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Effect of regrinding and pulp aeration on the flotation of chalcopyrite in chalcopyrite/pyrite mixtures

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ABSTRACT

In flotation, regrinding is used to increase the liberation of value minerals, such as chalcopyrite, from gangue minerals. As a result of the significant decrease in particle size following regrinding, flotation efficiency is generally worst. In this study it is shown that the flotation recovery (and rate) of chalcopyrite in chalcopyrite–pyrite mineral mixtures, which is lower after regrinding, could be partially restored with aeration. The increase in Cp flotation upon aeration was attributed to the creation of a more favourable environment for xanthate adsorption and dixanthogen formation whereas the decrease in Cp flotation after prolonged aeration is due to its increased oxidation as a result of galvanic interactions with pyrite upon regrinding. The amount of aeration required for maximum flotation decreased monotonically with increasing feed grind particle size. To rationalize the observed chalcopyrite flotation trends, a simple model which involves competitive adsorption of xanthate and oxygen on chalcopyrite and pyrite surface sites and underpinning oxygen consumption and mineral oxidation effect is proposed.

1. Introduction

As a result of the discovery of new ore bodies with declining grades that require fine grinding for increased liberation and efficient mineral separation, the flotation of fine particles has become increasingly important in recent times. At present, the greatest challenge which confronts flotation separation in the mineral industry is related to decreased efficiency in processing fine particles. Generally, fine hydrophobic particles, <10 μm in size, display low flotation rate and poor recovery at conventional residence time in plan flotation machines [1,2]. The poor flotation behaviour of hydrophobic particles <10 μm has been attributed to their low collision efficiency with rising bubbles due to their low mass/momentum [3]. Increasing the degree of turbulence/impeller speed and/or residence time during flotation has been found to increase the recovery of fine particles [4–6]. Fine grinding is however often required in practice. The key driving force for fine grinding is enhanced liberation of values and gangue minerals. The creation of new surfaces during fine grinding however, leads to the exposure of more active sites to alterations of the particle surface chemical properties which may influence significantly the flotation pulp chemistry (e.g. dissolved oxygen concentration and pulp oxidation potential). This may, in turn, significantly affect collector adsorption and oxidation, ultimately controlling the mineral particle flotation performance [7,8]. It is well known that a minimum amount of oxygen in the pulp (reflecting low Eh) is required for xanthate collectors to adsorb and, therefore, promote sulphide mineral flotation while an excess amount of oxygen (high Eh) leads to marked surface oxidation which has a detrimental effect [9–11]. Furthermore, galvanic interactions between sulphide mineral particles and the grinding media, with different electrocatalytic abilities for the reduction of oxygen, may induce changes in pulp and mineral surface chemistry which can affect the flotation performance [12–14]. Due to the complexity of copper ores, in which chalcopyrite and chalcocite are often finely interlocked with pyrite, to improve the value minerals liberation and copper flotation recovery and grade, regrinding to very fine particle size (sometimes <20 or 15 μm as practiced at the Prominent Hill copper mine and Telfer) is commonly performed. However, in most cases an increase in valuable mineral recovery is not observed upon regrinding, or at least not to the expected extent, suggesting that particle size and liberation are not the only key influential factors in determining the flotation performance [15].

This paper particularly investigates the effect of regrinding and aeration on the flotation performance of chalcopyrite and pyrite mineral mixtures. To rationalize the flotation behaviour, the amount of air/oxygen reacting with chalcopyrite and pyrite and the resulting particle products formed is measured. Finally, a simple reaction mechanism which takes into account oxygen and xanthate concentrations and mineral surface area is proposed to qualitatively describe the observed chalcopyrite flotation.

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2. Experimental

2.1. Materials and reagents

Single minerals of chalcopyrite (Cp) and pyrite (Py) obtained from Mannum Minerals and Peruvian Mine, respectively, were used for this study. The details of the chemical compositions of the minerals are reported in Table 1. Complementary XRD analysis showed that the chalcopyrite sample contained 6.3 wt.% pyrite while the pyrite contained 10% impurities of virtually no copper sulphide. A two stage grinding procedure (primary grinding to d_{80} of 105 μm and secondary grinding to d_{80} of 10, 20, and 30 μm) was used to produce the feed for this study. Fig. 1 shows the particle size distribution curves for the 10, 20, 30 and 105 μm grind size. The primary grinding was achieved in a closed Galigher laboratory mill with stainless steel grinding media while that of the secondary grinding was achieved in an attritioning mill (IsaMill) with ceramic beads (Ø3.5 mm) (Xstrata Technology, Australia) as the grinding media. Sodium isopropyl xanthate (SIPX) collector and polypropylene glycol (Dowfroth 250; AR grade) frother were used, both obtained from Cytec (Stamford, USA). Analytical grade lime and demineralised water were used throughout the experiment for pH control.

