



Decoupling rheology and slime coatings effect on the natural flotability of chalcopyrite in a clay-rich flotation pulp



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ABSTRACT

Clay minerals are a commonly occurring gangue in mineral beneficiation operations and occur in a variety of ore deposits, including porphyry copper ores. The presence of clays in mineral beneficiation circuits can cause a wide range of different problems, including that of slime coatings on valuable minerals and elevated viscosities of flotation pulps. The aim of this work was to decouple and estimate the relative magnitudes of the slime coatings and pulp rheology effects on the natural floatability of chalcopyrite in a model mineral system. It was found that the two phenomena can be partially isolated and that they both have a detrimental effect on flotation performance. It was also found that the level to which these two phenomena affect chalcopyrite recovery is of a similar order of magnitude.

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1. Introduction

Clay minerals are a commonly occurring gangue in mineral beneficiation operations and occur in a variety of ore deposits, including porphyry copper ores (Bulatovic et al., 1998; Petruk, 2000), low grade nickel ores (Senior and Thomas, 2005), platinum group metal deposits (Visser et al., 1994) and diamond bearing ores (Boshoff et al., 2007). In some rare cases, the proportion of clay minerals within such ores can reach as high as 80% (Burdukova et al., 2008). Not only are high proportions of clay minerals in concentrates penalized by smelters, but the presence of these minerals in flotation and comminution pulps causes a wide array of problems for the flotation operations, including higher reagent consumption, poorer selectivity and impeded flotation kinetics (Connelly, 2011).

Clay minerals fall within the phyllosilicate mineral family (Deer et al., 1978). Phyllosilicate minerals are so named because they typically display a platy/leafy habit, *fylo* meaning “leaf” in Greek. There are a large number of different types of phyllosilicates in existence, with a variety of different properties. The faces and edges of phyllosilicates carry different electrical charges, making these particles electrostatically anisotropic. The faces (or basal planes) tend to be negatively charged, while the edges carry a charge that alters from positive to negative as a function of pH (Van Olphen, 1951).

The presence of clays in mineral beneficiation circuits can cause a wide range of different problems. The soft nature of these miner-

als induces the production of a large amount of slimes. The presence of slimes has a negative impact on slurry rheology, with detrimental effects on both flotation and comminution. As rheology describes flow properties of fluids it has a significant impact on flotation cell hydrodynamics. Schubert (2008), argues that low pulp viscosity causes a reduction in turbulence damping, which increases the probability of particle/bubble attachment, while also increasing the probability of detachment. This was verified by Xu et al. (2011), who found that increased pulp viscosity resulted in an increased stability of the bubble/particle aggregates, thus decreasing detachment probability. The effect of increased pulp viscosity was also shown to have a detrimental effect on gas dispersion, through limiting the turbulent region of flotation cells to a small “turbulent cavern” surrounding the impeller (Bakker et al., 2009; Shabalala et al., 2011).

Another detrimental and well known effect of very fine particles on flotation is commonly referred by the generic term “slime coatings”. In this phenomenon, relatively coarse valuable particles become either completely or partially coated by a layer of hydrophilic slimes, thereby rendering the valuables hydrophilic and inhibiting collector adsorption. The mechanism responsible for such coatings is widely believed to be the electrostatic attraction between oppositely charged mineral particles (Arnold and Aplan, 1986; Del Giudice, 1934; Edwards et al., 1980; Sutherland and Wark, 1955; Trahar, 1981). Please note that for the sake of brevity, this electrostatic interaction will be referred to as “slimes coatings” throughout the manuscript. However, very little is known on the relative magnitude of the impacts that both of these effects have on flotation recovery.

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The objective of this work was to study both the effect of slime coatings and pulp rheology on the recovery of chalcopyrite from a clay rich flotation pulp, with the aim of decoupling the two effects and attempting to quantify their relative magnitudes. This was achieved by studying a model batch flotation system in the absence of reagents. The pulp rheology was manipulated by changing the proportion of clay material in the gangue component of the flotation pulp, while the slime coatings were manipulated by altering the pulp pH. The rheological properties of the flotation pulps were monitored by measuring the flow behaviour of flotation tailings. An attempt was made to quantify the effect of slime coatings by measuring the interaction between clay minerals and chalcopyrite using zeta potential distribution measurements.

2. Materials and methods

2.1. Mineral samples

The chalcopyrite (CuFeS_2) was handpicked high-grade mineral from Mt Lyell, Tasmania. The stoichiometric composition of chalcopyrite is 34.6% Cu, 30.4% Fe and 35.0% S and the chemical analysis indicated that the sample was very high grade and contained negligible impurities.

The kaolinite ($\text{Al}_2\text{Si}_2\text{O}_5(\text{OH})_4$) mineral sample was obtained from Wards Scientific. Chemical analysis indicated that the kaolinite sample contained 22.1% Al, 17.7% Si and 0.94% Ti as the major elements present while quantitative phase analysis (QPA) based on X-ray diffraction (XRD) analysis indicated that the major components of the sample were kaolinite (~72% with an additional ~8% dickite) and gibbsite (~17%).

