Breakage and liberation characteristics of low grade sulphide gold ore blends

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ABSTRACT

Within the scope of the evaluation and optimization of a grinding circuit, the breakage and mineral liberation characteristics of three low grade sulphide gold ore blends have been investigated by Bond tests, batch grinding tests, and mineral liberation characterization. The tests were conducted in a size range from 0.063 to 2 mm. It was found that the breakage of all blends follows a first-order behaviour for all feed sizes. The work index was correlated with the quartz content and the breakage rate deceleration parameter, which both showed a linear relationship. The correlation between breakage function fineness parameter and first-order rate constant also satisfied a linear relationship. The breakage parameters established from batch grinding and grindability studies indicate differences in the breakage behaviour of the three ore blends. However, the mineral liberation properties of the valuable phase in three blends show minor differences.

1. Introduction

In the minerals industry, it is important to understand how mills will respond to variations in the grindability of ores coming from different parts of a deposit. It is based on the fact that comminution accounts for approximately 65–85% of all energy used for processing ore (Deep level mining consumes the major part in some mines) and that only 1–2% of the supplied energy is translated to the creation of new surface area (Tromans, 2008). Nevertheless, comminution circuits determine the success of overall mineral processing plants; sufficient comminution products are the basis of good results in beneficiation, extraction and recovery stages, and vice versa (Wills and Finch, 2016). The main purpose of comminution is to liberate valuable minerals from the gangue prior to subsequent beneficiation processes such as flotation or leaching. However, the performance of comminution circuits is typically modelled, designed or assessed based on product size reduction rather than liberation. In order to properly design, diagnose, monitor and optimize comminution processes, the liberation characteristics of ore minerals have to be of equal interest and should not be ignored. If such aim is achieved, not only is energy saved by size reduction processes, but also, any subsequent separation stage becomes easier and cheaper to operate.

One of the main challenges in mineral liberation is the changing grinding and liberation behaviour of the material as the mineralogical composition of the feed varies with time. Therefore, the ability to predict how minerals act during grinding will be important for two reasons: (1) the output of processing plant can be projected based on present condition, and (2) further actions can be taken in order to meet the target product size.

The grinding result is determined by two components, the circuit or equipment on one side and the material itself on the other. Studies on the correlation between mineralogical composition of the material and the grinding properties of blends, ores or pure minerals provide information about the integral or individual liberation characteristics. Thus, it becomes possible to estimate and predict the liberation based on mineralogical data of a deposit, even before a mine is in operation.

Another approach in studying the grinding behaviour is by determination of Bond's work index (Bond and Maxson, 1943). This is the comminution parameter which expresses the resistance of material to crushing and grinding. It is derived from the Bond grindability test, which is a dry laboratory simulation of closed circuit grinding. Apart from Bond’s work index, selection and breakage functions are used to describe the grinding kinetics. This is based on the theory of comminution that considers the process as being represented by two events (Kelly and Spottiswood, 1990): (1) the fracture event, where a particle is selected for breakage (represented by the selection function), and (2)

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the fracture process, where a broken particle produces a given distribution of fragment sizes (represented by the breakage function).

The present paper analyses, compares and correlates the grinding and liberation characteristics of three low grade sulphide gold ores. Three properties were investigated from grinding tests: Bond’s work index, breakage function, and selection function. The modal mineralogy and liberation characteristics of the ores were also determined in order to provide the correlation between the mineralogical composition and their grinding behaviour. The practical background of this work is to predict key grinding circuit parameters in order to be prepared for future challenges.

2. Theory

It is generally accepted that the rate of disappearance of particles being ground in a mill is proportional to the amount of particles present. This assumption known as the first-order breakage law, results in a similarity between milling and chemical reactions (Reid, 1965). Thus, the rate of breakage of the material that is in the top size interval 1 is expressed as (Austin, 1972):

$$\frac{dw_1}{dt} = S_1 w_1(t)$$  \hspace{1cm} (1)

where $S_1$ is the specific rate of breakage (time$^{-1}$). If $S_1$ does not change with time (i.e. first-order breakage process), Eq. (1) integrates to:

$$w_1(t) = w_{10}e^{-S_1 t}$$  \hspace{1cm} (2)

That is,

$$\log[w_1(t)] = \log[w_{10}(0)] - \frac{S_1 t}{2.3}$$  \hspace{1cm} (3)

where $w_1(t)$ is weight fraction of mill hold up, that is of size 1 at time $t$. Therefore, a plot of $w_1(t)$ vs. $t$ should give a straight line on a log-linear scale and $S_1$ can be determined from the slope of the plot. However, at longer grinding times, $S_1$ decreases as fine material start accumulating in the mill causing a non-first order breakage, i.e., slowing down breakage (Austin and Luckie, 1972).

