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Technical Note
Effect of ball size change on the performance of grinding and flotation circuits

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ABSTRACT

At the Sarcheshmeh copper mine 41,000t of ore per day with an average grade of 0.9% Cu is ground to 70% passing 75 microns. Grinding is effected in 8 parallel lines of 8x5m ball mills in a closed circuit with cyclones. The make-up balls are 80-mm forged alloy steel balls and the average ball consumption is 750g/t of ore ground. In order to optimize ball size distribution inside the mill, based on the previous investigation instead of using only 80mm make-up balls a combination of 80 and 60mm (75% to 25% by weight) ball charge was used in one of the mills. To evaluate the performance of three ball types namely forged alloy steel balls (A), ductile cast iron (C), high chromium cast iron (D) and also a combined (80 and 60mm) charge of forged alloy steel balls (B), four identical parallel mills were charged accordingly. These four mills were sampled during a period of one year. The amount of material finer than 75 microns of cyclone overflows was selected as a criterion for the performance evaluation. It was observed that in a mill with the combined make-up balls the amount of fine produced was 4% higher than the other mills. The consumption of ball types A, C and D were 730, 710 and 534g/t, respectively. The results of laboratory flotation tests showed that due to an improvement in the production of fines an increase of 1.4% in the overall copper recovery is obtainable. Due to the promising results, the new make-up ball regime was implemented in all mills in the plant. © 2004 SDU. All rights reserved.

Keywords: Ball size; Flotation; Grinding; Liberation degree

1. INTRODUCTION

The Sarcheshmeh copper ore body in Southeast of Iran in Kerman province which may rank as third or fourth largest in the world contains 1 billion tonnes averaging 0.90% copper and 0.03% molybdenum. It is located southeast of Iran and has been processing 41,000t/d of ore since 1982. The mine produces 140,000 tonnes of copper and 3,500 tonnes of molybednite concentrate per year. After three stages of crushing, the ore with a F_{80} of 12.7mm is fed to 8 parallel ball mills in closed circuit with cyclones to produce a product 70% finer than 75 microns. The concentrate of the first stage of flotation is reground and the tailing constitutes the main component of the plant final tailings. A concentrate with an average grade of 31% Cu is obtained after cleaning and re-cleaning stages. Total recovery of the plant varies between 84 and 88% depending on the operating conditions and ore type.

At the Sarcheshmeh complex, 52% of total electrical energy is consumed in the concentration plant. Grinding section uses 72% of the energy consumed in the plant which in turn is only 5% of the total grinding cost. Thus, an improvement in the grinding operation not only is economically important but actually has a direct effect on the performance of the downstream separation process i.e., flotation.

In the grinding circuit 80-mm forged alloy steel balls are used with an average consumption of 750g/t of ore milled. This translates to more than $100,000 per month for each mill, about 80% of the total grinding costs. The relatively high cost of grinding balls and their critical effect on the grinding efficiency initiated a project into a detailed study of their quality and size distribution. The significant effect of ball size on the grinding efficiency has been mentioned in the literature (Gupta and Kapur, 1974; Austin *et al.*, 1976;

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Austin and Klimpel, 1985; Mankosa, *et al.*, 1986). Only in one instance the size distribution of the entire charge of a 2.75m by 3.05m ball mill was measured and reported (Vermeulen and Howat, 1989). It is believed for the first time at the Sarcheshmeh copper mine a mill of much larger size was emptied and the ball size distribution was measured. A detailed study on this subject was reported elsewhere (Banisi *et al.*, 2000).

2. BACKGROUND

2.1. A new make-up ball regime

The ball size distribution was obtained from the examination of entire charge (280,534 balls with a mass of 305,180kg). The measured ball size distribution was a valuable piece of information that could be used to change the obtained ball size distribution to the desired one. It was decided to add two make-up ball sizes (as opposed to the usual practice of adding a single size make-up balls at the plant) to alleviate the problem of not having enough small balls.

