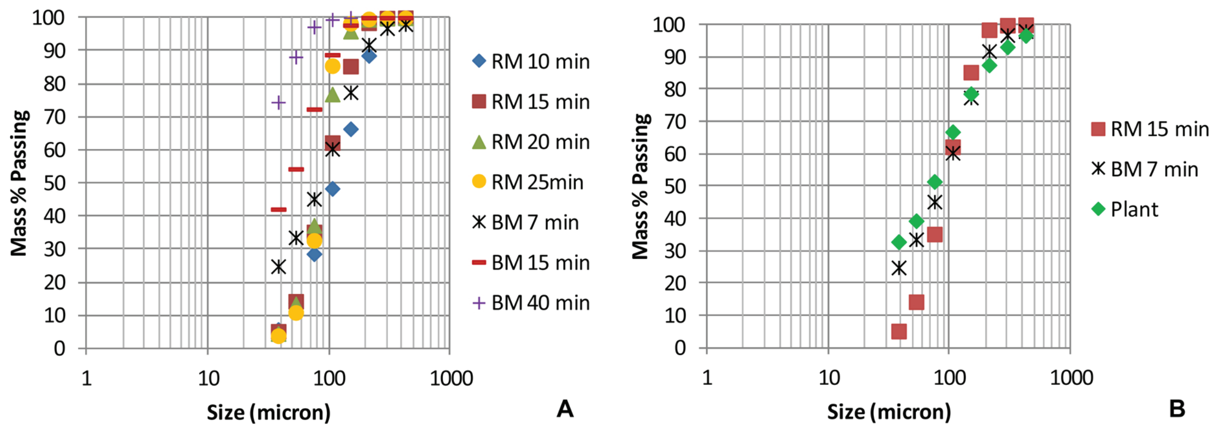



**FIG 10** - Cumulative mass percentage passing versus size for (A) laboratory rod and ball mill tests and (B) plant survey versus the most similar ball and rod mill test.

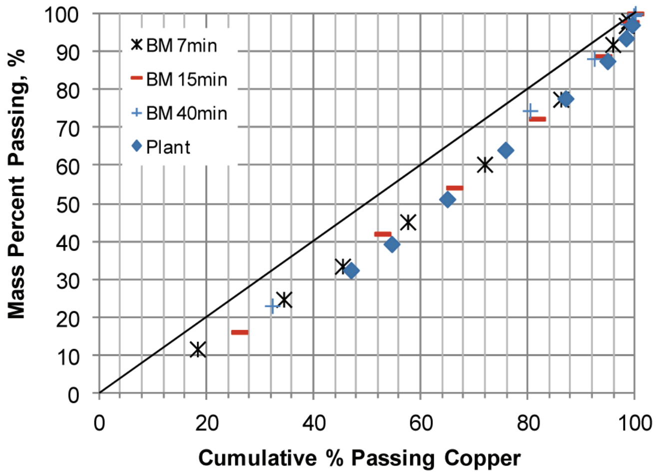



**FIG 11** - Cumulative copper percentage passing versus the cumulative mass percentage passing in the laboratory ball mill tests in comparison to the plant feed.

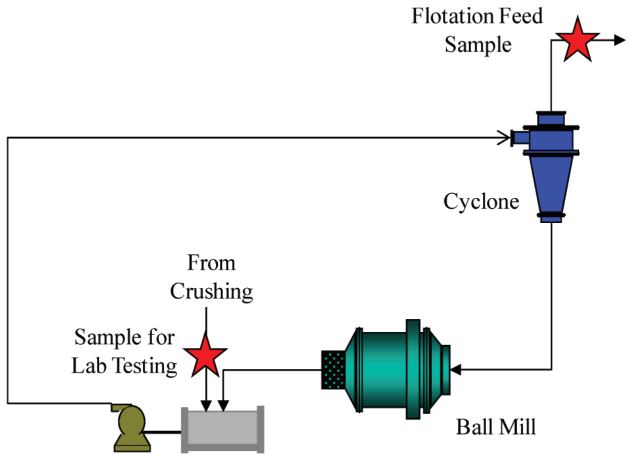



**FIG 12** - Plant grinding flow sheet for case study 2 denoting sampling points.

shape and the proportion of preferential grinding remained fairly similar over this period.

