Mineral liberation by high voltage pulses and conventional comminution with same specific energy levels

Eric Wang, Fengnian Shi *, Emmy Manlapig

The University of Queensland, Sustainable Minerals Institute, Julius Kruttschnitt Mineral Research Centre, Qld 4068, Australia

Article info

Article history:
Received 9 December 2010
Accepted 9 December 2011
Available online 14 January 2012

Keywords:
Mineral liberation
High voltage pulse breakage
Energy efficiency
Mineral recovery

Abstract

Comparative comminution between high voltage pulses and conventional grinding, at the same specific energy levels, shows that the electrical comminution generates a coarser product with significantly less fines than the mechanical breakage. However, minerals of interest in the electrical comminution product are better liberated than in the conventional comminution with an over 95% statistical significance. There is a potential to use less energy in the electrical comminution to generate the similar degree of mineral liberation as in the mechanical comminution. Distribution of the liberated minerals demonstrates that, in the electrical comminution product, a large percentage of the liberated minerals appear in size fractions coarser than 53 μm; while in the mechanical comminution product, the liberated minerals are accumulated in fine and very fine size fractions. Therefore there may be potential benefits in recovering the coarse liberated minerals in the electrical comminution product, prior to further grinding.

1. Introduction

Over the past decades great improvements have been made in the understanding of comminution equipment, simulation and control, but nevertheless, enhancement in liberation should also be a major aim. The release of valuable minerals from the rock matrix is an essential process in comminution. The ideal liberation method is by intergranular breakage; where breakage occurs along the grain boundaries of the mineral phases, allowing preservation of natural grain size and minimization of over-grinding. Liberation enhancement is sought for two main reasons. Firstly, if liberation is achieved without needing to grind particles to fine sizes, less energy is required. Secondly, over-grinding is very costly and produces fines that tend to interfere with the subsequent separation process, making the downstream processing both inefficient and more expensive.

The problems of the current liberation technology are: the high capital costs of the comminution process and its maintenance; a relatively high consumption of energy (Tromans, 2008); and the generation of excess of fines which are difficult to manage and often end up in the tailings. The insufficient liberation and over-grinding of minerals under the conventional methods, indicates that mechanical comminution is not an ideal liberation technology, rather, it is a technology of reducing the dimensions of the ore, with liberation occurring more as a side effect.

In existing comminution processes (such as SAG and ball mills), direct control over liberation does not exist due to random breakage in the mechanical process. The only control parameter is the product size distribution. By generating fine particles, which geometrically minimizes the percentage of composite fragments at the boundary with the rock matrices, an acceptable amount of liberated particles can be ensured for downstream separation. Increasing liberation without the over production of fines is difficult in mechanical comminution. The development of a specific selective liberation method, by preferable intergranular breakage of ore particles, becomes an important task for mineral processing.

Various attempts have been reported on novel methods of ore fragmentation, such as fragmentation by induction heating then followed by water cooling (Fitzgibbon and Veasey, 1990), and microwave treatment with the objective of mineral liberation enhancement (Kingman et al., 2000; Haque, 1999). The expectation behind these approaches was to induce intergranular breakage of aggregates, by understanding the differences in thermal properties and deformations of the constituents of the composites. The results showed that the liberation effect was small or non-existent, while the energy consumption required for the heating was high.

A unique liberation process, involving the disintegration of ores by high voltage pulses has been suggested (Andres, 1977; Anon, 1986), as a possible route by which mineral liberation properties can be enhanced beyond conventional breakage methods. The development of a plasma tubular structure in the electrical breakdown of complex dielectrics has an intergranular character. Selective liberation at coarse size ranges can be economical and suitable for use in separation techniques, such as leaching, gravity
separation or flotation. If enhanced liberation of ore can be achieved using high voltage pulses, then this could lead to appreciable economic benefits for mineral processing operations, since less material needs to be ground to a specific grind size. These benefits are primarily related to both direct and indirect energy savings.

A report on the benefits to diamond liberation using high voltage pulses (Andres, 1994), suggested that diamonds liberated by pulses had no single mechanical defect and were cleanly detached from kimberlitic matrices. Diamonds were recovered from the matrix cavities with undamaged patterns to their original crystallization. A small number of reports suggested that liberation properties of precious and base metal ores, such as emeralds, PGM, gold, chalcopyrite (Andres, 2010), can be enhanced through the use of high voltage pulse breakage. Chernet (2010) reported the use of high voltage pulses to release individual grains of gold, electron and other minerals of interest, preserving their original texture, shape and size for detailed study of their morphology, surface features, grain size and composition.