The following model was used by Austin et al. (1984) to express the effect of particle size on the specific rate of breakage:

$$S_i = \frac{a \lambda^x}{1+\left(\frac{\lambda}{x_i}\right)^{\gamma}} \lambda \geq 0$$  \hspace{1cm} (4)

where $x_i$ is the upper limit of the size interval $i$ in mm, and $a, \lambda, \mu$ and $\lambda$ are the model parameters. $\alpha$ and $\lambda$ are characteristic constants which depend on the material properties. $a$ is a positive number normally in the range 0.5–1.5. $\lambda$ is also a positive number, an index of how rapidly the rates of breakage fall as particle size increases. $\mu$ is a characteristic constant dependent on mill conditions and can also depend on material properties as it implies how fast grinding occurs. $\mu$ is dependent on mill conditions.

The weight fraction of the material broken initially from size interval $j$ down to the size interval $i$ is defined as the primary fragment distribution (breakage function) $B_{ij}$. This is conveniently presented in cumulative form as:

$$B_{ij} = \sum_{k=0}^{i} b_{ij,k}$$  \hspace{1cm} (5)

The values of $B_{ij}$ can be estimated from size analysis of the grinding product after a short grinding interval and with an initial mill charge. Here, the material is predominantly in size $j$ (Austin et al., 1984) (i.e. one-size fraction BII method). According to BII method:

$$B_{ij} = \frac{\log[(1-P_i(0))/(1-P_r(t))]}{\log[(1-P_i(0))/(1-P_r(0))]} \hspace{1cm} j > i$$  \hspace{1cm} (6)

where $P_i(t)$ is the mass fraction smaller than size $i$ at time $t$. $B_{ij}$ can also be fitted to an empirical function relating particle size $x_i$ as

$$B_{ij} = \beta_i \left(\frac{x_j}{x_i}\right)^{\gamma_i} \hspace{1cm} \phi_i > j$$  \hspace{1cm} (7)

$$\phi_i = \beta_i \left(\frac{x_j}{x_i}\right)^{\gamma_i}$$  \hspace{1cm} (8)

where $\phi$, $\beta$, $\gamma$ and $\sigma$ are model parameters dependent on material properties. Thus cumulative breakage functions $B$ are the same for different ball filling ratios, mill diameters, etc. (Austin et al., 1984). Values of $\beta$ are generally between 2.5 and $5/\gamma$ mainly in the range of 0.5–1.5. $\phi$ represents the fraction of fines produced in a single fracture step, depends on the material and ranges from 0 to 1. If $B_{ij}$ values are independent of the initial size (i.e. dimensionally normalizable), then $\sigma = 0$. This parameter characterizes the degree of non-normalization.

3. Experimental

3.1. Material and initial sample preparation

Samples of the feed to SAG mill at Acacia’s Buzwagi Gold Mine, in Tanzania, were taken. The ore had a maximum size of 200 mm and three samples were taken, (S-1, S-2, and S-3), which were representative of the three main ore types. Based on mineralogical reports from the mine, gold and silver occurs as inclusions in pyrite, inclusions in unaltered chalcopyrite, free grains, inclusions in quartz, and inclusions in bornite. Copper occurs primarily in the chalcocite-chalcopyrite replacement grains. It was reported by the mine that the three ores showed different milling behaviour. The mass and assay of the samples are presented in Table 1. For mineral liberation analyser (MLA), Bond, and grinding tests, representative sub-samples were prepared by stage crushing (laboratory jaw and cone crushers), followed by splitting (riffle splitter). Each sub-sample was then treated separately according to recommended protocols (see below).

