Improved characterisation of ball milling energy requirements for HPGR products
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Accepted for publication in: Minerals Engineering
Published version found online at: http://www.sciencedirect.com

Abstract
This paper describes a method for assessing the downstream milling energy requirements for high pressure grinding rolls (HPGR) products based on a Bond mill test procedure. Multiple trade-off studies have reported the performance of HPGR versus SAG milling with energy savings of between 11 and 32 per cent. One factor that is often inconsistently defined in these studies is the change in the Bond Ball Work Index (BBWi). The Bond test can overestimate the reduction in ball milling energy requirements for HPGR products, not due to a change in the breakage characteristics of the particles, but because the Bond test feed for a sample crushed in a HPGR has a greater fines content than a conventionally-crushed sample. This paper rigorously assesses the actual change in BBWi achieved through the use of HPGR technology.

The reduction in BBWi was found to not be dependent on the ore hardness and thus the expression of the change as a per cent is incorrect. When the size distribution of the HPGR product was matched to the crushed product, more than 95% of the samples tested resulted in a reduction in Work Index, with an average reduction of 1.9 kWh/t. A proportional reduction was seen when the Size Specific Energy (SSE) was calculated for the reconstituted samples. Six samples were tested where the original size distributions were retained and these saw greater reduction in BBWi than SSE.

A gold mine was surveyed to investigate the ball milling requirements of an industrial HPGR circuit. Two surveys of the ball mill operating at different conditions found that milling efficiency could be improved by 20%. These results highlight the importance of the mills operating conditions on energy efficiency as they can overwhelm the potential benefits of HPGR pre-conditioning.

1 Comparative Bond work index data were supplied by Frank van der Meer, Evert Lessing and Ric Stocco of Weir Minerals.
2 Design, laboratory and operating data from Tropicana Mine were supplied by Nick Clarke, Mike Di Trento of AngloGold Ashanti, and Fred Kock and Brian Putland of Orway Mineral Consultants (OMC)

Keywords
HPGR; Bond; Work Index; Comminution; Energy

Highlights
1. The ball milling requirements of HPGR and standard crusher products are measured
2. Methods for assessing the energy consumption of the Bond Ball Mill are discussed
3. The average reduction in Work Index of HPGR products was 1.9 kWh/t
4. A similar magnitude of reduction was observed in the Size Specific Energy
5. Ball milling efficiency can vary significantly independently of HPGR operation

Graphical Abstract
Relationship between reduction in BBWi and measured SSE75 with HPGR feed preparation instead of staged crushing. Circled data points are from tests where the natural size distributions were used and the two size distributions from crushing and HPGR were not matched.

Introduction
The introduction of high pressure grinding rolls (HPGRs) to the minerals industry has not led to it overtaking the well-established semi-autogenous grinding (SAG) technology despite the widely Please reference as: Ballantyne, G.R., Hilden, M. and van der Meer, F., 2017. Improved characterisation of ball milling energy requirements for HPGR products. Minerals Engineering (accepted 8 Jun 2017).
reported efficiency benefits. Multiple trade-off studies comparing different comminution circuit options have estimated that HPGR circuits can offer energy savings of between 11 and 32 per cent compared with SAG milling circuits (Davaanyam et al., 2015). The confined-bed breakage mechanism employed by the HPGR requires less energy to achieve the same degree of size reduction as SAG milling. The cost of the grinding media consumed in SAG mills provides an additional benefit that is greater for more competent ores. However, the focus of this paper is the reduced energy requirements of ball milling circuits following HPGRs. This is typically quantified based on Bond ball work index (BBWi) tests of HPGR products, which generally report a significantly lower BBWi value than for the HPGR feed (Wang, Nadolski et al. 2013). The implications for the equipment sizing of the downstream ball mills can be significant (Patzelt et al., 2006). Schöner (1988) estimated that replacing existing circuits with HPGR-ball mill circuits could increase circuit capacity by 12-25%, and reduce energy by 10-20%. However, for this to be achieved the HPGR must produce at least 10% of the final circuit product and, furthermore, damage those particles remaining unbroken.