The proposed make-up balls routine was implemented for one of the ball mills and its efficiency in terms of ball consumption and grinding performance was compared with the other parallel mills for a period of one year.

2.2. Methods of grinding and flotation circuits evaluation

The main criterion for the evaluation of the grinding circuit was chosen to be the amount of $-75\mu m$ in the cyclone overflow. Since this factor was directly related to the performance of the flotation circuit and it was also acceptable to the plant personnel. In addition, the amount of $-75\mu m$ of ball mill discharges and 80% passing size of the cyclone overflows (P_{80}) were also measured for each sampling campaign. The ball consumption of all mills was recorded regularly over the sampling period.

In order to evaluate the effect of ball size change on the flotation response, various kinetics models were examined (Ahmed and Jameson, 1989; Ek, 1992; Yuan *et al.*, 1996). The comparison of models based on the sum of squares of errors indicated that the simple first order kinetics model was the appropriate one. The model relates the rate constant (k) and the ultimate cumulative recovery (R_∞) in the following form:

$$R = R_\infty [1 - \exp(-kt)] \tag{1}$$

where t is flotation time and R is cumulative recovery at time t.

In addition to comparison of k and R_∞ of some samples, the cumulative recovery at a specific time obtained in a laboratory flotation cell for all samples was also compared.

3. EXPERIMENTAL

To evaluate the primary grinding circuit, four of eight parallel ball mills with their cyclones were selected for sampling. The make-up ball characteristics of each mill are shown in Table 1. In ball mill No.2, 75% 80mm and 25% 60mm balls were charged. The average weight of the balls in each mill is about 300 tons.

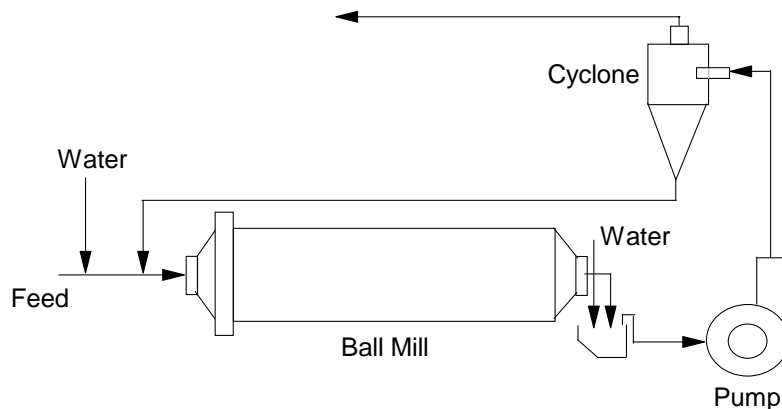The grinding circuit of the plant is shown in Fig. 1.

Figure 1. The grinding circuit of the Sarcheshmeh copper mine

Simultaneous samples from fresh feed, mill discharge, underflow and overflow of the cyclones of mills under study were taken. The average retention time of material in the grinding circuit is approximately 8 minutes. A sampling period of 2 hours was chosen in which every 15 minutes one sampling increment was taken. Before and during sampling, the circuit was checked for the steady-state operation. This was achieved by screening (at 75μm) individual 10-minute samples from the ball mill discharge. If the obtained value for the percent passing was within ±5%, it was assumed that the steady-state operation had been reached and maintained. The required weight of samples for each point was calculated using Gy's equation (1979): 25kg for the fresh feed, 10kg for the ball mill discharge and cyclone underflow and 2kg for the cyclone overflow. The fresh feed, cyclone underflow and overflow samples were screened using 19mm, 2380μm, and 295μm respectively down to 38μm. All samples below 295μm were wet screened with a 38-micron screen. The sub-sieve analysis was carried out using the Warman Cyclosizer. To alleviate the effect of other operating variables on the performance of the circuit at the time of survey, the sampling of all mills was carried out on the same day. To determine the accuracy of the results, the samples were split in two and all samples were analyzed separately. For all tests, the circulating load was calculated from size analysis data using a least square method.