3.2. Mineral liberation analysis

A ball mill equipped with a screen (1 mm in this case) was used for preparation of samples for mineral liberation studies. The mill discharge was sieved into five fractions, $−1+0.5$ mm, $−0.5+0.25$ mm, $−0.25+0.125$ mm, $−0.125+0.063$ mm and $−0.063$ mm. A broad sample from the mill discharge (i.e. 0–1 mm) was also included. The fractions were handed to automated mineralogical characterization. The measurements were performed at the Department of Mineralogy, TU Bergakademie Freiberg, using FEI MLA 600F system (Fandrich et al., 2007; Gu, 2003; Sandmann, 2015; Sandmann et al., 2014; Sandmann and Gutzmer, 2013). Feed samples to the system were prepared as polished grain mounts (Leißner et al., 2016; Sandmann, 2015; Sandmann and Gutzmer, 2013). Three sample splits were prepared from each fraction and used for MLA measurement. The mounts were carbon-coated prior to measurements in order to obtain an electrically conducting surface. Upon bombardment of an electron beam in a SEM, a mineral phase will backscatter electrons at an intensity defined by its average atomic number as well as releases X-rays characteristic of elements that are present. The measurement of Backscattered Electron (BSE) intensities allow the segmentation of mineral phases within a
single particle, while X-rays analysis of those phases allow the identification of each mineral phase. The analysis of mineral liberation data was performed by using MLA Dataview software (Fandrich et al., 2007).

3.3. The standard Bond’s grindability test

The standard Bond’s test is a batch type dry test where the mill is operated in a locked cycle. The charge is ground for a number of mill revolutions, sieved on the desired screen size to remove undersize and then replacing the undersize with new feed. The procedure is repeated until a constant mass ratio of 2.5 for oversize and undersize is achieved for three consecutive cycles (equilibrium condition). In the current work, standard Bond’s test protocols were adopted and the complete description of the test may be found elsewhere (Bond and Maxson, 1943; Magdalinić, 1989; Man, 2002). The test gives the standard work index (Wi), which is calculated from the formula (Bond, 1952):

\[
W_i = \frac{44.5}{P_{80}^{(0.23)} \cdot G_{80}^{(0.82)} \left( \frac{10}{P_{80}} - \frac{10}{G_{80}} \right)}
\]

where \(P_{80}\) and \(G_{80}\) are 80%-passing size (µm) of the mill feed and product, respectively, \(P_i\) is the test screen or sieve size used (µm), and \(G_{80}\) is the net grams of sieve undersize produced per mill revolution. Since Bond had used short tons to determine the work index, Eq. (9) has to be multiplied by a factor of 1.1 in order to convert to metric tonnes.

3.4. Grinding tests

Samples were prepared also to study the grinding kinetics following the protocol as proposed by previous investigators (Austin et al., 1984). Four mono-size fractions (−2 + 1 mm, −1 + 0.5 mm, −0.5 + 0.315 mm, and −0.315 + 0.2 mm) were used as feed to the grinding tests prepared by sieving of the cone crushe product. A 30.5 by 30.5 cm, standard Bond mill was used for the tests. The mill design and operating variables are presented in Table 2. Then the sample and steel balls were loaded to the mill and dry grinding tests were run for different time intervals (1, 2, 4 and 8 min). After each grinding interval, the whole mill charge was discharged, and a representative sample taken for sieve analysis. Later, the material retained on the sieves were combined with the rest of the mill discharge as feed for the next grinding interval.

4. Results and discussion

4.1. Ore mineralogy and liberation characteristics

Table 3 gives the modal mineralogy of the three ore samples as determined with an MLA. Pyrite-pyrrhotite is considered as the predominant valuable mineral in this case. Also based on mineralogical reports from the mine pyrite-pyrrhotite phase is considered as the major gold-bearing mineral for the investigated material and is the main phase covered in this section. The gangue phases comprise mainly of quartz, feldspar, muscovite, and biotite-chlorite. Epidote and fluorite are the minor mineral phases.