Figure 13 shows a comparison between the copper size distribution as a function of size produced in the rod milling laboratory tests and in three sets of the monthly composite plant feed data, chosen such that a range of plant feed P80 values would be represented. As observed in case study 1, the rod mill tests produce a sharper size distribution than observed in the plant data.

Figure 14 shows that in both the laboratory rod mill and in the plant ball mill in closed circuit with cyclones there is preferential grinding of the copper compared to the host rock. There is some evidence that the degree of preferential breakage increases as P80 increases. However, it is clear that there is significantly more preferential breakage in the full-scale ball mill circuit than in the laboratory rod mill. This results in the P80 of the copper in the plant being much lower than in the laboratory tests for the same equivalent solids P80. For example, at 150 μm P80 in the laboratory, the copper P80 was 128 μm. In the plant at 155 μm solids P80, the copper P80 was 86 μm.

This example provides further evidence that the shape and degree of preferential grinding can be very different in the products of laboratory and full-scale grinding. One suspects that a ball mill laboratory test would have produced a size distribution with characteristics more similar to the plant.

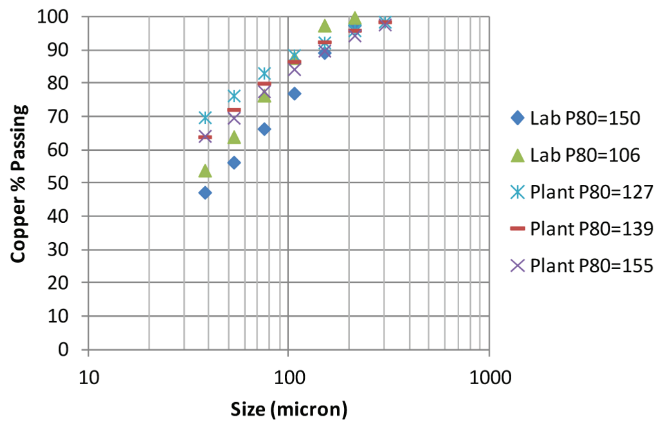



**FIG 13** - Copper percentage passing versus size achieved in laboratory rod mill tests compared to that observed in the plant feed (case study 2 data).

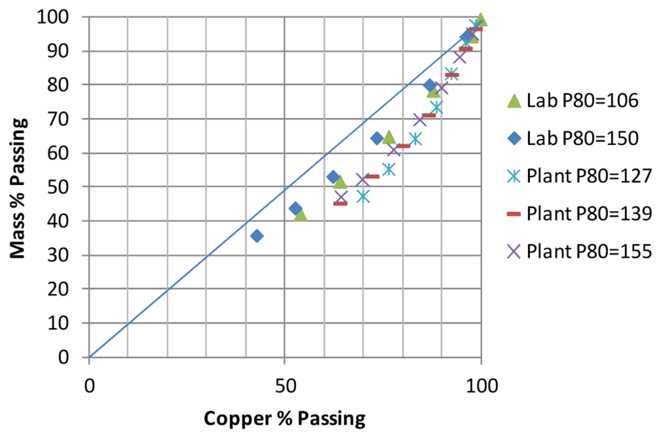



**FIG 14** - Copper percentage passing versus mass percentage passing achieved in laboratory rod mill tests compared to that observed in the plant feed (case study 2 data).

## EFFECT OF SIZE DISTRIBUTION PARAMETERS ON FLOTATION RECOVERY

Flotation recovery is strongly correlated with particle size and it is therefore likely that differences in the size characteristics of the feed to a flotation process will result in a change in performance.