Table 1
The make-up ball type and size used in four parallel mills

Ball Mill No.	Make-up Ball	
	Type	Size (mm)
1	forged alloy steel balls (A)	80
2	forged alloy steel balls (B)	80 and 60
3	ductile cast iron (C)	80
4	high chromium cast iron (D)	80

3.1. Flotation circuit

In order to perform the laboratory flotation tests, samples with a one-liter sampler were taken from cyclone overflow every 15 minutes. The percent solids of combined sample was adjusted using the plant water to keep the chemistry of the pulp constant. All flotation tests were performed soon after sample preparation to avoid oxidation. Table 2 lists the reagents used for flotation tests which were performed in a 5-liter cell. The value of pH was set to be 11.2 for all tests. All reagents were used based on the plant practice.

Table 2
Reagents used in flotation tests

Reagent	Reagent type	Amount (g/t)
Collector	R407[1]	15
	Z11[2]	10
Frother	A65[3]	15
	MIBC[4]	15

1: Mercaptobenzothiazole; 2: Xanthate; 3: Polypropylene glycol; 4: Methyl isoboutyl carbinol

The conditioning time was three minutes and the impeller speed was 1200rpm. Based on the retention time of the rougher cells determined by a tracer technique, in the laboratory flotation cell the concentrate was collected for eight minutes. The feed, concentrate and tailing samples were filtered, dried and assayed.
The flotation kinetics tests were performed in a 10-liter cell and the concentrate samples were collected in the intervals of 1, 2, 3, 5, 7, 9, 11, 15, 20 and 30 minutes. Samples were then dried, screened, and all size fractions were fractions assayed for copper. All chemical and size analysis data were adjusted by a mass balancing software (Movazen 2.1) (Yarahmadi and Banisi, 1998).

4. RESULTS AND DISCUSSION

4.1. Grinding circuit

The results of 24 series over a period of one year in terms of the amount of −75μm in the cyclone overflow (i.e., flotation feed) of four ball mills under the investigation are shown in Fig. 2.
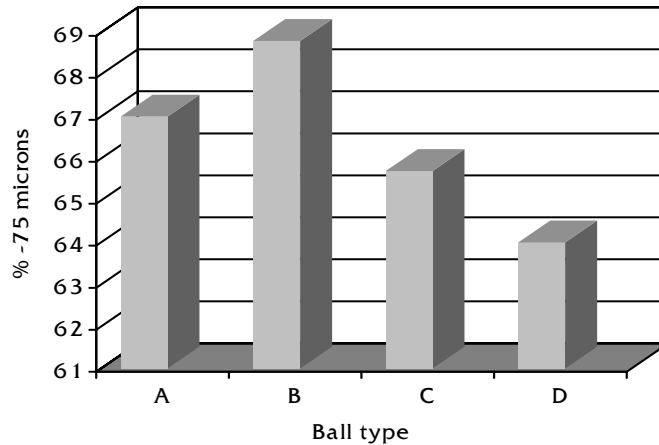
Figure 2. The amount of −75μm in the overflow of four ball mills (A: forged alloy steel balls (80mm); B: forged alloy steel balls (80 and 60mm); C: ductile cast iron (80mm); D: high chromium cast iron (80mm))

The results show that the combined make-up ball regime is superior to other regimes tested. An increase of 1.8-4% in the amount of -75μm in the cyclone overflow with the new regime is clear. It is expected that an increase of this quantity results in a significant improvement in the flotation circuit performance due to a higher degree of liberation of valuable minerals.

In order to determine the change in all particle size ranges, samples of cyclone overflow from mills with type A and type B make-up balls were screened. The samples were screened down to 38μm screen (Fig. 3).