The primary grinding circuit at Buzwagi Gold Mine incorporates a gravity unit which concentrates portion of gold contained in sulphides. The final gold recovery from the gravity concentrate is accomplished by intensive cyanidation. Therefore, liberation of sulphides has significant implications on the overall performance of the plant. After fine grinding, gold is recovered through flotation and cyanidation processes. Fig. 1 shows an example of the mineral grain size distribution of the major mineral phases as determined by flotation. All samples show that pyrite-pyrrhotite has the coarsest grain size distribution followed by quartz and feldspars. Importantly, it must be recognised that all measurements for particle size and mineral grain size presented in this findings are based on the equivalent circle diameter (ECD) and the liberation distribution is based on mineral free surface. Most of mineral beneficiation processes such as leaching or flotation require at least a surface of the valuable mineral to be exposed (Wang et al., 2012; Wills and Finch, 2016). Therefore in this study liberation is based on the cumulative mass percentage of the mineral in the > 95% liberation class (i.e. free surface of the particle).

The liberation of pyrite-pyrrhotite based on size fractions is shown in Fig. 2 for five fractions measured for each sample. The order is uniform for the three samples whereby the degree of liberation decreases with increase in particle size. Except for the fraction 0–63 µm, in all other fractions the degree of liberation for S-1 and S-3 is closely to each other and is higher than that of S-1. Furthermore, the degree of liberation for S-2 in the fraction 500–1000 µm is the lowest. This is likely due to artefacts of poor particle statistics caused by the low grade of pyrite-pyrrhotite in the ore and the fine mineral grains. However, except for the fraction 500–1000 µm the degree of pyrite-pyrrhotite liberation for all size fractions in all samples is higher than 75%. This indicates that pyrite-pyrrhotite can be liberated at relatively very coarse

<table>
<thead>
<tr>
<th>Mineral</th>
<th>S-1</th>
<th>S-2</th>
<th>S-3</th>
</tr>
</thead>
<tbody>
<tr>
<td>Biotite-Chlorite</td>
<td>7.57</td>
<td>7.31</td>
<td>8.44</td>
</tr>
<tr>
<td>Epidote</td>
<td>1.63</td>
<td>3.53</td>
<td>1.60</td>
</tr>
<tr>
<td>Fluorite</td>
<td>1.27</td>
<td>1.87</td>
<td>0.72</td>
</tr>
<tr>
<td>Feldspars</td>
<td>22.52</td>
<td>19.25</td>
<td>27.20</td>
</tr>
<tr>
<td>Muscovite</td>
<td>17.62</td>
<td>15.70</td>
<td>14.91</td>
</tr>
<tr>
<td>Pyrite-pyrrhotite</td>
<td>6.54</td>
<td>5.58</td>
<td>3.26</td>
</tr>
<tr>
<td>Quartz</td>
<td>39.88</td>
<td>42.45</td>
<td>41.55</td>
</tr>
<tr>
<td>Other</td>
<td>2.97</td>
<td>4.31</td>
<td>2.31</td>
</tr>
<tr>
<td>Total</td>
<td>100</td>
<td>100</td>
<td>100</td>
</tr>
</tbody>
</table>

Fig. 1. Particle and mineral grain size distribution for S-1. These distributions are taken from MLA.
sizes for the material investigated. Table 4 and Table 5 provide supplementary information on particle statistics for illustration.

Table 4
Pyrite-pyrrhotite particle counts for the fractions measured.

<table>
<thead>
<tr>
<th>Sample</th>
<th>Size class (µm)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>0-63</td>
</tr>
<tr>
<td>S-1</td>
<td>5874</td>
</tr>
<tr>
<td>S-2</td>
<td>7107</td>
</tr>
<tr>
<td>S-3</td>
<td>4914</td>
</tr>
</tbody>
</table>

The errors included in Fig. 2 refer to the standard error based on average liberation from MLA measurement of three sample splits for each fraction. The error values increase slightly with particle size but in overall are within acceptable tolerance. The errors may be explained by the difference in number of particles or grains in the sample splits measured. In addition, the epoxy stirring and cutting techniques may result to inhomogeneous distribution of mineral grains in the epoxy, which can also contribute to differences in measured values.

Fig. 2 shows the relationship between pyrite-pyrrhotite cumulative liberation and class mean particle size. It is seen that all samples have liberation > 80%. From this result, it can be concluded that pyrite-pyrrhotite can be economically liberated at very coarse size of approximately 200–400 µm, where the cumulative liberation is greater than 85% for all samples. Hence the grinding costs could be saved as the material do not require milling too fine for the liberation of valuable mineral phase to be achieved.