However, the reported energy consumption in the high voltage disintegration was high. A comparative assessment of the efficiency of liberation was conducted on a PGM (platinum group metal) ore in terms of energy consumption and recovery (Andres et al., 2002). The results suggested that energy consumption in the mechanical comminution of this ore was 50 kWh/t, while electrical disintegration consumed 90 kWh/t. The question was raised of what would happen to the mineral liberation if the two breakage routes were conducted at the same specific energy level.

In addition to the high energy consumption, the literature also suggests that better liberation results using high voltage pulses often show only marginal improvements. In general, there is a lack of statistical analysis of the comparative tests providing unambiguous evidence of the improved liberation. As a result, the ability of high voltage pulse breakage mechanisms to improve mineral liberation has remained an uncertainty to the mining industry.

In this study a detailed experimental program was undertaken to compare the mineral liberation results of two sulphide ores and one PGM ore. Each was subjected to two routes of treatment; one by high voltage pulses and the other by conventional mechanical breakage, both with the same specific energy input. Statistical analysis of the data was performed. The major outcomes of the investigation are presented below.

2. Experiment

2.1. Materials tested

Two sulphide ores and one PGM ore were selected for this study:

- Ore 1 sample was from a copper–gold mine located in New South Wales, Australia. The copper–gold mineralization occurs as quartz veins, sheeted quartz sulphide veins and stockworks. The gold occurs mainly as free grains in quartz or on the margins of sulphide grains. The principal copper sulphide minerals are chalcopyrite and bornite. The major silicate minerals are quartz, orthoclase, hornblende, chlorite, anorthoclase, and the non-silicate minerals include magnetite, calcite and anatase. Grain size distributions of the valuable mineral chalcopyrite show 10% passing 20 μm, 50% passing 80 μm and 90% passing 287 μm. Pyrite grain sizes appear finer than chalcopyrite at the P90 size: with 10% passing 24 μm, 50% passing 88 μm and 90% passing 160 μm.

- Ore 2 sample was from a low grade gold ore mine located in Western Australia. The primary gold-bearing sulphides in the waste rock and wall rock are pyrite and arsenopyrite.

Carbonates are extensive on the site, consisting predominantly of calcite and ankerite. The secondary mineral precipitates in localized areas include hexahydrite, halite, gypsum, basanite, siderite and hematite. Grain size distributions of the valuable mineral pyrite show 10% passing 61 μm, 50% passing 322 μm and 90% passing 608 μm.

- Ore 3 sample came from a platinum group metal (PGM) mine located in South Africa. The Platreef plays host to a number of metals, most notably the platinum group metals, as well as gold, silver, nickel, copper and cobalt. Pyrrhotite and pentlandite are the major minerals bearing the valuable metals, although some PGMs are associated with other minerals. The rock types include feldspatic pyroxenite, pyroxenite, parapyroxenite, serpentine and calc-silicate. Grain size distributions of the valuable mineral pyrrhotite show 10% passing 38 μm, 50% passing 355 μm and 90% passing 608 μm.

Table 1 gives the mineral composition of the three ore samples tested for the mineral liberation study, determined with an MLA (Mineral Liberation Analyser) instrument installed at the JKMRC (Julius Kruttschnitt Mineral Research Centre).

The amount of samples received at the JKMRC varied from 300 kg to 900 kg for each ore, collected manually by mine site personnel. The coarser (+45 mm) particles were crushed using a large jaw crushe at the JKMRC pilot plant. The whole sample from one site was combined and screened to obtain the required size fractions of 37.5–45 mm, 9.5–12.5 mm and 2.36–3.35 mm. The prepared samples (approximately 300–400 kg) were transported to Switzerland for high voltage pulse experiments using selFrag equipment. The 9.5–12.5 mm size was tested for the liberation study, while the other sizes were used for a pre-weakening study.

2.2. Equipment

The high voltage pulse breakage experiment was conducted with selFrag Lab equipment installed at selFrag AG based in Switzerland. The name of selFrag refers to selective fragmentation. The equipment consists of a high voltage power supply, a high voltage pulse generator, portable process vessels and a lifting table for easy loading and unloading of the process vessel. More details can be found in an earlier publication (Wang et al., 2011).

The SelFrag Lab is designed to process mineralogical and geological samples in the one kilogram range. The operation occurs in a wet batch processing mode. For the liberation tests, the operating parameters were set at 100–120 kV for the high voltage discharge, a 20–40 mm electrode gap, a 2 Hz pulse frequency, with the number of pulses being varied to achieve the desired discharge energy. When the predetermined voltage is reached, the energy of the pulse generator is discharged from the electrode through the solid sample to the grounded bottom (counter electrode) of the process vessel. This charging and discharging cycle repeats itself at a given frequency until the selected number of pulses has been reached.

