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Effect of oxidation potential and zinc sulphate on the separation of chalcopyrite from pyrite

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Abstract

The effects of oxidation potential (Eh) and zinc sulphate on the separation of chalcopyrite from pyrite were investigated at pH 9.0. The flotation recovery of these minerals is Eh dependent with maximum separation obtained at 275 mV SHE. Zinc sulphate addition improved this mineral separation at an Eh value of 275 mV by selectively depressing pyrite flotation. A different result was obtained at lower Eh values where zinc sulphate addition improved chalcopyrite flotation but had no or little effect on pyrite flotation. These opposite effects of zinc sulphate on mineral flotation were reconciled by examining the surface species of these minerals. The selective depression of pyrite flotation by zinc sulphate was also confirmed in the flotation of two copper ores.

Keywords: flotation; chalcopyrite; pyrite; depression; zinc sulphate; oxidation potential

1. Introduction

Chalcopyrite is often associated with pyrite in sulphide ores. Its separation from pyrite becomes more difficult if pyrite misreports into the copper concentrate because of its accidental activation by copper released from chalcopyrite during grinding or conditioning (Fuerstenau, 1982; Chryssoulis et al., 1992; Finkelstein, 1997). Various reagents have been used to depress pyrite flotation, such as polymers, oxygen or ions such as hydroxide, ferric, carbonate, cyanide or sulphite, or a combination of these reagents (e.g., Fuerstenau, 1982; Finkelstein and Allison, 1976; Ball and Rickard, 1976; Draskic et al., 1980; Houot and Duhamet, 1992; Senior and Trahar, 1991; Boulton et al., 2001, 2003a,b). Zinc sulphate has been also used as a depressant, mainly to depress sphalerite flotation in the separation of chalcopyrite or galena from sphalerite (Clarke et al., 1995; El-Shall et al., 2000; Pearse, 2005). The mechanism of depression of sulphide mineral flotation by zinc sulphate and its selectivity have however not been studied in great details.

This study investigates the effects of oxidation potential (Eh) and zinc sulphate in the separation of chalcopyrite from pyrite at pH 9.0 in mixed mineral system and in two copper ores. Surface analyses are also used to explain the flotation results and to investigate the mechanism of zinc sulphate in the depression of pyrite flotation.

2. Experimental section

All chemicals used were of analytical grade. The collector, sodium isopropyl xanthate (SIPX) was further purified by dissolution in acetone and recrystallization from petroleum ether as described elsewhere (Montalti et al., 1991). Pyrite and chalcopyrite were obtained from

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Table 1
Mineralogical composition (wt.%) of the mineral samples

Samples	Fe	S	Cu	Pb	Si	Ca	Mg	Zn
Pyrite	45.6	52.0	0.12	0.05	0.255	0.26	0.025	0.07
Chalcopyrite	29.0	32.6	31.2	0.01	2.32	0.14	0.045	0.145
Ore 1	1.3	0.9	0.5					
Ore 2	25.5	29.3	1.9	0.04	16.3	0.26	0.69	<0.04

Huanzala Mine in Peru and Moonta Bay in Australia, respectively. Ore 1 is a copper sulphide ore containing primarily chalcopyrite and bornite with minor amount of chalcocite, covellite and vallerite; pyrite is the main gangue sulphide mineral. Ore 2 is also a copper sulphide ore with chalcopyrite and pyrite as the main sulphide minerals and silica as the main gangue mineral. The chemical compositions of these samples are reported in Table 1.

For the mixed mineral flotation experiments, a mineral feed of 20 g chalcopyrite and 80 g pyrite was used as representative of the ratio of copper sulphides to iron sulphides commonly found in real ores. These minerals (100 g; 0.6–3.2 mm) were ground together with 12 mild steel rods in a rubber lined-Galigher mill with 0.15 dm^3 of Adelaide water and 1250 g/t (per ton of mineral) of Na_2CO_3 to produce a mill discharge with a particle size d90 of 45 μm, a pH of 9.0 and an Eh of −230 mV (SHE). The pH was kept constant at 9.0 throughout conditioning and flotation with addition of Na_2CO_3. The conditioning times for Na_2CO_3, $ZnSO_4$, SIPX and the frother Dowfroth250 were 5, 5, 2 and 1 min, respectively. The Eh was controlled during the conditioning step by purging nitrogen and oxygen to obtain the desired Eh value before collector addition. Flotation experiments were performed with an Agitair flotation machine with a total pulp volume of 1.5 dm^3 at an air flow rate of 3 dm^3/min.

