



# Incorporating fluidised-bed flotation into a conventional flotation flowsheet: A focus on energy implications of coarse particle recovery



Bellson Awatey\*, William Skinner, Massimiliano Zanin

Ian Wark Research Institute, University of South Australia, Mawson Lakes Campus, South Australia 5095, Australia

## ARTICLE INFO

### Article history:

Received 13 October 2014

Received in revised form 19 January 2015

Accepted 28 January 2015

Available online 4 February 2015

### Keywords:

Energy consumption

HydroFloat separator

Denver flotation cell

Coarse particle flotation

Recovery

## ABSTRACT

In the current work, we investigated a methodology to assess the potential energy savings when a fluidised-bed separator (i.e., the HydroFloat) is incorporated into a conventional flotation flowsheet for flotation of coarse sulphide minerals (chalcopyrite). Overall metallurgical performance of the new flotation set-up is compared to a single stage conventional flotation flowsheet. In the modified flowsheet, coarser particle sizes (+ 150  $\mu\text{m}$ ) from a coarse grind ( $d_{80} = 500 \mu\text{m}$ ) were floated in the fluidised-bed separator whereas the finer particles ( $- 150 \mu\text{m}$ ) were channelled to a conventional, agitated flotation cell. The coarser and high grade concentrate from the fluidised-bed separator was then reground and floated again using a mechanically agitated flotation cell. The energy used in grinding and regrinding was calculated using the bond work index formula and compared with the energy consumed when the total mass of sample was ground fine ( $d_{80} = 150 \mu\text{m}$ ) and floated in the mechanically agitated flotation cell only. Observations made from the results indicate that the floatability of the coarse chalcopyrite particles in the fluidised-bed separator depended mostly on their liberation classes. The remaining unrecovered copper was either completely or mostly encapsulated by gangue minerals. Though the metallurgical performance of the conventional flowsheet was superior in terms of copper recovery, the modified flowsheet showed promise in terms of energy efficiency. The methodology developed herein may be used for assessing coarse flotation strategies as a function of ore mineralogy type.

© 2015 Elsevier B.V. All rights reserved.

## 1. Introduction

Froth flotation remains one of the widely preferred separation technique used in the mineral processing industry to concentrate valuable minerals from their gangue host minerals. Like other separation techniques, flotation requires that large rocks are crushed, and in most cases ground to fine sizes depending on the physical and chemical properties (density, grain size, texture, other mineral associations, chemistry, etc.) of the ore, to liberate the valuable mineral(s) to be recovered. The process of breaking down the ore to liberate the value mineral(s) is termed comminution.

In the case of conventional flotation of base metals, the preferable particle size range for good recovery usually varies between 10 and 150  $\mu\text{m}$  [7]. This means that significantly high consumption of energy and grinding media would be required to breakdown the boulder size ore into the appropriate particle size for efficient flotation recovery [5]. In a recent study of various mining operations data, Curry et al. [4] showed that the overall cost distribution ratio between Mining, Milling (or processing) and General and Administration (Mine:Mill:G & A) ranges between 43:43:14 and 45:45:10. About 70–75% of the total electrical energy consumed in most mineral processing plants is used in comminution

processes [12,14]. The finer the product size, the higher the energy consumption rate [13]. Evidence of this claim can be found in Bond's third theory of comminution [3], expressed as follows:

$$W = 10W_i \left( \frac{1}{\sqrt{P_{80}}} - \frac{1}{\sqrt{F_{80}}} \right) \quad (1)$$

where  $W$  is the energy input,  $W_i$  is the bond work index (expresses the resistance of the material to comminution),  $P$  is the 80% passing size in micrometres for the product and  $F$  is the 80% passing size in micrometres for the feed.

The poor flotation of coarse particles in conventional, mechanically agitated flotation cells is often attributed to the high turbulent energy dissipation caused by the rotating impellers (cell hydrodynamics) [6,8,16]. Increasing the impeller rotation speed increases the local turbulent energy dissipation rate [8], which in turn, increases the liquid/bubble velocities. This reduces the overall contact time between bubbles and particles during the bubble–particle attachment process, and increases inertial forces leading to detachment either in the pulp or on impact with the pulp–froth interface.

Mechanical flotation devices require huge electrical energy investments to operate. In a cost breakdown of large flotation cells by Rinne and Peltola [15], it was found that about 68% of the cells' total operating

\* Corresponding author. Tel.: +61 8 8302 3714; fax: +61 8 8302 3683.  
E-mail address: [bellson.awatey@mymail.unisa.edu.au](mailto:bellson.awatey@mymail.unisa.edu.au) (B. Awatey).

cost comes from energy consumption. The power draw of the mixing mechanism in a mechanically agitated flotation cell is proportional to a third of the impeller rotation speed [15].