2.2. Mineral grinding and flotation

In each test, 200 g of mineral (Cp alone or 100 g each of Cp and Py in mixtures) was added to 300 cm^3 of demineralised water and known amount of lime in a Galigher laboratory mill to produce a pulp with a particle size distribution (PSD) of 80% passing 105 μm at pH 10. The mill product was then conditioned in a 1.5 dm^3 flotation cell (Agitair LA-500R) at pH 10.5 with xanthate collector (200 g SIPX per ton of Cp in the feed) for 2 min. The conditioned pulp was reground in a 1 dm^3 laboratory IsaMill (attrition mill) to produce a PSD of 80% passing 30, 20 or 10 μm. Product from the attrition mill was transferred back to the 1.5 dm^3 flotation cell and conditioned without or with air (air flow rate = 3 dm^3/min) for different times. Known amounts of SIPX additional collector (Table 2) and 25 g/t Dowfroth frother were added and conditioned for 2 and 1 min, respectively. In order to ensure the same collector coverage, the collector dosage was scaled to the mineral (BET) specific [35] surface area (Table 2) after IsaMilling. Impeller speed and pH during conditioning and flotation in the cell were maintained at 1000 rpm and 10.5, respectively for all experiments. Four flotation concentrates were collected after cumulative times of 1, 3, 6 and 10 min at an air flow rate of 2.5 dm^3/min. The flotation froth was scraped every 10 s. The dry masses of the four concentrates together with their tails were measured and the samples assayed for their elemental compositions. Flotation rate constant (k) and maximum flotation recovery ($\%R_{max}$) were calculated by fitting the cumulative recovery (%R) versus its corresponding flotation time (t) of each concentrate with a first order rate equation, $\%R = \%R_{max}(1 - e^{-kt})$. Mineral recovery by true flotation and entrainment was calculated using the method developed by Ross [16].

2.3. Oxygen demand test

A number of techniques are currently used to determine the oxygen demand of flotation pulps and are all based on the method described by Spira and Rosenblum [17]. The method generally requires aeration/oxygenation of a freshly ground pulp to a particular oxygen level, after which aeration/oxygenation is ceased and the decay in oxygen level monitored as a function of time. This method is adequate for the laboratory, although it does not replicate the conditions in most processing plants. This is because in most processing plants the pulp is not aerated/oxygenated to a particular oxygen value or content but rather it is oxygenated or aerated for a constant amount of time. Another fundamental problem with the method is the fact that it is open to the atmosphere. As a result, it is likely that some amount of oxygen is lost to the atmosphere rather than being consumed by the pulp. With the DO demand experiment used in this study, all these short falls have been addressed.

Table 1
Chemical composition of Cp and Py samples used in the study.

Mineral	Elements (mass %)								
	Zn	Fe	S	Ca	Cu	Si	Pb	Mn	Mg
Cp	0.01	28.9	29.0	1.76	26.4	4.2	0.34	0.02	0.68
Py	0.18	44.9	53.5	0.44	0.13	0.7	0.02	0.02	0.08

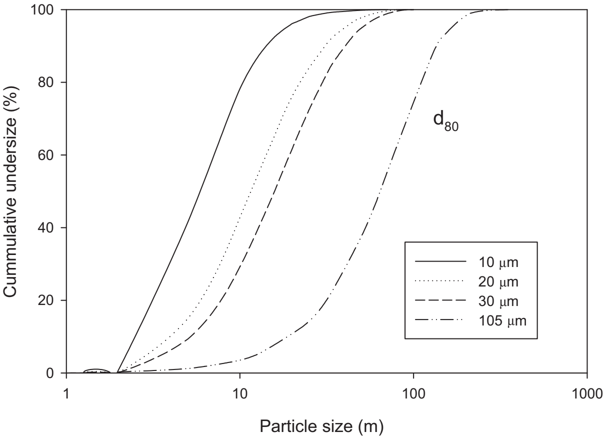

Fig. 1. Particle size distribution for primary and secondary grinding.

2.3.1. Procedure

The details of the apparatus used for the oxygen demand test have been described previously [18]. In each experiment, the IsaMill discharge pulp (either Cp or Py alone or Cp/Py mixture) was transferred into an airtight container and stirred for 1 min to obtain a homogeneous pulp after which the initial readings for pulp potential, DO, and pH were recorded. The pulp pH was adjusted to 10.5 and then air purged at a rate of 3 dm^3/min for 3 min, after which the air supply was cut off for 5 min. This cycle of air on and off was repeated several times until the oxygen concentration in the vessel reached equilibrium. The pulp pH, DO and pulp potential values were logged continuously at time intervals of 10 s. The oxygen demand rate constant (Kla) for Cp, Py and Cp/Py mixtures was calculated after the air supply was cut off for all the cycles (Eq. (1)).

$$DO = DO_o \times e^{-(Kla \times t)} \tag{1}$$

where DO and DO_o are the dissolved oxygen concentrations at time t and t = 0, respectively.

Dissolved oxygen (DO) (YSI membrane electrode), potential (Pt–Ag/AgCl electrode) and pH (glass combination) probes connected to a multi-metre (TPS 90-FLMV, TPS Pty. Ltd.) were used to measure the DO concentration, pulp potential and pH, respectively. Standard buffer solutions of pH 7 and pH 10 were used to calibrate the pH electrode.

Table 2
Collector concentration after secondary grinding scaled to mineral (BET) specific surface area for the various grind size (50 wt.% Py).

d_{80} (μm)	BET surface area (m^2/g)	SIPX (g/t) added after secondary grinding
10	1.76	33.1
20	1.33	25.0
30	0.76	14.3