The reduction in BBWi has been shown (Baum et al., 1997; Daniel, 2007; Esna-Ashari and Kellerwessel, 1988; Otte, 1988), at least in part, to be due to the appearance of “microcracking” fracturing of particles in the HPGR. Daniel (2007) observed microcracks in HPGR product using mineral liberation analysis while Lin et al. (2012) measured the specific internal surface area contained within them using tomography, but their contribution to reductions in BBWi has not been directly quantified.

Tavares (2005) found that HPGR product particles coarser than 1.5 mm required on average 35% less energy to fracture in comparison to crushing and produced finer progeny when subjected to single particle impact crushing. This weakening was found to increase with compaction pressure and was independent of particle position within the confined bed, but was not observed for particles finer than 1.5 mm. Tavares (2005) used this methodology to decouple the effects of increased fines production and particle weakening on the ball mill through simulation of multiple sequential breakage steps. Shi et al. (2006) also found that the increased grindability of HPGR products was greater when coarser closing screen sizes were used in the Bond Ball Mill test. van der Meer and Schnabel (1997) used pilot plant tests, torque mills of different sizes, a Hardgrove mill as well as standard Bond Ball mill tests to show that the grindability of HPGR products were reduced in comparison to crushed products, and that this reduction increased with applied pressure. Watson and Brooks (1994) supported the finding that the Work Index reduced linearly with increasing pressing force. Stephenson (1997) also found that the reduction in competence was ore-dependent, and either related to the rock structure (presence of vesicles reduced the degree of microcracks) or the fracture toughness (reduction in work index was only observed for ores with high fracture toughness). Morrell (2009) suggested that a reduction in work index of 5% could be assumed in the absence of full scale data.

There is some evidence that microcracks could lead to increased liberation if the mineral association promotes cracks following the inter-species grain boundaries (Battersby et al., 1992). The evidence for this is inconclusive and Daniel (2007) noted that accurate quantification of the effect of breakage mechanism on liberation as measured using SEM-based liberation analysis is not straightforward. Clarke and Wills (1989) compared tin ore crushed using a rod mill and HPGR and found that liberation was enhanced by compression breakage. Studies by Patzelt and Knecht (1996) and Baum and Ausburn (2011) for example have compared leachability of ores from HPGR and crushers finding that high-pressure rolls have a higher leaching rate, in particular in the coarsest size fractions. Additionally, Shi et al. (2006) found that HPGR treated material showed increased flotation responses. In contrast

however, neither Palm et al (2010) nor Solomon et al (2011) found evidence that HPGR offered any preferential breakage or increased flotation recovery for South African PGM ores. Kodali et al (2011) found no increase in leaching kinetics for a sulphide copper ore but some improvement for an oxide copper ore. Vizcarra et al. (2010) did not find any increase in liberation between compression and impact breakage and Garcia et al. (2009) and Xu et al. (2013) observed increased liberation only at impractical compression rates of 0.5 cm/day or slower. Therefore, although HPGR use compression breakage and increased liberation can be achieved via this breakage mechanism, the rate required for high throughput may be too fast for this to be realised in practice.

The Bond Ball Mill Work index (BBWi) is the industry standard procedure for assessing the grindability of ores. The test involves conducting a locked-cycle grinding test using Bond’s standard mill design. The locked-cycle test enables a batch process to emulate the grinding behaviour of a continuous mill with a 250% recirculating load. The mill product from each cycle is screened and the oversize is combined with fresh feed (of the same mass as the screen undersize which is removed) to maintain a constant filling. This process is repeated for as many cycles as is required to obtain a constant production rate of screen undersize. The standard feed for the test is prepared through staged crushing a sample to 100% passing a 3.35 mm sieve. Bond (1961) defines his work index according to:

\[ BBWi = \frac{4.91}{[P_{100}^{0.23} \times Gpr^{0.82} \times (P_{80}^{0.5} - F_{80}^{0.5})]} \] – Equation 1;

where \( P_{100} \) is the closing screen opening, in microns; \( P_{80} \) and \( F_{80} \) are the test product and feed 80% passing sizes, respectively, in microns; \( Gpr \) is the net grams per revolution averaged from the last three cycles. The numerator in Equation 1 of 4.45 published in the original equation corresponds to the work index for a short ton, whereas 4.91 should be used when using the metric tonnes.