The quartz (SiO_2) mineral was a sample obtained from Creswick Quartz, near Smeaton, Victoria. Chemical analysis indicated that the quartz contained 46.3% Si, equivalent to 99% SiO_2 based on a stoichiometric composition of 46.7% Si for quartz, and the levels of impurity elements detected in the sample were all less than 0.03% confirming the high quality of the quartz sample.

2.2. Reagents

The flotation tests were conducted in the absence of collector. The frother was a commercial grade polypropylene glycol (Cytac Aero froth 65) prepared as 0.25% w/v solution in distilled water. The flotation gas was bottled zero-grade synthetic air and the water used throughout the tests was distilled water. Dilute solutions of sodium hydroxide and sulphuric acid were added to set and maintain the pH during flotation tests.

2.3. Flotation procedure

The composition of the flotation pulp was made with pure minerals, with 2.5% valuable mineral and 97.5% gangue. The valuable mineral phase was represented by chalcopyrite, which corresponded to a Cu head grade of 0.87%. The gangue phase was represented by a mixture of quartz and kaolinite at a variety of ratios, ranging from 0% to 100% quartz (100% to 0% kaolinite).

In order to ensure that the particle size distribution of chalcopyrite remained consistent throughout the test series, the chalcopyrite was ground separately in a 1.4 dm³ ceramic mill for 7 min. The quartz and kaolinite (at a variety of proportions) were ground in a 5.3 dm³ ceramic grinding mill for 30 min. The ground samples were transferred into a batch cell and made up to a volume of circa 2750 ml for each test to give a flotation pulp density of 29 wt% solids.

Flotation tests were performed in a 3 dm³ modified Denver cell (Guy, 1992) in which the impeller was driven from below to allow

the whole surface of the froth to be scraped with a paddle at a constant depth and rate. The froth was scraped every 2 s for the first minute of flotation and every 5 s thereafter. The cell was fitted with a rubber diaphragm, sight tube and electronic sensor for automatic detection and control of the pulp level. A Radiometer PHM61 laboratory pH meter and pHC2401 combined pH electrode was used to measure the pH. Radiometer TTT80 titrators and ABU80 burettes were used to add acid/base to set and maintain the pH.

Frother solution was added at a rate of 2 ml min⁻¹ continually commencing 2 min before flotation and for the duration of the test. The flotation gas was zero grade synthetic air used at a flow rate of 8 L min⁻¹. The flotation procedures used for the mineral-quartz mixtures followed a standard procedure developed by CSIRO. The ground pulp was transferred to the flotation cell and the water level was raised to a preset level. The pH was then adjusted to the test value. The addition of frother commenced and after 2 min concentrate collection started. Concentrates were taken after 0.5, 1, 2, 4 and 8 min. At the end of each test, while the pulp was still being mixed in the flotation cell, two 50–60 ml sub-samples of the flotation tailings was collected using a polypropylene syringe for rheology measurements.

The flotation rate coefficients were calculated using the following expression, where $f(t)$ is the fraction of chalcopyrite remaining unrecovered after time t , η is the fraction of non-floating chalcopyrite, ϕ is the fraction of fast-floating chalcopyrite which has a flotation coefficient of k_f and k_s is the rate coefficient of the slow-floating chalcopyrite, as shown in Eq. (1) (Frew and Trahar, 1982):

$$f(t) = \eta + \phi \cdot e^{(-k_f t)} + (1 - \phi - \eta) \cdot e^{(-k_s t)}. \quad (1)$$

2.4. Particle size measurement

The particle size distribution of the flotation tailings was estimated by light scattering, using a Malvern Mastersizer 2000. Sample aliquots were placed in an ultrasonic bath for 1 min before measurement. Each of the two tailings samples was sub-sampled twice, with the particle size distribution of each sub-sample measured in triplicate. The aggregate of twelve resulting measurements was used as an average particle size distribution curve.

2.5. Zeta potential measurements

Zeta potential measurements were performed using the CAD Zeta compact instrument. A CCD camera captured the movement of charged particles within an electric field. This video footage was then analysed for particle trajectories and output a range of particle mobilities. These mobilities were converted to zeta potentials using the Smoluchovski Equation (Dukhin and Derjaguin, 1976).

The mineral suspensions were conditioned at pH 11 for 2 h and sonicated for 20 min. After sonication, the samples were allowed to stand for further 60 min. A small sample (0.1 ml) of the supernatant was taken and diluted to 500 ml. The resultant dilute suspension was examined with the sample viewer of the zeta meter instrument. Further 0.1 ml supernatant aliquots were added to the 500 ml dilute suspension until the concentration of particles (as seen in the instrument sample viewer) was sufficient for taking good measurements.

The extremely dilute nature of the supernatant suspension meant that its exact particle size distribution could not be experimentally determined by light scattering, due to insufficient obscuration values. However a calculation of Stokes velocity (Bird et al., 2007) required for the particles to remain in suspension after 60 min in 15 cm of liquid, determined that the average size of chalcopyrite and kaolinite particles had to be smaller than 2.5 and 3.5 μm respectively.

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