Ore 1 (1000 g; 0.6–3.2 mm) was ground with 12 mild steel rods (7.5 kg) in a stainless steel mill with 0.72 dm^3 of Adelaide water and 2 kg of lime to produce a mill discharge particle size d80 of 150 μm, a pH of 9.0 and an Eh of around 140 mV (SHE). The pH was kept constant at 9.0 throughout conditioning and flotation with addition of lime. The conditioning time for the ore, zinc sulphate, the collector S-8985 (a mixture of dithiophosphates from Cytec) and the frother (MIBC) were 5, 5, 2 and 1 min, respectively.

Ore 2 (1000 g; 0.6–3.2 mm) was ground with 16 mild steel rods (10.2 kg) in a stainless steel mill with 0.5 dm^3 of Adelaide water and 1.8 kg of lime to produce a mill discharge particle size d80 of 86 μm, a pH of 9.0 and an Eh of around −220 mV (SHE). The pH was kept constant at 9.0 throughout conditioning and flotation with addition of lime. The conditioning times for the ore, zinc sulphate, the collector AERO3477 (from Cytec) and the frother (Dowfroth250) were 3, 5, 2 and 1 min, respectively.

For both ores, flotation experiments were conducted in an Agitair flotation machine with a total pulp volume of 2.5 dm^3 at an air flow rate of 3.0 dm^3/min. In all flotation experiments, concentrates were collected at 0.5, 2, 4 and 8 min of flotation. Flotation rate constant (k) and maximum flotation recovery (% R_{max}) were calculated by fitting the cumulative recovery (% R) of each concentrate versus its corresponding flotation time (t) with a first order rate equation: % R=% R_{max} (1−exp(−kt)).

Electrophoretic mobilities were measured in a vertically mounted cell inside a Rank Brothers microelectrophoresis Mark II apparatus. The mineral particles were conditioned at pH 9.0 for 20 min in a 10^{-3} M KNO_3 aqueous solution and their mobility was measured from pH 11 to pH 5. The rate of pH decrease was the same in all the experiments. At least 10 mobility measurements at each of the two stationary planes were performed at each pH (successively reversing the platinum electrode polarization), and the average mobility was converted to zeta potential using the Smoluchowski equation.

Ethylene diaminetetraacetic acid (EDTA) was used to extract oxidation products from the mineral surface (Kant et al., 1994). Flotation concentrates (2 g) were mixed with 0.1 dm^3 of a 3% EDTA solution for 10 min and then separated by filtration. The EDTA solution was purged with high quality nitrogen gas before and during the extraction to prevent further mineral oxidation. The amount of surface metal oxidation products extracted by EDTA was measured by Inductively Coupled Plasma Mass Spectroscopy by Amdel Pty. Ltd., Australia.

A X-ray photoelectron spectrometer (XPS), Perkin Elmer Physical Electronics Division (PHI) 5100, with a MgKα X-ray source operated at 300 W was used to measure the species on the mineral surface down to a

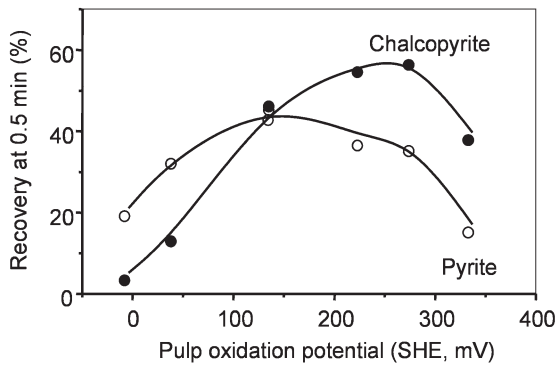

Fig. 1. (filled circles) Chalcopyrite and (empty circles) pyrite recoveries at 0.5 min of flotation in mixed mineral experiments as a function of Eh (pH=9.0; [SIPX]=200 g/t).