Fig. 4 presents the enrichment characteristics of pyrite-pyrrhotite with particle size. This parameter was calculated as the ratio of the grade or content of pyrite-pyrrhotite in a given size fraction to that in the feed sample and illustrates that much of the mineral is enriched in the size range 250–500 µm. Sample S-2 has the highest enrichment in this size range followed by sample S-3 and sample S-1. Surprisingly, sample S-1, shows an enrichment also at the coarsest size fraction.

4.2. Work index

The work indices of ore samples were also determined and compared with values established during plant design as presented in Table 6. The work index for the Buzwagi ore deposit during design ranged from 14.5 to 16.5 kWh/t and currently ranges from 17.20 to 18.67 kWh/t. This increase of the work index indicates a change in the hardness of the ore as the pit deepened during the last 7 years. This has a negative impact on milling energy costs because as the production progresses, increased ore hardness increases the energy consumption per tonne of the milled ore. (Abbey et al., 2015) investigated the
variation in work indices with increasing pit depths for two gold fields in Ghana and found that the competence of the ore and work indices increase with increasing mining depth. As the mining progresses, new deposits may be discovered, which may have different characteristics from the plant design, a situation requiring re-assessment of the grindability and work indices of the different ores encountered. By re-assessment, better blends will be developed as well as prediction of appropriate tonnages for the existing ore types in order to be efficiently treated by the available plant design.

4.3. Determination of the specific rate of breakage

The breakage rates \( (S_1) \) were determined by plotting the fraction retained on the top size for each size interval against time according to Eq. (2). An example of the results is presented in Fig. 6 for sample S-1. It can be seen that the grinding process of all size fractions follows a first-order relation and values of \( S_1 \) could be obtained from the slope of the plots. The obtained \( S_1 \) values for each mono-size fraction for the three ore samples are shown in Table 7.

The variation of measured and estimated specific rates of breakage \( (S_1) \) with top size of each mono-size fraction is shown in Fig. 7. Estimated values were obtained by non-linear fitting of the experimental data into Eq. (4). For easier interpretation of the results, plots of estimated \( S_1 \) values have also been included in Fig. 7, extending slightly below and above the experimental range. It is seen that \( S_1 \) increases sharply until a certain point for all ore samples and then decreases slightly. This implies that there is an optimal feed size where the highest rate of breakage occurs (Bozkurt and Özgür, 2007; King, 2001; Toke et al., 2002; Umucu et al., 2015; Yan and Eaton, 1994). After passing that size, the values increase slowly and finally begin decreasing. This can be explained by the fact that larger particles are difficult to be ground by the media used and hence grinding efficiency decreases. Furthermore, S-2 and S-3 had higher decrease in specific rate of breakage at coarser sizes compared to S-1. This may be explained by the differences in hardness of the material as indicated by their work indices (i.e. Table 6).

The model parameters for the selection function, \( a, \alpha, \mu \) and \( \lambda \) were calculated from Eq. (4) by a non-linear regression technique using SOLVER function in Microsoft EXCEL. The method aims at searching for the best combination of the fitting parameters of a model by minimization of residual error between experimental and predicted values (Katubilwa and Moys, 2009; Sand and Subasinghe, 2004). The obtained model parameters for the three samples are presented in Table 8. It is seen that the first-order breakage rate constant \( a \) is higher for S-1 than S-2 and S-3. This indicates that sample S-1 had higher milling rates compared to S-2 and S-3.

The parameter \( \lambda \) determines how fast the specific rate of breakage,
decreases as the feed size increase. The comparison in terms of \(\lambda\) values in Table 8, indicates that sample S-2 had the highest rate of decrease in specific rate of breakage as feed size increases. This can be explained by the fact that S-2 was the hardest material compared to S-1 and S-3 (Table 6) and it should be expected that softer materials grind easily and faster compared to harder ones.

### 4.4. Determination of the breakage function

The values of cumulative breakage distribution function \(B_{i,j}\) were determined from size distributions at shorter grinding times (1 min in this case) using BII method, Eq. (6). The values of \(B_{i,j}\) for all size fractions and each material are plotted against the dimensionless relative size (Fig. 8) to determine if the values of \(B_{i,j}\) are normalizable. \(B_{i,j}\) values are said to be normalizable if the fraction which appears at sizes less than the starting size is independent of the starting size (Bozkurt and Özgür, 2007; Kelly and Spottiswood, 1990; Yan and Eaton, 1994). In terms of plots, the curves should superimpose upon each other, if \(B_{i,j}\) values are normalizable (Yan and Eaton, 1994). From Fig. 8, it can be seen that all samples show a typical normalized behaviour as the progeny distribution did not depend on the feed particle size and it follows that the value for the parameter \(\sigma\) was zero.