There is a shift in the size distribution curve of the mill with the combined make-up balls which indicates the production of finer particles in all sizes. The amount of this improvement is about 2-4%.
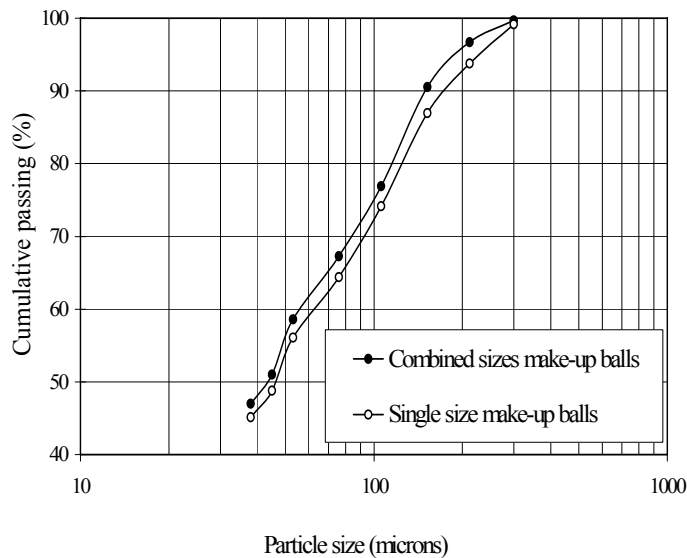

Figure 3. Size distribution of cyclone overflow of two differently charged ball mills

The 80% passing size of cyclone overflow (P_{80}) of four mills along with the percent finer than 75μm of mill discharges were also determined for a combined samples over a period of six months. Table 3 summaries the results.

The mill with combined make-up balls (80 and 60mm) produces between 1.7 and 3.5% more fines (-75μm) than other mills. A decrease of 5.5% (from 120μm to 113.4μm) in P_{80} when using combined make-up balls clearly shows a significant improvement in the grinding efficiency.

Table 3
A summary of results of 24 tests obtained from four ball mills

Mill No. (Ball type)	% -75μm – Mill discharge	P_{80} (μm) – Cyclone overflow
1 (A)	26	120
2 (B)	27.7	113.4
3 (C)	24.67	124
4 (D)	23.5	132.2

The ball consumption for ball types A, B, C and D was 705, 730, 710, and 534g/t respectively. Since the cost of ball types C and D was 18 and 123 % higher than type A (or B) the former ball types were not found to be economical. A rather small increase in the ball consumption of ball type B compared to A was compensated by a significant improvement in the grinding circuit efficiency.

4.2. Flotation

To evaluate the effect of make-up ball change on the rougher flotation circuit, samples of cyclone overflow (i.e., floatation feed) of two differently charged mills were subjected to laboratory flotation tests. Over the sampling period, 31 samples simultaneously were taken. It is worth mentioning that all necessary measures were taken to keep the operating parameters for the two mills constant before and during sampling campaign. The results of comparative flotation tests with a flotation time of 8 minutes are shown in Fig. 4.
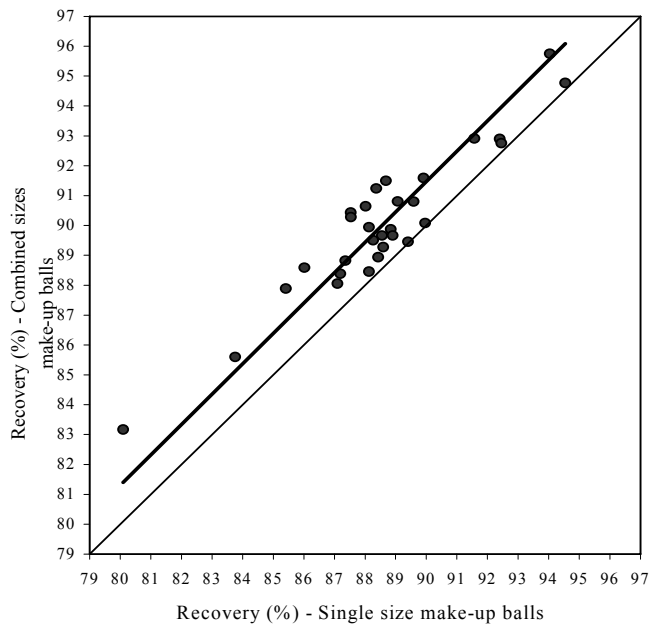