An implicit requirement of the Bond test is that its feed and product size distributions should be, at least approximately, parallel (Musa and Morrison, 2009). The product from a HPGR (and SAG mills) tend to contain a higher proportion of fine material than a crushed feed specified for the standard Bond test (van der Meer and Gruendken, 2010). However, even with significant differences in the feed size distributions, the product size distributions from the Bond tests have been found to be identical and solely controlled by the closing screen aperture (Shi et al., 2006). Hence the Bond ball mill test should be modified when applied to the grindability of HPGR product. Figure 1 shows the product of progressive crushing to 100% -3.35 mm (standard Bond procedure) and the product of a HPGR locked-cycle test with a closing screen of 3.35 mm. Although the 80% passing sizes are similar, in this example, there was 14% material finer than 75 µm in the HPGR product compared with 8% in the crusher product, but the difference can be greater in some circumstances. The Bond test procedure can account for differences in the quantity of fines in the feed by calculating the net grams per revolution produced. And although Rowland and Kjos (1980) developed many correction factors, none of these relate to the degree of fines in the feed. Amelunxen and Meadow (2011) discussed a methodology for modifying the BBWi by introducing a net correction factor (CFnet) to allow for different degree of fines with HPGR, SAG or crushed products, but the method remains proprietary. There also exists a number of indirect methods that are used by practitioners such as the ‘phantom’ cyclone approach developed at the Julius Kruttschnitt Mineral Research Centre (JKMRC) and functional performance analysis (Bartholomew, et. al., 2014). Because of its wide acceptability in the industry, the Bond ball mill test is the most pragmatic choice for investigating the ball milling requirements of HPGR products, but it may not be the most accurate. This study aims to separate the
pre-weakening (or microcracking) effect of HPGR from the fines production in the Bond ball mill test results.

![Figure 1 - Size distributions of the product from HPGR locked cycle test with 3.35 mm screen and staged crushing to 100% -3.35 mm using the same ore.](image)

**Methods**

Two approaches are described in the literature to reduce the influence of extra fines for determining the BBWi of HPGR products and isolate any potential pre-weakening effect. (Stephenson, 1997) described reconstituting the HPGR product to match the size distribution of the staged crush size distribution. This approach will be explored further in this paper. Alternatively, a full population balance model can be developed of the Bond mill as per the (Rajamani and Herbst, 1984) method. Two other approaches have avoided using the Bond work index altogether. Patzelt et al. (2005) utilised a proprietary laboratory (dry, open-circuit) mill test that relates the generation of fines to the energy input. And Tavares (2005) conducted single particle impact crushing to investigate the relative strengths of the HPGR and crushed products.

**Reconstitution approach**

HPGR products typically have a high fines content that could cushion the grinding action in the Bond test, reduce the grinding efficiency, and consequently increase the work index. The change in fineness is also not captured by the $F_{80}$ value that is used in Bond’s analysis, as this is more closely related to the closing screen aperture chosen for the test. To eliminate the influence of feed size distribution, comparison of the Work indices of the feed sample and the HPGR products has been screened into between 6 and 10 narrow size fractions, for instance the fractions 2.8-3.35 mm, 2.0-2.8 mm, 1.0-2.0 mm, 0.5-1.0 mm, 0.125-0.5 mm, and -0.125 mm. From this feed stock of size fractions, the initial Bond test feed is reconstituted into the same size distribution as the conventionally crushed sample.