This allows fitting of \(B_{i,j}\) values into Eq. (7) by a non-linear regression technique and hence calculating the model parameters. In this case, the parameters \(\phi\), \(\beta\), and \(\gamma\) were estimated by fitting Eq. (7) to \(B_{i,j}\) values for each feed size using the SOLVER function in EXCEL. This is a non-linear optimization procedure which searches for the best combination of these parameters that minimizes the residual error between experimental and predicted \(B_{i,j}\) values (Katubilwa and Moys, 2009; Sand and Subasinghe, 2004). The average breakage function model parameters determined for all feed size classes for the three samples are given in Table 9.

The \(\gamma\) value characterizes the relative amount of fines produced from the breakage of top size material. A higher value implies that milling products are coarser. In general, softer materials would display lower values of \(\gamma\) as compared to harder materials (Makokha and Moys, 2006). By comparison in terms of \(\gamma\) values in Table 9, it is seen that more fines are produced in the milling of S-3. This can also be supported by the higher reduction ratios noted for this sample as reported in Fig. 5(b). In addition, \(\phi\) values are the same for all three samples, indicating that the breakage of the top size was the same for all samples.

Further, it is postulated that the first term in the breakage distribution function Eq. (7) describes the size distribution of the fine fraction in the progeny particles, resulting from localised fracture of large particles (cleavage breakage), while the second term describes the size distribution of the coarse fraction in the progeny particles, resulting from the main fracture of particles (shatter breakage) (Kelly and Spottiswood, 1990; King, 2001). Now, since \(\phi\) equals 1 (Table 6), the breakage distribution function for the tested material is reduced to a simple exponential function. The sophisticated model is strongly simplified and since only the first component of the function remain, it can be concluded that the breakage mechanism of the material was mainly by cleavage.

### 4.5. Observed correlations

The variation of breakage function fineness parameter \(\gamma\) with first-

### Table 8

<table>
<thead>
<tr>
<th>Material</th>
<th>(a)</th>
<th>(a)</th>
<th>(\mu)</th>
<th>(\lambda)</th>
</tr>
</thead>
<tbody>
<tr>
<td>S-1</td>
<td>0.53</td>
<td>1.50</td>
<td>0.77</td>
<td>1.65</td>
</tr>
<tr>
<td>S-2</td>
<td>0.50</td>
<td>1.50</td>
<td>0.84</td>
<td>1.80</td>
</tr>
<tr>
<td>S-3</td>
<td>0.41</td>
<td>1.50</td>
<td>0.93</td>
<td>1.78</td>
</tr>
</tbody>
</table>

### Table 9

<table>
<thead>
<tr>
<th>Material</th>
<th>(\gamma)</th>
<th>(\beta)</th>
<th>(\phi)</th>
</tr>
</thead>
<tbody>
<tr>
<td>S-1</td>
<td>0.82</td>
<td>2.50</td>
<td>1.00</td>
</tr>
<tr>
<td>S-2</td>
<td>0.83</td>
<td>2.50</td>
<td>1.00</td>
</tr>
<tr>
<td>S-3</td>
<td>0.72</td>
<td>2.50</td>
<td>1.00</td>
</tr>
</tbody>
</table>
order breakage rate constant, $a$ was investigated (refer Table 8 and Table 9) and is shown in Fig. 9. The values of $\gamma$ seem to satisfy a linear relationship with the first-order rate constant, $a$ with correlation coefficient of 0.90. It follows that $\gamma$ may be predicted from the first-order rate constant as follows:

$$\gamma = 0.92a + 0.35$$

(10)

The relationship can be further interpreted that more fines are expected for a slow breaking material. This is in agreement with what is displayed in Tables 8 and 9.