Figure 4. Flotation results of mills charged with single and combined make-up balls

Recovery values obtained from the samples of combined make-up balls are significantly higher than the mill with the single size make-up balls. A linear fit of the data is also shown in Fig. 4. On average an increase of 1.4% in recovery when combined make-up balls are used is evident. The 95% confidence intervals of the recovery of types A and B were 88.60±7.94 % and 90.06±5.98%, respectively.

To further investigate the flotation response of samples taken from the two parallel grinding circuits, the flotation tests were performed over a 30-minute period. Fig. 5 presents the results in terms of cumulative recovery versus flotation time.

The first order kinetics model fits for both cases are also shown in Fig. 5. An overall improvement in the flotation response in all times is evident when combined make-up balls are used. Table 4 gives the first order kinetics model parameters for two tests.
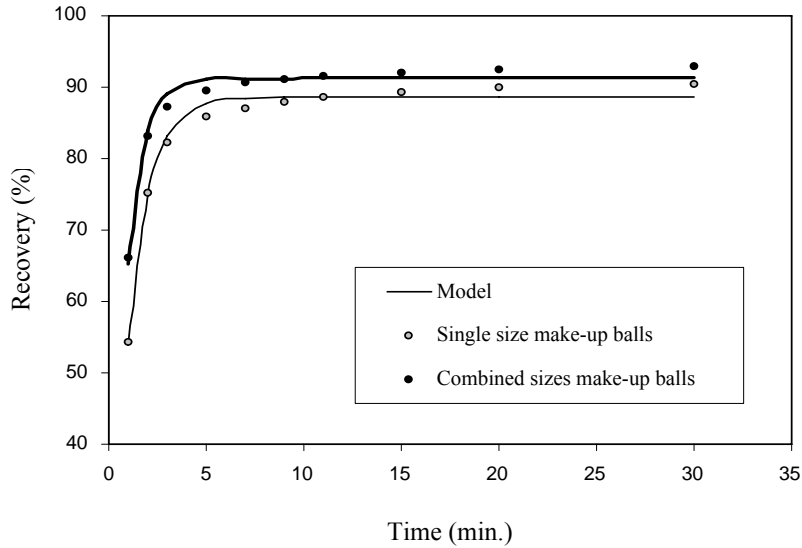
Figure 5. Cumulative recovery versus time for two mills with different make-up balls

Table 4
Values of the first order kinetics model parameters for two different flotation tests

Condition	k (1/min)	R_∞ (%)
Single size make-up balls	0.9	88.75
Combined sizes make-up balls	1.3	91.25

The kinetic tests indicated even a larger difference (2.5%) in the recovery of two different make-up ball regimes when floated for 30 minutes.