Any variations in voltage, electrode gap or number of pulses will be reflected in the total pulse energy in the selFrag tests. The selFrag machine is equipped with a display system, from which the

<table>
<thead>
<tr>
<th>Mineral</th>
<th>Ore 1</th>
<th>Mineral</th>
<th>Ore 2</th>
<th>Mineral</th>
<th>Ore 3</th>
</tr>
</thead>
<tbody>
<tr>
<td>Chalcopyrite</td>
<td>2.2</td>
<td>Chalcopyrite</td>
<td>0.03</td>
<td>Pentlandite</td>
<td>0.2</td>
</tr>
<tr>
<td>Bornite</td>
<td>0.1</td>
<td>Pyrite</td>
<td>2.4</td>
<td>Pyrrhotite</td>
<td>0.3</td>
</tr>
<tr>
<td>Pyrite</td>
<td>0.4</td>
<td>Arsenopyrite</td>
<td>0.1</td>
<td>Chalcopyrite</td>
<td>0.2</td>
</tr>
<tr>
<td>Silicates</td>
<td>84.4</td>
<td>Silicates</td>
<td>83.7</td>
<td>Silicates</td>
<td>98.4</td>
</tr>
<tr>
<td>Non-silicates</td>
<td>8.2</td>
<td>Non-silicates</td>
<td>11.1</td>
<td>Non-silicates</td>
<td>0.8</td>
</tr>
<tr>
<td>Other</td>
<td>4.7</td>
<td>Other</td>
<td>2.7</td>
<td>Other</td>
<td>0.1</td>
</tr>
</tbody>
</table>
applied voltage, electrode gap, pulse frequency, number of pulses, and total pulse energy can be recorded. The total pulse energy is displayed in Joules. Since the specific energy in kWh/t has been widely used by mining companies around the world, the SI unit (International System of Units) Joule was converted to kWh, then divided by the sample mass in each batch test to obtain the specific energy in kWh/t. Note that the pulse energy is regarded as the net breakage energy, which does not account for the energy loss due to transform efficiency and during the machine idle period.

A laboratory rod mill installed at the JKMRC pilot plant was employed as a mechanical breakage device to treat the ore samples. Mono size fractions of the ore samples, similar to those treated by selfFrag, were ground in the rod mill for a given period of time. Both selfFrag and rod milling processes were in open circuit breakage. The net mill power consumption was calculated using a method reported by Shi et al. (2006). In this method, the mill no-load energy and gross energy at various mill revolutions were recorded using a precise energy metre. Fig. 1 shows the measured energy consumption data of the rod mill. A linear regression of the trends was performed to establish the relationship between the energy consumption and mill revolutions. The net energy was taken as the difference between the gross energy and the no-load energy. It was found that an ore breakage property, such as hardness, did not significantly affect the rod mill energy consumption. The mill energy consumption was dominated by the rod charge volume, the ore filling and the mill driving mechanism. At an identical rod charge, ore filling and driving system, the required specific energy for the rod milling to match that in the selfFrag test can be controlled by mill revolutions (i.e. the grind time). The mill revolutions can be converted to mill energy consumption using the calibrated energy relationships shown in Fig. 1. The specific energy is calculated as the mill energy consumption divided by the known sample mass. Note that the rod mill energy figures quoted in this paper indicate the net energy consumption, which, in this case, is much smaller than the gross energy consumption.

2.3. Mineral liberation measurement

Representative samples of the selfFrag and rod mill products were taken for quantitative mineralogical analysis using MLA. The XBSE (back-scattered electron) measurement collects a series of BSE images which undergo a number of steps of image processing. An X-ray spectrum is collected from each mineral grain identified in the pre-segmentation of the back-scattered electron image. The software used for the quantitative mineralogical analysis in this work was MLA SUITE 2008.

Two liberation indicators, the cumulative mass percentage of mineral in the >95% liberated classes and the Phase Specific Interfacial Surface Area (PSISA), were adopted to compare the liberation results between selfFrag breakage and conventional mechanical breakage.

2.3.1. The cumulative mass percentage of the minerals of interest in the >95% liberation classes

The MLA data gives the percentage mass of the minerals of interest at various liberation classes, either in an instantaneous or a cumulative form. It is believed that a particle with greater than 95% liberated mineral is easy to be recovered in a downstream process such as flotation. Hence the cumulative mass percentage of the mineral in the >95% liberation class was used to refer to mineral liberation in this study.

The liberated mineral deportment by size can be computed as a product of the mineral distribution by size and the mass percentage of the mineral in the >95% liberated classes, while the mineral distribution by size is computed as the product of modal abundance of the mineral and the mass retained on size, divided by the head abundance in the 0–3.35 mm product. This refers to the amount of >95% liberated minerals of interest in the size fraction.