Steeper size distributions will usually result in much more material in the intermediate fast-floating size classes, increasing recovery. It also has the added advantage of resulting in less ultra-fine production – ultra-fines are generally adverse to flotation, as they:

- float very slowly
- have a highly reactive surface, which can lead to greater presence of oxidation products in solution coating and retarding coarse particle flotation
- have a high surface area, so use a disproportionate amount of collector, starving coarse particles of reagent
- can result in a viscous pulp, which can slow flotation rates.

Preferential grinding can be advantageous, as it can result in less overall grinding requirements (ie can grind to a coarser size but achieve the same degree of valuable particle liberation).

Crosbie *et al* (2005) compared the flotation recovery and grade achieved after grinding using either rod milling or a VRM (the size distributions of which have already been presented in Figure 7). They found that the rate of flotation and the grade versus recovery achieved in batch flotation testing was superior for the VRM product. They also commented that they had observed much greater changes in flotation response for other ores treated in their Anglo American Research Laboratories (AARL) pilot plant facilities using these two different grinding technologies.

To demonstrate the extent to which a change in size distribution can affect flotation, the authors have simulated a 'predicted' flotation recovery for the laboratory and plant size distribution data presented for case study 2. These simulations have been performed using Equation 1, for two typical but different shapes of size recovery curve that have been referenced from industrial circuit data. Figure 15 shows the size recovery curves used in the analysis and the resulting overall predicted copper recoveries for the plant and laboratory grinding data presented in case study 2.

The first size recovery curve exhibits very high similar recoveries in the -100 µm size fractions which contain a majority of the copper mineral in all the grinding distributions (Figure 13). In this case, there is very little difference in the overall predicted recovery (Figure 15b). For the second size versus recovery curve, there is much more variation in recovery in the size classes where the valuable mineral resides. This results in quite significant differences in the predicted recovery, especially for coarser solid P80 values. As particle size decreases, the recovery in the laboratory tests increases significantly, whereas in the plant this increase is less significant. This is presumably due to the fact that the laboratory grinding distributions are much sharper and therefore a drop in P80 moves the copper into floatable classes without significantly increasing the proportion of poorly floating ultra-fines. At coarser particle sizes, however, the sharper distribution and the lack of preferential grinding result in much more material in poorly floating coarse classes and therefore much lower recoveries.

It is therefore concluded that in circumstances where a substantial portion of the valuable mineral resides in size classes with varying recovery, recoveries predicted in laboratory test work can be very different to that which will be achieved in the plant when significant differences exist between the laboratory and plant grind distributions. These differences will be amplified by any changes in the size versus recovery curve that might result as a consequence of a different size distribution (eg coarser particle reagent starvation due to excessive ultra-fines).

## IMPLICATIONS FOR GEOMETALLURGICAL TESTING

In this paper it has been demonstrated that the shape and degree of preferential grinding can be very different in the product of a laboratory grind and a plant grinding circuit.

This, depending on the shape of the size versus flotation characteristics of the ore, can have a significant impact on the recovery that would be produced from a laboratory geometallurgical flotation test. The shape of the size versus recovery relationship will change from ore type to ore type, and be a function of the degree of liberation and surface contamination and reagent coverage of the particles in the ore stream.

It's therefore important when using geometallurgical testing to compare floatability of different ore types, or to predict the performance in full-scale plants, or to assess optimum plant grind size, that the characteristics of the particle size distribution produced in laboratory grinding be as similar to that of the plant as possible.

Plant product size distributions often have a low modulus (wide rather than sharp size distribution) and exhibit some degree of preferential grinding, examples of which are shown in Figure 16. (Note that the shape of these curves will not only be a function of the grinding circuit employed but also of the ore characteristics, and therefore one should not directly compare them to assess different grinding technologies.) Laboratory rod milling, which is often used in geometallurgical testing programs, has been demonstrated in this paper to produce size distributions with a high modulus and little preferential grinding. Laboratory ball milling, on the other hand, produces a wider size distribution and more preferential breakage and therefore seems more appropriate for geometallurgical testing if a general rule of thumb is to be applied.