In order to evaluate the relationship between work index, $W_i$ and quartz content in the material, the quartz content values in Table 3 and the work index values in Table 6 have been plotted in Fig. 10. The work index seem to satisfy a linear relationship with quartz content of the ores with the correlation coefficient of 0.95 and the relationship can be expressed as follows:

$$W_i = 0.60\cdot \text{Quartz (wt. %)} - 6.49$$

(11)

The relationship could be explained by the fact that quartz is the hardest mineral compared to others in the material (hardness value of 7 based on Mohs hardness scale). Therefore, the work index for the Buzwagi Gold ore may be predicted based on known quartz content of the material.

In another step, experimental work indices are compared with predicted values from Eq. (11) (see Fig. 11). The prediction has been extended beyond experimental limits for better understanding of the trend. Based on the experimental range, the two sets of values are almost similar, indicating higher model prediction capability.

The dependence of the breakage deceleration rate parameter $\lambda$ on the material characteristic work index was also investigated as presented in Fig. 12. It can be seen that a strong linear correlation ($R^2 = 1$) is obtained between the two parameters. Therefore, the breakage deceleration rate can be predicted based on the work index of the material as follows:

$$\lambda = -0.10W_i - 0.11$$

(12)

Since work index is the measure of material hardness, this correlation implies that milling of harder ores will experience higher deceleration rates. Eq. (12) was used in the prediction of breakage deceleration rates and the outcome is compared with experimental values as presented in Fig. 13. Also the two sets of values are very close to each other indicating that a single model is adequate for prediction of breakage deceleration rates.

5. Conclusions

The breakage and mineral liberation properties of three low grade sulphide gold ore blends from Buzwagi Gold Mine have been investigated and the followings conclusions are drawn:

5.1. Mineral liberation studies

- Pyrite-pyrrhotite is the valuable mineral phase in the material with an abundance of 6.54, 5.58 and 3.2% for S-1, S-2 and S-3, respectively. Additionally, quartz, feldspar, muscovite and biotite-chlorite
are the main gangue phases in the material.
- The mineral grain size distribution of quartz and feldspars closely follows the particle size distribution of the host particles. Therefore, those minerals phases show a good liberation from the others.
- Pyrite-pyrrhotite mineral grain size distribution is coarser compared to most gangue minerals present. This indicates that the milling energy is more applied in grinding of the gangue mineral phases (e.g. quartz and feldspars).
- Pyrite-pyrrhotite liberation based on fractions is higher than 75% for all samples in fractions < 500 µm.
- Pyrite-pyrrhotite is liberated at relatively coarse size (i.e. approx. 200–400 µm), with > 85% cumulative liberation for each sample.
- Higher pyrite-pyrrhotite enrichment is displayed in the size range 250–500 µm, with S-2 having the highest enrichment followed by S-3 and S-1.
- Pyrite-pyrrhotite mineral grain size in S-3 is coarser than those in other samples.
- Based on reduction ratios, S-3 fractions used for MLA were finer than those for S-1 and S-2.
- The lower pyrite-pyrrhotite liberation levels achieved in S-2 could be contributed to material hardness.

5.2. Grindability and batch kinetic grinding

- The work index of the three ore blends are 17.2 kWh/t, 18.67 kWh/t and 18.47 kWh/t respectively, indicating differences in breakage properties between the three blends.
- The grinding process follows first-order for all feed size fractions in all three samples.
- Sample S-2 has higher value of breakage rate deceleration parameter, λ indicating that S-2 has the highest rate of decrease in specific rate of breakage (S) as feed size increases.
- Sample S-1 has the highest breakage rate constant than S-2 and S-3.
- The primary breakage distribution function, B0 show a typical normalized breakage behaviour for all three ore blends.
- Ore sample S-3 has the highest value of fineness parameter, γ indicating that milling of sample S-3 produced more fines than milling of S-1 and S-2.
- The material has the same breakage behaviour of the top size as indicated by same values of the breakage parameter ϕ.

5.3. Observed correlations

- The variation of breakage fineness parameter γ with first-order rate constant a satisfy a linear relationship with good correlation coefficient (R² = 0.90). The content of quartz in the material and the breakage rate deceleration parameter λ were each correlated with the work index. The two also gave linear relationship with ore work index with R² = 0.95 and R² = 1.0, respectively. These variations may be used to provide an estimate of respective breakage parameters for ore blends investigated.

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References