The same is performed for the required new feed additions between each Bond test cycle. Thus, the sample for the determination of the Bond Work index of the HPGR products will have the same size distribution as the samples of the conventionally crushed, HPGR feed.

In breakage processes, the more competent components in the feed tend to report to the coarser fractions in the product size distribution (Bueno et al., 2013). Thus when preparing a Bond test feed (which requires the material to be sized to minus 3.35 mm), scalping the crushed product without recycling the oversize, might skew the mineralogical composition of the Bond test feed. Therefore, the sample for Bond WI determination on the HPGR product was taken from closed circuit HPGR product or laboratory locked-cycle tests, where minerals and particle sizes have equal opportunity to be subjected to the HPGR process. In addition, in a closed circuit arrangement, an enrichment of the harder components from the ore may take place in the recycle stream and consequently in the HPGR feed. To ensure a properly prepared feed, only the steady state product of the closed circuit HPGR crushing was used. This fraction provided a fully representative sample for the Bond procedure and the results thereof can be considered as indicative of the final HPGR product. In using this technique consideration should be made to the fact that in reconstituting the crusher product size distribution, the coarse particles are enriched. The result of this will depend on whether the coarse particles are more competent due to surviving the HPGR or weaker due to an increased degree of microcracking.

**Size Specific Energy (SSE) approach**

The Size Specific Energy (SSE) is defined as the specific energy required to generate new material finer than a particular marker screen aperture (Ballantyne et al., 2015). The most common marker size is 75 µm, producing the SSE75 with the units kWh/t-75µm. The SSE75 is a useful measure for comparing the grindability of the products from crushing and HPGR because it accounts for the increase in fines in the feed and most of the surface area is contained in the -75µm fraction. The SSE75 can either be calculated from single particle impacts (Ballantyne et al., 2015) or from Bond test results (Levin, 1992). The SSE75 results in this paper are obtained from Bond test data.

The calculation of SSE75 from the Bond ball test is similar to the approach taken by Patzelt et al. (2005). That technique compared the grindability of two HPGR fractions (centre and total) and conventional crushed products in a progressive grinding test in a dry, open-circuit laboratory ball mill *(ibid)*. The amount of fines (-90 µm) was measured in the feed and then subsequent to each progressive grinding step along with the specific energy input to the mill. Figure 2 shows the results from two ore samples that were presented by Patzelt et al. (2005) with the amount of fines in the feed subtracted from the product to calculate the generation of new fines. The HPGR products had between 10 and 20% more fines in the feed, and thus the conventional BBWi values were reduced by 10 to 20% *(ibid)*. However, when the generation of new fines was calculated, both the crushing and HPGR samples exhibited the same competence in the ball mill. Ore 1 was less competent with a Size Specific Energy at 90 µm (SSE90) of 18.3 kWh/t-90µm, whereas Ore 2 had a SSE90 of 22.9 kWh/t-90µm.
Figure 2 - Generation of -90 µm material with increasing specific energy applied in a laboratory ball mill on the product from crushing and HPGR calculated from data published by Patzelt et al. (2005).

It is possible to calculate the SSE using the industrial standard test procedure of the Bond ball mill test, however it requires a measurement of the energy consumption of the mill. The Bond test relies on empirical fitting parameters determined by Bond and expressed as exponents on the closing screen size (0.23) and the net grams per revolution (0.82). These parameters were fitted to equate the results from the standard Bond Ball test to a standard pilot plant utilising a 2.44 m diameter ball mill (Bond, 1961). As highlighted by Levin (1989), the results from the Bond test can be used to calculate the predicted power draw of the mill. The standard Bond (1952) equation can be used to calculate the specific energy required to go from a specific feed particle size ($F_{80}$) to a product size distribution ($P_{80}$):

$$\text{Specific energy (kWh/t)} = 10 \times \text{BBWI} \left( P_{80}^{0.5} - F_{80}^{0.5} \right) - \text{Equation 2.}$$