2.3.2. Phase Specific Interfacial Surface Area (PSISA)

The phase specific interfacial surface area between the mineral phase of interest and the other minerals, can be used to gain an indication of the extent of valuable mineral interlocked with the other minerals (Fandrich et al., 1997). The surface area of the valuable mineral, that is in contact with other minerals (gangue) per unit volume, can be found from MLA results using Eq. (1). The smaller value of PSISA indicates a better liberation of the valuable mineral.

\[
\text{SvAB} = \frac{\text{SSA} (1 - \%\text{free})}{C0}\tag{1}
\]

where SvAB is the phase specific interfacial surface area of valuable mineral per unit volume of valuable mineral, or the interfacial surface area between phase A and phase B per unit volume of the particle. SSA is specific surface area of the valuable mineral or phase A, and %free is the fraction of the valuable mineral surface that is not possible to do MLA measurements on all sub-samples due to cost and time constraints. In the experimental program, one feed size (9.5–12.5 mm) from each ore was used for the liberation study, with the other feed sizes used for the pre-weakening study. For Ore 1, more detailed MLA measurements were performed on seven size fractions of the products generated by the two breakage routes, with two energy levels (E3 = 8.9 kWh/t and E6 = 21.9 kWh/t). For the other two ore samples, on the other hand, MLA measurements were only performed on two sizes of the products generated by the two routes with six energy levels. In addition, MLA measurements were conducted on a number of repeated tests to estimate the experimental error. In total, over 1000 kg of ore samples (including both selfFrag samples and mechanical breakage samples) were processed, with close to 400 MLA datasets available for the study.

2.4. Estimation of experimental errors

To estimate the MLA measurement errors associated with the liberation data, a method published by Leigh et al. (1993) was initially investigated. In this method, the measurement variance of a mineral was estimated from the total number of the mineral particles liberated and the total number of the associated mineral particles, as given in Eq. (2), and the standard deviation (SD) was determined from the measurement variance.
\[
\sqrt{\text{var}(Y)} \approx 1.12Y(1-Y) \left( \frac{1}{N_0} + \frac{1}{N_1} \right)
\]  

where \( Y \) is the cumulative liberation yield expressed as a fraction (0 ≤ \( Y \) ≤ 1) at composition C, \( N_0 \) is total number of locked particles with composition less than C, and \( N_1 \) is the total number of particles with composition of at least C.

On the other hand, of all the different particle sizes in the three ore samples tested by selFrag and the mechanical breakage mode, there were 170 sets (2 \times 85 pairs of duplicate tests) of MLA data available to estimate the experimental errors in the present study. The mean and SD of each pair of duplicate tests were determined, and the Coefficient of Variation (CoV) was calculated as a ratio of the SD to the mean value. The particle size-averaged CoV was multiplied by a measured MLA value, to give a SD associated with this measurement. Comparison of the SD values determined by the duplicate tests and by Eq. (2), shows that Eq. (2) gives smaller SD values. This is attributed to the fact that the SD estimated by Eq. (2) is for the errors associated with MLA, but the SD from the duplicate tests include all the errors in sample preparation, selFrag or mechanical breakage, product sizing and the MLA measurement. The SD estimated from the experiment is therefore considered more realistic, as it covers all the errors in the various experimental steps, and was consequently used in this study.

It was found through statistical analysis, that the difference in the means of the CoV between the selFrag and the mechanical breakage tests is insignificant. The electrical and mechanical comminution data were therefore combined to increase the sample size in estimating the mean of the CoV. It was also found that the CoV increases dramatically with particle size, which reflects the variation in the number of particles counted in the MLA measurements. For example, the size-averaged CoV in modal abundance was 0.025 for the 0.106–0.15 mm size fraction, in which approximately 20,000 particles were counted. The size-averaged CoV increased to 0.228 for the 1.18–1.7 mm size fraction, in which only 200–300 particles were counted. Therefore the particle size-dependent CoV was generated from the duplicate tests and was used to estimate the confidence intervals (CIs).

The 95% CI of CoV was computed from Eq. (3):

\[
\text{CI of CoV} = \text{CoV} \pm \frac{t \times \text{SD}}{\sqrt{n}}
\]

where \( t \) is the deviate of the normal distribution, which can be computed from the Student's \( t \)-distribution as a function of the probability and the degrees of freedom. For a small sample size where \( n = 2 \), \( t = 12.7 \) for 95% confidence, which reflects the very large uncertainties for the mean of a very small sample size. The deviate \( t \) decreases with sample size: when \( n \rightarrow \infty \) and \( t = 1.96 \) for the same 95% confidence.