A general rule of thumb, however, is dangerous because as plant designs evolve and different technologies are employed for grinding and classification (eg HPGR units rather than SAG mills, fine vibratory screens rather than hydrocyloning), the shape and degree of preferential grinding of the size distribution of an ore in the plant will change. Geometallurgical testing programs should consider these variables in their testing regime. During laboratory grinding calibration, the shape of the size distribution should be measured and the size fractions should be assayed to enable an assessment of the degree of preferential grinding. The laboratory grinding machine and parameters should be chosen and varied to

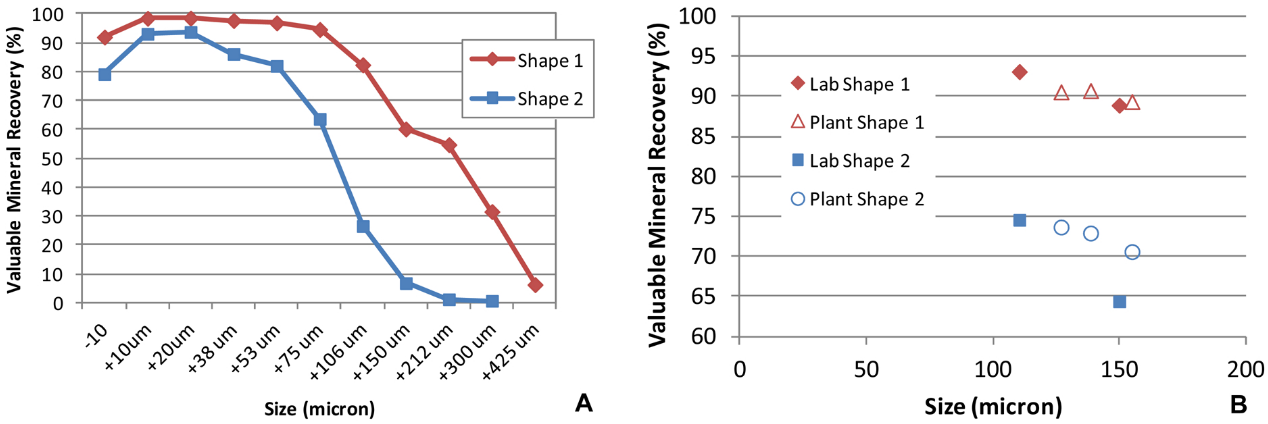



**FIG 15** - (A) Different valuable mineral recovery by size curves and (B) the predicted overall valuable mineral recovery for these shaped curves as a function of solids P80 for the laboratory and plant size distributions presented for case study 2.

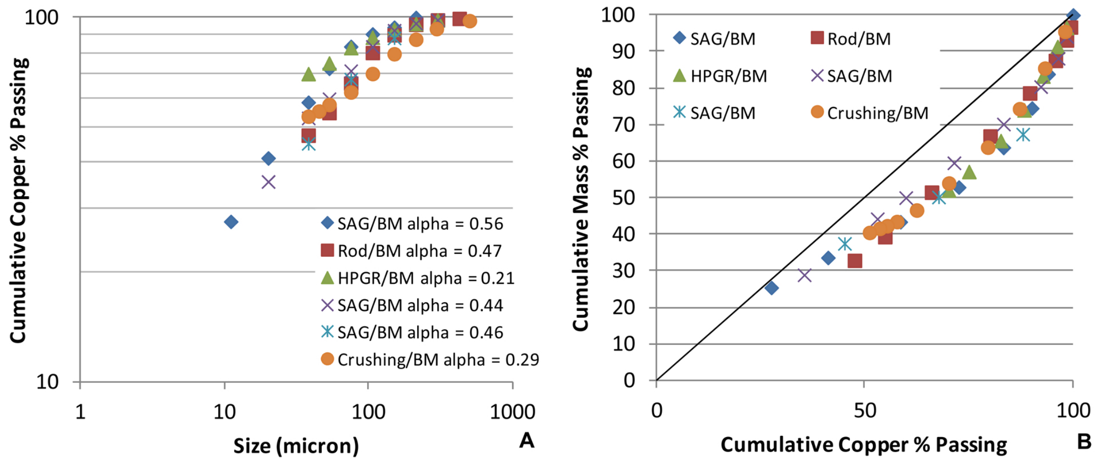



**FIG 16** - Examples of (A) shapes of cumulative copper size distribution versus size curves and (B) preferential grinding of copper of products from full-scale comminution circuits treating copper ores.

produce a feed as similar as possible to that produced from the plant. Where the plant is yet to be built, the distribution should be made to be as similar as possible to that produced from other plants processing a similar ore and utilising similar grinding and classification technologies.