In order to evaluate the size-by-size recovery, for both cases each size fraction was assayed. The mass balanced data are shown in Fig. 6. The highest recovery was found for particles between 10 and 50μm and recovery dropped below 10μm and above 50μm. This trend has been reported by various investigators (Trahar and Warren, 1976; Trahar, 1981; Drzymala, 1994; Heiskanen and Kallioinen, 1995). It is interesting to see that the trend of both curves for single and combined make-up balls, as expected, is similar.
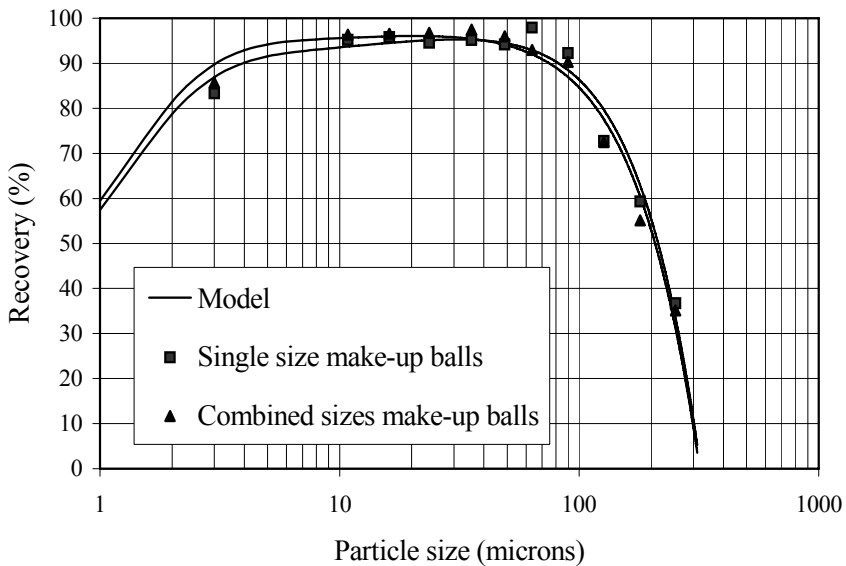

Figure 6. The recovery versus particle size relationship for single and combined make-up balls

The concentrates collected at each time intervals were screened and assayed to obtain the flotation rate constant for each size class (Fig. 7).
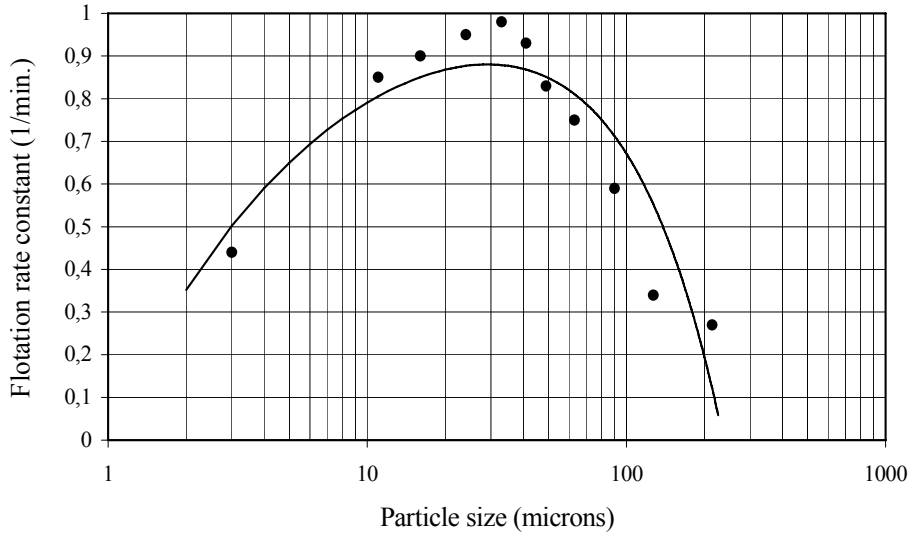

Figure 7. Variation of flotation rate constant with particle size.

An empirical model was also fitted to the data:

$$k = 1.296 - 0.005D - 1.380/D^{0.5} \tag{2}$$

where D is the particle size (>1µm) and k is the rate constant (1/min). Particles about 20-40µm had the highest rate of flotation. It can be concluded that the larger the amount of particles above 70µm, the higher the reduction in the copper recovery. It is then expected that the higher recovery achieved when combined make-up balls used results from the production of a lower amount of large (>70µm) particles.