Substituting equation 1 into equation 2 will result in a calculation for the specific energy requirements of the Bond’s 2.44m pilot mill directly from the passing screen size ($P_{1}$) and the grams per revolution (Gpr):

$$\text{Specific energy (kWh/t)} = 49.1 / \left( P_{1}^{0.23} \times \text{Gpr}^{0.82} \right) - \text{Equation 3.}$$

Equation 3 can be rearranged to provide the effective joules per revolution ($J/\text{rev}$) that can be expected of the pilot mill:

$$\text{Energy (J/rev)} = (177 \times \text{Gpr}^{0.18}) / \left( P_{1}^{0.23} \right) - \text{Equation 4.}$$

Levin (1989) used a similar (but slightly different) calculation and found that the average energy value of a Bond test was 1425 x 10^{-6} kWh/min or 71 J/rev. However, analysing the same data using the equation above resulted in an average figure of 62 J/rev. In using an average figure, Levin (1989) assumed that although the energy consumption is dependent on both the grams per revolution and the closing screen, these factors are related and thus there were ‘core (normal) values’ that could be averaged. Values of energy consumption outside the defined ‘core (normal) values’ were thus determined to be errant. However, when applied to a larger dataset, the values are found to be more normally distributed with fewer errant values (see Figure 3).
In addition to expanding the ‘core (normal) values’ identified by Levin (1989) to a distribution, adding the expanded database identified that the energy consumption was highly dependent on both closing screen aperture and net grams per revolution (Figure 4). Although the net grams per revolution is dependent on the closing screen size, this relationship is not sufficient to overcome the relationship developed by Bond (1961).

Daniel (2007) measured the net electrical power draw of a laboratory Bond mill to be 91.4 J/rev. This experimental value is comparable to 93 J/rev estimated by Bond (1949); 91.4 J/rev estimated using the Morrell (1992) power model; and 93.2 J/rev estimated by the Hogg and Fuerstenau (1972) power model.
model. The power draw has also been measured using a torque meter for Bond tests conducted at Weir’s KHD laboratories in Germany, yielding a mean of 81.5 J/rev and a standard deviation of 3.7 J/rev (see Figure 5). This energy requirement of the mill is not affected by external parameters such as closing screen size or grindability unless the frictional characteristics, density or viscosity changes. The effective power draw calculated using the Bond (1961) formula therefore encompasses the inefficiencies of the laboratory mill in comparison to the pilot mill. Since the closing screen and net grams per revolution is used to calculate the BBWi, there is a relationship between BBWi and the effective energy efficiency of the laboratory mill (Figure 6). The lower energy efficiency of the laboratory mill may also be due to running dry—Tüzün (2001) found that compared to regular dry Bond tests, a wet Bond test used 30% fewer revolutions. The lower energy efficiency at higher BBWi may also be a reflection of the lab-scale mill having insufficient energy to efficiently grind coarse competent rocks.

![Figure 5](image-url)  
*Figure 5 – Calculated, estimated (Daniel, 2007) and measured energy consumption from Bond tests with the grams per revolution (g/rev) indicated by the width of the data points.*
A calculation or measurement of the energy consumption of the laboratory is necessary to calculate the SSE75 from the Bond (1961) test. This investigation highlighted that using the Bond (1961) calculated energy is flawed because it takes into account the inherent inefficiencies of the laboratory test. However due to its widespread acceptability, it will be used as an initial estimate. The SSE75 can be calculated using Equation 3 as a basis and including the generation of new –75μm:

\[
\text{SSE75 (kWh/t–75μm)} = \frac{49.1}{[P^{0.23} G\text{pr}^{0.82} (\%–75μm\text{prod} - \%–75μm\text{feed})]} \quad \text{– Equation 5.}
\]

The actual power draw of the mill was also measured for selected tests using a torque sensor on an instrumented ball mill. This measurement was used to obtain a ‘measured SSE’ which was compared to the above ‘calculated SSE’.