Eq. (3) gives a lower limit and an upper limit of the confidence level. These limits provide a way to judge whether or not the difference in liberation criteria between the mechanical and electrical comminution methods is statistically significant. For a more rigorous assessment, the upper limit of the 95% confidence interval was employed in the comparison, i.e. the difference in the liberation criteria between the two methods must exceed a larger interval to show statistical significance.

3. Results

The results presented in this section are mainly from Ore 1. Results from the other two ores are used to validate the trends observed from the Ore 1 sample.

3.1. Particle size distribution

Despite the fact that breakage occurred with the two comminution methods at the same specific energy levels, particle size distributions appeared completely different. Fig. 2 displays the particle size distribution curves of the products comminuted mechanically by a rod mill, and electrically by the selFrag, for particles in a mono feed size 9.5–12.5 mm. The comparison was made at two identical specific energy levels: 8.9 kWh/t and 21.9 kWh/t respectively. At both energy levels, the mechanical breakage generated significantly finer products than the electrical breakage. The energy efficiency (in terms of kWh per ton of net production of particles less than a specified size), was lower in the electrical breakage (i.e. higher kWh required) than in the conventional mechanical breakage, if the electrical comminution was used solely for particle size reduction.

Fig. 2 also indicates that the electrical comminution generated remarkably less fines than the mechanical breakage with identical specific energy. As the specific energy increases from 8.9 kWh/t to 21.9 kWh/t, the amount of material less than 0.106 mm increased from 3.5% to 8.6% by the electrical comminution; whereas, it changed from 17.7% to 47.7% by the mechanical comminution, with more than five times more fines (−0.106 mm) being generated by the latter method. Thus electrical comminution may also find applications in processes where the minimization of fines generation is critical.

3.2. Mineral liberation data processing

In order to investigate the size by size mineral liberation trend in the whole product size range, more detailed mineralogical analysis was carried out on seven fragment size fractions for Ore 1. Two minerals in Ore 1 were used as indicators of the valuable minerals: chalcopyrite, a major copper-bearing mineral; and pyrite, a major gold-bearing mineral.

The seven MLA size fractions for Ore 1 included: 2.36–3.35 mm, 1.18–1.7 mm, 0.6–0.85 mm, 0.106–0.15 mm, 0.053–0.075 mm, 0.01–0.053 mm and −0.01 mm. As it was difficult for MLA to achieve a good result in the finest size fraction (−10 µm), the mineralogical data for this size fraction was converted from wet chemical analyses together with the MLA data in the coarser particle size fractions, using a standard method developed at the JKMRC.

MLA measurements were not performed on every consecutive product size fraction due to cost and time constraints. To reconstruct the liberation data for the whole product size range, a data interpolation technique was adopted. The solid symbols in Fig. 3

![Fig. 2. Particle size distribution curves of the products comminuted mechanically and electrically for Ore 1 particles in a mono feed size of 9.5–12.5 mm.](image-url)
were the measured MLA data, covering both coarse and fine ends to avoid extrapolation. The three size fractions at the fine end were all measured, as these size fractions were close to the grain sizes and often associated with large variations in the liberation data. A cubic spline function embedded in an Excel spreadsheet was employed to interpolate the MLA data (the hollow symbols) in the five missing size fractions from the measured data: 1.7–2.36 mm, 0.85–1.18 mm, 0.3–0.6 mm, 0.15–0.3 mm and 0.075–0.106 mm making up a complete size range from 0 to 3.35 mm. The MLA data in the measured size fractions were kept unchanged in the cubic spline interpolation process.

### 3.3. The >95% liberated minerals deportment

Figs. 4 and 5 present the >95% liberated minerals deportment by mass for Ore 1 particles (in the feed size 9.5–12.5 mm), comminuted mechanically and electrically with specific energy levels of 8.9 kWh/t and 21.9 kWh/t respectively. The size-by-size mineral deportment data were combined to form four size fractions in the figures for a clearer demonstration of the trend. The two minerals of interest, chalcopyrite and pyrite from Ore 1, were investigated. The error bars are the 95% confidence intervals, estimated from the upper limit of the SD and are subject to more rigorous assessment.

Figs. 4 and 5 illustrate that there is little >95% liberated chalcopyrite and pyrite in the coarse product size fractions (+0.30 mm). This is largely attributed to the nature of the mineral grain size. As described in Section 2.1, 90% of the chalcopyrite grain size in Ore 1 is smaller than 0.287 mm, and for pyrite the 90% passing size is 0.16 mm. It would therefore not expect to see much liberated chalcopyrite and pyrite minerals in the +0.3 mm size fractions.