The effect of grinding on the degree of liberation has been the subject of many studies; but due to the complex nature of the problem, an accurate expression has not been yet proposed (Machado Leite, 1992; King and Schneider, 2000). In order to study the effect of grinding on the degree of liberation of three copper containing minerals namely chalcopyrite, chalcocite and covellite a few polished samples were prepared. Mineralogical study was performed by the mineral counting method using a polarized microscope. Fig. 8 presents variation of degree of liberation of three minerals in terms of particle size.
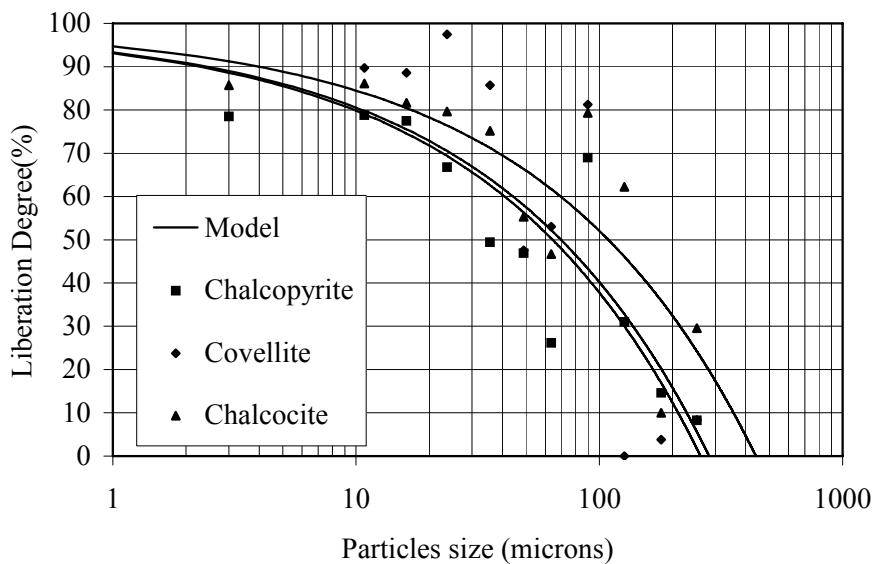

Figure 8. Liberation degree versus particle size for three copper bearing minerals

Although due to the errors involved in this method of mineralogical studies the data points are scattered, the trend is clear. For all three minerals by increasing the particle size the degree of liberation decreases especially for particles above 50µm. An empirical model was also fitted to the data:

$$L = 100 - a D^{0.5} \tag{3}$$

where

L: liberation degree; a: constant; D: particle size (µm)

Table 5 shows the value of the model parameters for three minerals.

Table 5
The value of constant (a) in the liberation model

Mineral	A
Chalcopyrite	6.15
Chalcocite	5.90
Covellite	4.74

5. CONCLUSIONS

- ➤ The effect of moving from a single size to two-size make-up balls was evaluated in an 8 by 5m mill. It was observed that the mill charged with the two-size make-up balls could produce up to 4% more fines (<75µm) than the other mills.
- ➤ The consumption of three ball types namely forged alloy steel balls (A), ductile cast iron (C), and high chromium cast iron (D) was determined in four identical parallel mills over a period of one year. The consumption for ball types A, C and D was 705, 710, and 534g/t, respectively.
- ➤ It was observed that the P_{80} (80% passing size) of product of the grinding line with the mill charged with combined make-up balls (80 and 60mm) decreased by 5.5% (from 120µm to 113.4µm) compared to the other lines.
- ➤ Based on the laboratory flotation tests, on average an increase of 1.4% (95% confidence) in recovery can be obtained if the combined make-up balls are used.
- ➤ The kinetic tests indicated an increase both in the rate constant (0.9 vs. 1.3 1/min) and the maximum theoretical recovery (88.75 vs. 91.25%) when combined make-up balls used.
- ➤ The size-by-size recovery analysis revealed that particles between 10 and 50 have the highest recovery and beyond these points the recovery decreases significantly.
- ➤ The highest rate constant obtained from flotation tests was found to be for particles about 20-40µm.
- ➤ Mineralogical study of polished samples showed that the degree of liberation of chalcopyrite, chalcocite and covellite minerals decreases by increasing particle size especially above 50µm.
- ➤ Due to the promising results, the two-size make-up balls regime was implemented in all eight parallel ball mills of the plant.

ACKNOWLEDGEMENTS

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