The SSE75 has been calculated and measured from the database of Bond Ball mill tests using the methodologies outlined previously. The calculated SSE75 compared well with the BBWi calculated from the same test (Figure 7), although the SSE75 is a slightly greater value (as shown by the parity line). The BBWi corresponds to the specific energy required to reduce intact rock structure to a P80 of 100 μm; whereas the SSE75 represents the energy required to produce a tonne of new minus 75 μm material. Therefore, the difference between the two measures mathematically equates to the specific energy required to reduce the material from a P80 of 100 μm to 100% passing 75 μm. In reality this is not exactly true as the gradient of the particle size distributions plays an important role in these calculations. The measured SSE75 was higher again than the calculated SSE75 as the measured energy consumption was significantly greater than the calculated energy. Also, because the energy consumption is measured, the energy does not vary with closing screen size or grindability, but with the filling and speed of the mill.
Results

200 Bond Ball Work Index (BBWi) tests were conducted at Weir’s KHD laboratories on products from both HPGR and progressive crushing. The samples came from a range of commodities and countries: 32 gold, 73 copper, 67 iron and 28 from other commodity groups, and these occurred in 24 countries globally. The HPGR products were reconstituted to be identical to the progressive crushing product before conducting the BBWi test, and therefore any differences are likely to be due to a weakening effect. As shown in Figure 8, the crushed product was consistently measured to have a higher BBWi than the HPGR product. The repeatability of Bond tests has been measured by Weier (2016) to be ±4.8% (95% Confidence Interval) which is significantly smaller than the average difference. The reduction in BBWi with HPGR processing was not related to the Work Index of the ore and thus the standard method of representing the reduction as a percentage is not applicable (see Figure 8). The average reduction in BBWi across all these tests was 1.7 kWh/t and a standard deviation of 1.1 with no significant influence of the ore-types evaluated (copper, gold and iron ores). Stephenson (1997) found the reduction to be dependent on ore-type, therefore a more detailed description of the gangue mineralogy and grain structure may show some dependence. The distribution of BBWi from the tests was converted into a box-and-whisker plot and the median was used to plot the reduction on the Bond Energy Intensity Curve (Ballantyne and Powell, 2014). The distribution shown for both HPGR and crush product do not represent the experimental error, but the distribution of the database results. The position of the crushed ore near the 50th percentile is a reflection of the fact that the majority of the underlying energy curve have been measured on crusher product. The reduction in BBWi from crushed to HPGR product is shown to be significant as it results in an effective reduction from the 48th percentile to the 20th percentile on this Energy Curve (Figure 9).
The SSE75 calculation for the Bond tests in the Weir database were not expected to dramatically alter the results seen from the BBWi because the HPGR product size distributions are matched to the product from crushing. This result was confirmed when the reduction in BBWi was plotted against the reduction in the measured SSE75 (see Figure 10). In addition to all the tests with matched size distributions between HPGR and crushed products, six tests were conducted where Bond tests were conducted on the natural products. These tests are highlighted with red circles in Figure 10. It was observed that when the Bond test was conducted on the naturally finer HPGR products, the reduction in BBWi was greater than the reduction of SSE75. The reduction in SSE75 was significantly lower in all but one case and most tests actually showed an increase in competence when the HPGR was used as the feed preparation. This increase is most probably a result of the additional fines cushioning the impact events within the mill.

![Figure 10 - Relationship between reduction in BBWi and measured SSE75 with HPGR feed preparation instead of staged crushing. Circled data points are from tests where the natural size distributions were used and the two size distributions from crushing and HPGR were not matched.](image)