Comparison of the >95% liberated minerals deportation between the mechanical and electrical comminution in the 0.053–0.30 mm size range, shows that the selfFrag product contains a higher amount of liberated chalcopyrite and pyrite minerals than the rod mill product. In the 0.053 mm size, the trend reverses, with higher >95% liberated mineral deportation associated with the mechanical comminution. As the product size decreases, the associated SDs decrease, and the difference between the two comminution methods becomes pronounced, with an over 95% significance. Paired t-tests on the difference of the >95% liberated chalcopyrite and pyrite, between the two breakage routes with the same specific energies, also indicate that the difference is real, with an over 95% significance for both energy levels.

### 3.4. Distribution of the >95% liberated minerals in product

Since the +3.35 mm size fraction is unlikely to contain >95% liberated minerals of interest due to the mineral grain size nature, it would be interesting to compare the distribution of the >95% liberated minerals in the 0–3.35 mm products generated by the two comminution methods. For each size fraction, the mass deportment of >95% liberated minerals were re-calculated to 100% for the 0–3.35 mm product. The size-by-size distributions of the >95% liberated minerals were then combined to form four size fractions for a clearer demonstration of the trend. Table 2 presents the distributions of the two minerals of interest.

The electrical product contained significantly more >95% liberated chalcopyrite and pyrite in the size fractions coarser than 0.053 mm, while the mechanical breakage generated more >95% liberated minerals in the size fractions finer than 0.053 mm, particularly at high specific energies. For example, at 21.9 kWh/t specific energy, the mechanical breakage product exhibits a split of 33–67% of the >95% liberated chalcopyrite in the +0.053 mm and the 0.053 mm size fractions. In comparison, the electrical product has a split of 78–22%. Both chalcopyrite and pyrite minerals liberated with the two specific energy levels show the similar trend.

The data provide strong evidence of better liberation of the valuable minerals at coarser sizes by the electrical comminution.

### 3.5. PSISA

The PSISA refers to the extent of chalcopyrite or pyrite interlocked with other minerals – the smaller value of PSISA indicating...
a better liberation of the valuable minerals. Fig. 6 presents the PSI-SA of chalcopyrite and pyrite comminuted mechanically (ME) and electrically (SF) with a specific energy level of 8.9 kWh/t (E6) from feed 9.5–12.5 mm, and the error bars indicating the 95% CI.

In general, the majority of the PSISA of chalcopyrite and pyrite is smaller in the electrical product than in the mechanical product, indicating that the chalcopyrite and pyrite minerals produced by the electrical comminution are less interlocked with other minerals. Since the large measurement errors are associated with the particles coarser than 0.15 mm, the differences in the coarse sizes did not reach a statistically significant level. However, at the less than 0.15 mm size fractions, the majority of the data suggests that the difference is significant. The PSISA data from the Ore 1 products, comminuted with 21.9 kWh/t specific energy, show the similar trend. The PSISA data agree with the >95% liberated mineral deportment trends presented in Sections 3.3 and 3.4, and confirm that the electrical comminution produced better liberated minerals for Ore 1 sample.

![Fig. 5. The mass deportment of >95% liberated chalcopyrite and pyrite by size, for Ore 1 progeny particles comminuted mechanically (ME) and electrically (SF) with a specific energy level of 21.9 kWh/t (E6) from feed 9.5–12.5 mm, and the error bars indicating the 95% CI.](image1)

![Fig. 6. Phase Specific Interfacial Surface Area (PSISA) of chalcopyrite (CCP) and pyrite (PYR), comminuted mechanically (ME) and electrically (SF) at a specific energy 8.9 kWh/t, with the error bars indicating the 95% CI.](image2)

<table>
<thead>
<tr>
<th>Size (mm)</th>
<th>ME E3 (%)</th>
<th>SF E3 (%)</th>
<th>ME E6 (%)</th>
<th>SF E6 (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>+0.3</td>
<td>4.88</td>
<td>6.89</td>
<td>1.80</td>
<td>15.23</td>
</tr>
<tr>
<td>0.106–0.3</td>
<td>19.63</td>
<td>28.34</td>
<td>13.05</td>
<td>32.22</td>
</tr>
<tr>
<td>0.053–0.106</td>
<td>24.87</td>
<td>25.03</td>
<td>18.07</td>
<td>30.86</td>
</tr>
<tr>
<td>Sum &gt; 0.053</td>
<td>49.38</td>
<td>60.26</td>
<td>32.92</td>
<td>78.31</td>
</tr>
<tr>
<td>Sum &lt; 0.053</td>
<td>50.62</td>
<td>39.74</td>
<td>23.33</td>
<td>41.99</td>
</tr>
<tr>
<td>Total</td>
<td>100.00</td>
<td>100.00</td>
<td>100.00</td>
<td>100.00</td>
</tr>
</tbody>
</table>
3.6. Validation