The merits of using ball milling over rod milling should be assessed. Speed, loading of the mill and percentage of solids have been shown to play a relatively small role (Lawry, Schuurmans and Quast, 1998), so a change is not likely to result in a large change in the shape of distribution produced, but could be used to make modest modifications. Maybe it will be necessary to consider more radical changes in the design of our laboratory devices (eg aspect ratio) or a complete change in grinding technology. There is a need for more research in this area.

It may not always be possible to produce exactly the same size distribution as the plant so if extreme differences are expected, then corrections could be made to geometallurgical testing results, which require greater accuracy. One option is to measure the size versus recovery achieved in the batch tests and use this in conjunction with the expected plant size distribution to calculate an overall recovery.

Routine measurement of laboratory grinding size parameters (not just P80) may also highlight reasons for the difference in performance of different ore types. For example, soft host rock may exhibit very different preferential breakage to a harder host rock, which could mean that tests performed at the same solids P80 may have a different valuable mineral P80, resulting in a change in flotation recovery. Different ore types may result in different modulus values and this may be the cause of differences in recovery at what seems the same P80. This type of understanding could lead to alternative parameters for input into flotation geometallurgical predictions. It could also enable better tailoring of the process to maximise process efficiencies for different ore types.

Another important observation in this work is that the preferential breakage curve (cumulative percentage passing metal versus cumulative percentage passing solids) remains the same for a particular grinding device for different product P80 values (refer to Figures 6, 8, 11 and 14). This confirms the observations by Bazin *et al* (1994). This relationship has the potential to enable effects of grind size to be incorporated into a flotation geometallurgical model prediction. Grinding models usually produce only an estimate of the size distribution. Using the preferential breakage curve, this distribution can be used to estimate the valuable mineral distribution and coupled with the size recovery relationship expected for a particular ore, can be used to predict effects of grind size on overall circuit recovery. This will be a topic of a future publication.

## CONCLUSIONS

It has been demonstrated that flotation recovery will be not only a function of P80 but also of the sharpness or modulus of the valuable mineral size distribution and the degree of preferential grinding that occurs during comminution. The magnitude of the effect is dependent on the shape of the size versus recovery relationship in the full-scale circuit.

In laboratory geometallurgical testing, therefore, which aims to predict full-scale flotation circuit performance, it is important that these parameters of the size distribution be as similar to that which will be produced from the full-scale grinding circuit as possible. Classically, rod mills have been used in the laboratory for geometallurgical testing, as these devices were thought to produce a size distribution that better represented full-scale than ball mills. Examples presented in this paper, however, dispute this rule of thumb. Current designs of full-scale plants tend to produce size distributions with wide rather than sharp size distributions and exhibit significant degrees of preferential grinding of sulfide minerals. Laboratory ball mills, in the case studies presented, resulted in size distributions which were more similar to the full-scale plant than the laboratory rod mill.

The authors, however, warn against assuming this will be the case in every geometallurgical testing program. Examples have been presented in this paper which demonstrate how different comminution technologies used in combination with different classification regimes can result in changes in the sharpness of a particle size distribution and the degree of preferential breakage. As comminution circuits evolve, therefore, the type of grinding that will be required in the laboratory to represent full-scale behaviour will likely change. It is therefore recommended that during geometallurgical flotation testing, the size distribution produced as well as the valuable mineral distribution be measured in the feed and investigated to determine if it matches that expected (or produced) in the full-scale comminution circuit. Corrections to predictions or a modification to the laboratory grinding methodology may be required when large discrepancies are detected.

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