**Discussion**

The Bond ball mill test is the industry standard in testing the grindability of ore, but it is dependent on the ore being prepared via staged crushing. It can be used to assess the grindability of ore from alternative processes such as HPGR, but this method comes with risks. The increased fines found in the HPGR product has the potential to cushion impacts within the mill as well as providing more material close to the product screen size in the feed. The reconstitution method utilised here enables the size distributions to be matched, thus avoiding these problems. However, in so doing, it may introduce alternative bias in the results. The HPGR shows the natural tendency to preferentially break softer material. Thus, the coarser, more competent, material that is concentrated when the sample is reconstituted to the crushing size distribution. This would have the effect of reducing the perceived benefit of HPGR over crushing which is a lower risk than the alternative.
Industry implications
The reduction in BBWi following HPGR crushing has been presented in many industrial design case studies (Amelunxen and Meadows, 2011; Oestreicher and Spollen, 2006; Wang et al., 2013). One such example was Vanderbeek et al. (2006) who attributed a 10% reduction in Bond work index at Cerro Verde to HPGR induced microcracking. However, no expectation of a weakening effect was assumed in the design process for equipment sizing; the ball mills were sized based solely on the increased fines expected from a SAG mill in comparison to a Cone crusher product. In a subsequent study at the Cerro Verde site, Koski et al. (2011) found that microcracks were present in coarse HPGR product at high pressures, however the ball mills required 20% more capacity than designed. This shortfall was attributed to neither achieving the expected fines generation in the HPGR nor the measured reduction in BBWi. Had the measured reduction in BBWi been accounted for in the design, the difference between design and performance of the ball mills would have been 34%. Due to the commercial importance of achieving design throughput capacity and the cost implications of under- (or over-) designing a circuit, accurate determination of the ball milling requirements of HPGR product is imperative.

Tropicana Gold Mine provides another case study in the ball milling requirements following HPGR. Anglo Gold Ashanti’s Tropicana Gold Mine is located in the newly developed Albany-Fraser Orogen, east of Kalgoorlie, Australia. Following an extensive feasibility study, a HPGR-ball milling circuit was justified based on the reduced power requirements and reduced operating expenses relative to SAG milling. The HPGR-ball mill circuit was estimated to require 75% of the energy of a standard Semi-Autogenous Grinding (SAG) mill circuit. Even with an anticipated greater capital cost, the HPGR-ball mill circuit was found to give significantly better financial returns in all cases investigated in the feasibility study (Ballantyne et al., 2016). A comparison between the specific energy requirements of the HPGR, crush and SAG based circuits (SABC) from the feasibility study is shown in Figure 11 on the Tonne Intensity Energy Curve (Ballantyne and Powell, 2014). The specific energy requirements of all options lie in the upper quartile of major operating mines in the sector, however the HPGR circuit is considerably more efficient. This benefit combined with the high cost of electricity at the mine motivated the decision to employ a HPGR circuit design. Analysis of the grindability of HPGR product was conducted in the feasibility study and it was found that although a reduction was seen when the natural size distributions were tested, when the reconstitution method was used, this reduction disappeared. A series of timed grinds were then tested, but it was determined that there was not enough confidence in the results to include it in the final design.
A survey was conducted at the Tropicana mine to compare the performance of the HPGR and ball mills with the design. The ore competence was similar to the design, the Bond Standard Work index was 18.8 kWh/t and A*b was 39.8, compared with the design values of 18.9 kWh/t and 31.5 respectively. Laboratory locked-cycle HPGR tests were also conducted on the ore. The energy required to produce new minus 75 μm material, or Size Specific Energy (SSE75), was used to compare the laboratory, design and operating efficiency of the circuit (see Figure 12). The efficiency of the HPGR during the survey was similar to both the overall circuit design and the HPGR laboratory results, reflecting the accuracy that is possible for design and operation of HPGR circuits. This is made more remarkable since the throughput and grind sizes are significantly different between the different cases. There are two results for the ball mill that reflect the performance of the ball mill in two separate surveys, one conducted with the ball mill operating at 83% critical speed (Nc) and the second at 75% critical. The performance of the mill improved at the lower speed, with a higher throughput, finer grind at lower power consumption achieved at 75% critical speed. This result was thought to be due to a combination of poor motor power efficiencies and reduced grinding performance. A slip energy recovery (SER) drive is used to control the mill speed by running at slower than synchronous speeds, therefore rotor current and shaft power are not directly proportional. The site have also identified differences in ball size, trajectory and liners profiles at this point in time. It is clear from these results that an operating ball mill can perform more, or less, efficiently than the HPGR in response to changes in internal grinding conditions, independently from HPGR operation.
Figure 12 - Comparison between circuit design, HPGR laboratory locked-cycled tests and survey results from Tropicana mine on the SSE75 Intensity Energy Curve. Two surveys were completed with the ball mills operating at different critical speeds (Nc).