Three ore samples were tested in the mineral liberation study. In order to validate the trends observed in Ore 1, (tested with 8.9 kWh/t and 21.9 kWh/t), the test results for Ore 1 at two additional energy levels (5 kWh/t and 10 kWh/t), and the results for Ores 2 and 3 with the same energy levels, are presented in this section. One mineral was selected as the liberation indicator for each ore sample in this section: chalcopyrite for Ore 1, as it is the major copper-bearing mineral; and pyrite for Ore 2, as the major gold-bearing mineral; and pyrrhotite as one of the major PGM associated minerals for Ore 3. The product sizes 0.106–0.15 mm were analyzed for liberation by MLA. Fig. 7 displays the >95% liberated mineral deportment for the three ore samples, in a feed size of 9.5–12.5 mm, comminuted mechanically and electrically with specific energy levels of around 5 kWh/t and 10 kWh/t respectively. The error bars indicate the 95% CI.

The data demonstrated that the electrical comminution generated significantly more liberated chalcopyrite and pyrrhotite minerals from the Ore 1 and Ore 3 samples. For the Ore 2 sample, the deportment of the liberated pyrite mineral generated by the electrical comminution at 5 kWh/t was not as large as the mechanical breakage product, but it became significantly higher at the 10 kWh/t specific energy level.

The graphs of PSISA of the valuable minerals in the three ore samples are presented in Fig. 8. Similarly, the chalcopyrite mineral in Ore 1 and the pyrrhotite in Ore 3, show less interlocking with other minerals in the samples comminuted by high voltage pulses than with those comminuted mechanically, with the difference being over 95% significant. For the pyrite mineral in the Ore 2 sample, the electrical comminution produced smaller PSISAs than the mechanical breakage product, but the difference did not reach the 95% significant level.

The additional data have further confirmed the better liberation trend by the electrical comminution method, in all three ore samples tested, at various energy levels.

4. Discussion

4.1. Energy in relation to size reduction

The effect of energy on electrical comminution is one major scope of this investigation. Fig. 2 illustrates that the electrical breakage generated much coarser products than the conventional mechanical breakage at identical specific energy levels. A product fineness indicator, $t_{10}$, was employed as a size reduction parameter. The parameter $t_{10}$ is defined as product cumulative percentage passing 1/10th of the parent particle size (Napier-Munn et al.,...
The larger $t_{10}$ value indicates a finer product. For example, for a feed size 9.5–12.5 mm as tested in the Ore 1 sample, the geometrical mean size is 10.9 mm. The cumulative percent passing 1.09 mm (1/10th of 10.9 mm) in the selfFrag product at 8.9 kWh/t is 24.8%, which generated a $t_{10} = 24.8\%$, while the mechanical breakage product generated a $t_{10} = 66.0\%$ with the same specific energy (Fig. 2).

A relationship between the size reduction parameter $t_{10}$ and the specific energy was established from the Ore 1 test results. This relationship was applied to estimate the specific energy required for selfFrag to achieve the same $t_{10}$ in the conventional breakage product. The relationship predicts that 2.1 times the energy is required by electrical comminution to achieve the same $t_{10}$ as in conventional mechanical breakage. The estimated energy consumption agrees with the data in the literature. Andres et al. (2002) reported a specific energy of 50 kWh/t used in mechanical breakage and 90 kWh/t in electrical disintegration. Finkelstein and Shuloyakov (1996) showed that to disintegrate a cube with 40 mm side down to 0–2 mm size required work amounting to 20 kWh/t by the electrical breakage, as compared to 3–10 kWh/t in the full scale mechanical breakage machines and 15–17 kWh/t in the laboratory scale mechanical facilities.

The data suggest that the electrical breakage is not an ideal comminution method if used merely for size reduction. However, the recent research has provided evidence that high voltage pulse comminution may be employed for particle pre-weakening, rather than size reduction, to improve energy efficiency in downstream processing (Wang et al., 2011). Mineral liberation is another area of potential application for electrical comminution, by which better liberation of valuable minerals can be achieved with the same specific energy despite the coarser product.

### 4.2. Energy required for similar degree of liberation

As discussed above, the electrical comminution generated coarser product but with better liberated minerals than mechanical breakage. Since the specific energy used to generate the liberated minerals in each test is available, it is possible to compare the energy requirement for similar degree of liberation. Table 3 presents the mass% of >95% liberated chalcopyrite in the 0–3.35 mm product and the energy required for electrical comminution to achieve the same degree of liberation as mechanical comminution. The mass% of >95% liberated chalcopyrite is calculated as a product of the head mineral modal abundance and the head deportment of >95% liberated chalcopyrite in the 0–3.35 mm product.