Conclusions

The high efficiency of the grinding mechanism of the HPGR has been demonstrated in previous studies and is not disputed, but the ball milling requirements following HPGR has not hitherto been clearly outlined. Design studies investigating the trade-off between SAG and HPGR based circuits are regularly being conducted with lower Bond work index in the HPGR circuits. The reduction in BBWi is reportedly explained by the presence of microcracks generated under the high forces of HPGR crushing, however the increased fines in the HPGR product, relative to stage crushing, also plays a large role in reducing the apparent BBWi. The standard BBWi for stage-crushed feed is used for SAG circuits because of the apparent lack of microcracks. The high proportion of surface breakage that occurs in SAG milling can result in the generation of more fines than the HPGR product. Thus if the difference in BBWi is solely due to the difference in fines in the feed and not a change in competence, the BBWi of SAG product could be less than HPGR product, not greater. Therefore, the effect of fines in the feed needs to be isolated from the effect of microcracks when using the results from the BBWi to determine the ball milling requirements following HPGR.

The hypothesis that this paper aimed to explore is that using per cent passing a marker size such as 75 µm is a more effective way of assessing the grindability of an ore than a P80 because it is more closely related to the surface area generation. The feed to Bond Ball Mill tests had similar P80S even when the amount of fines were different. When the natural size distribution from staged-crushing and HPGR were characterised, the reduction in BBWi for the HPGR was much greater than the reduction in SSE75. When the HPGR product was reconstituted to have the same size distribution as the crushing product, similar degrees of reduction were observed for BBWi and SSE75. Thus, using
the SSE75 is preferred to the BBWi to characterise the grinding requirements of HPGR product versus crusher product. A methodology to calculate the SSE75 from a standard Bond Ball Mill test has been described. This methodology will help identify undersized ball mills following equipment that generate different size distribution gradients from the standard staged crushing product.

This analysis exclusively focused on the Bond ball test, but in so doing, highlighted many flaws in this test for the characterisation of HPGR products. The HPGR action is likely to result in weaker coarser progeny, due to microcracking, and intact fines. In addition, the HPGR product is typically screened at coarser sizes than 5 mm. The fact that the standard Bond test is conducted on -3.35 mm material means that the selective weakening of coarser particles is not isolated. The mineral processing practitioner should make every effort to separate the effect of changes in the shape of the size distribution from the strength of the rock when characterising ore for ball milling following HPGR. More measurement of the degree of weakening and microcracking would be obtained by combining single particle strength measurements with bulk tomography measurements. This would enable the strength by size relationship to be determined and correlated to a measure of degree of microcracking. This methodology would allow the full HPGR progeny to be measured and could also be used to measure the degree of preferential liberation that occurred. These results should then be compared to the product from a pilot scale SAG mill to reduce the bias that could be created from scalping the crushed feed.

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Acknowledgements

With thanks to the Coalition for Energy Efficient Comminution (CEEC) who provided the resources to complete this analysis and Weir Minerals, AngloGold Ashanti and Orway Mineral Consultants (OMC) who showed enthusiasm and openness to collaborate in this investigation. As stated in the authorship notes: comparative Bond work index data were supplied by Frank van der Meer, Evert Lessing and Ric Stocco of Weir Minerals. And design, laboratory and operating data from Tropicana Mine were supplied by Nick Clarke, Mike Di Trento of AngloGold Ashanti, and Fred Kock and Brian Putland of Orway Mineral Consultants (OMC).