Table 3 illustrates that at 8.9 kWh/t specific energy (E3) level, mechanical comminution produced 0.3% of >95% liberated chalcopyrite in the 0–3.35 mm product of Ore 1. To achieve the same degree of chalcopyrite liberation, electrical comminution only requires 4.8 kWh/t, representing 46% energy saving.

At 21.9 kWh/t specific energy (E6) level, however, electrical comminution would require more energy to generate the same mass% of the liberated chalcopyrite than the mechanical comminution. This was attributed to the fact that at high specific energy, mechanical comminution generated a large percentage of fines (−10 μm) that were smaller than chalcopyrite grain sizes (refer to Section 2.1), and contained a large amount of liberated minerals.

In contrast, electrical comminution did not significantly increase the mass% of liberated chalcopyrite at the higher specific energy level.

The data suggest that in electrical comminution the amount of liberated mineral produced and the amount of specific energy applied is not in a linear relationship. There is an optimal range of specific energy application in electrical comminution, which will give the minimum amount of energy requirement to generate a unit mass of liberated minerals of interest.

Note that in this study, the experiment was conducted in an open circuit with the same specific energy levels for both mechanical comminution and electrical comminution, with a focus on investigation of mineral liberation in the −3.35 mm product. It remains a future study on the energy efficiency and mineral liberation relationship for a closed circuit to break all +3.35 mm particles to the −3.35 mm size fractions.

### 4.3. Mineral liberation distribution

The present study has found that a large amount of liberated minerals are distributed in the coarser size fractions in the electrical comminution product. This is contrary to what is found in the mechanical breakage product, in which large amounts of the liberated minerals are cumulated in the fine or very fine size fractions. Taking Ore 1 as an example: in the mechanical breakage product, there was 67% liberated (in the >95% liberation classes) chalcopyrite mineral accumulated in the −53 μm size fractions. In comparison, 78% of the liberated chalcopyrite was found in the larger than 53 μm size fractions from the electrical comminution. It is once again emphasized that the two products were generated with the same specific energy levels.

Such a distribution of the liberated mineral in the electrical comminution product may find a potential benefit for the recovery of valuable liberated minerals at coarse sizes and at an early processing stage. Since it is well known that fines have an adverse influence on recovery, if the liberated minerals can be recovered before further grinding to less than 53 μm (depending on the nature of the grain size of the mineral of interest), economic benefits and energy savings may be realised.

### 5. Conclusion

A detailed experimental program was undertaken to compare the mineral liberation results of two sulphide ores and one PGM ore subjected to two routes of treatment by high voltage pulses and conventional mechanical breakage. One distinguishing feature of this study is that the two routes of comminution were conducted at identical specific energy levels. Extensive mineral liberation analysis was conducted to provide sufficient data for statistical analysis of the comparative results.

The data demonstrated that at the same specific energy level, the electrical comminution generated a much coarser product with significantly less fines than were generated in the mechanical breakage. Despite the coarser product, the percentage of the >95% liberated minerals of interest in the electrical comminution product was higher than that in the mechanical breakage product, generated with the similar specific energy, and the difference was statistically over 95% significant. This trend has been examined and confirmed by various liberation criteria, and with different minerals of interest.

The experimental results show that at low specific energy input level (E3), electrical comminution requires 46% less energy than the mechanical comminution to produce the similar degree of liberated chalcopyrite. However, at high specific energy input level
(E6), the electrical comminution requires more energy. There may exist an optimal range of specific energy application in electrical comminution, which will give the minimum amount of energy requirement to generate a unit mass of liberated minerals of interest for a specific ore.

It was found that in the mechanical comminution product, a large percentage of the >95% liberated minerals was accumulated in the fine and very fine size fractions (~53 μm). In contrast, a large percentage of the liberated minerals appeared in the larger than 53 μm sizes in the electrical comminution product. Therefore there are potential benefits to the recovery of the liberated minerals at coarse particle sizes prior to further grinding.

Acknowledgement

The authors would like to acknowledge financial support from the Australian Research Council Linkage Scheme (AMSRI – LP0667828), AMIRA International, the State Governments of South Australia and Victoria, and the sponsors of AMIRA International Project P924: BHP/Billiton, Rio Tinto, Orica Explosives, Anglo Platinum, Xstrata Technology, Freeport McMoran and AREVA NC. SelFrag AG kindly provided the high voltage pulse testing facility for the experiment. Assistance provided by Dr. Alexander Weh of selFrag in the experimental work and by Dr. Elaine Wightman of the JKMRC in MLA measurement is appreciated. Useful discussion on statistical analysis was held with Prof. Tim Napier-Munn of the JKMRC. The initial work from Dr. Mike Daniel in arranging the preliminary experiment with selFrag is acknowledged. Support from Anglo Research, Newcrest, Newmont and Xstrata mining companies in sample collection is gratefully acknowledged.

References