Unlike conventional flotation cells, fluidised-bed separators operate without rotating impellers. Particle suspension is achieved by the smooth and hindered settling conditions of the particles rather than by the turbulent conditions of the traditional flotation cell. The tendency for a significant reduction in energy consumption, increase in coarse particle attachment and subsequent recovery is therefore increased.

There are numerous advantages that fluidised-bed separators exhibit over conventional flotation cells with regard to coarse particle flotation. Some of these are discussed in detail in Jameson [7], Kohmuench et al. [9], Kohmuench et al. [11] and Kohmuench et al. [10].

The aim of this study was to investigate the overall metallurgical performance and energy consumption rates when a fluidised-bed separator is incorporated into a conventional flotation set-up for coarse sulphide mineral flotation. This was done by comparing two flotation flowsheets. In the first part of the study (Flowsheet 1), the total feed mass was ground coarse ( $d_{80}$  of 500  $\mu\text{m}$ ) and split into fine ( $-150 \mu\text{m}$ ) and coarse ( $+150 \mu\text{m}$ ) fractions. The fines were floated in a mechanically agitated cell (i.e., the Denver flotation cell), and the coarse particles were channelled to a fluidised-bed separator (i.e., the HydroFloat). The coarse concentrate from the fluidised-bed separator was then reground to a  $d_{80}$  of 150  $\mu\text{m}$  and floated again in the Denver flotation cell, while the tailings were rejected. In the second part of the study (Flowsheet 2), the feed was ground fine ( $d_{80}$  of 150  $\mu\text{m}$ ) and floated in a single stage conventional flotation in the Denver flotation cell. The two flowsheets were then compared in terms of overall recovery and energy consumption in the comminution circuits.

## 2. Experimental

### 2.1. Materials

Chalcopyrite ore was used in all the tests in this study. The ore was obtained from a copper mine in Australia. A chemical assay of the ore shows that it was a very high grade copper ore which contains 4.0% Cu, 7.5% S, 7.6% Fe, 3.0% Ca, 29.4% Si, and 0.9% Al. The mineralogical composition of the ore by particle size is shown in Table 1. We observe from Table 1 that the distribution of chalcopyrite increases as the particle size decreases, whereas the concentration of the other minerals remains almost constant across the size fractions. Fig. 1 shows SEM images of the ore indicating that the chalcopyrite was in simple and complex locking textures with the gangue mineral associations. The fine particles were, however, mostly fully liberated or had a simple locking texture whereas the coarser size fractions mostly showed complex locking textures.

### 2.2. Methodology

The set-up shown in Fig. 2(A) is modified after that which was previously presented in Awatey et al. [2]. In the current set-up the entire mass of sample undergoes primary grinding prior to flotation, unlike

the previous set-up where the flotation feed was a crushed product. This variation was important because of the difference in the particle size distributions and the difference in ore mineralogy for the two flowsheets.

The ore was first crushed in a laboratory jaw crusher, followed by a cone gyratory crusher to reduce the size of the lumps. The sample was then passed through a roll crusher and the product was screened through a 2.3 mm aperture size sieve. The oversize of the sieve was further crushed in the roll crusher and screened again. This process was repeated until all the ore passed through the 2.3 mm sieve. The experiments were divided into two parts. In the first part, 2 kg of the sample was wet ground in a Galigher laboratory rod mill with stainless steel rods ( $2.4 \times 25 \text{ cm}$ ,  $1.5 \times 25 \text{ cm}$  and  $1.2 \times 24.3 \text{ cm}$ ) for 18 min until about 80% passed through a 500  $\mu\text{m}$  aperture size sieve. The ground sample was screened with a 150  $\mu\text{m}$  sieve to separate the coarse particles from the fines. The coarser size fraction was prepared for fluidised-bed flotation whereas the finer particles were channelled to the mechanically agitated cell for conventional flotation.

In the second part of the experiment, the entire feed mass (2 kg) was ground to a  $d_{80}$  of 150  $\mu\text{m}$  in the same Galigher laboratory rod mill for 43 min using the same number of stainless steel rods. The entire mill product was then conditioned and prepared for flotation in a 4.5 litre conventional flotation cell. A Malvern Mastersizer was used to measure the size distribution of the feed material after grinding.

### 2.3. Reagents

Potassium amyl xanthate (PAX) and polypropylene glycol (PPG425) were used as the collector and frother respectively. Diesel oil was used as a collector extender to enhance attachment of particles to bubbles. Lime was used as the pH modifier.

### 2.4. Conditioning and flotation

The same collector concentration (450 g/t) was used for the tests based on the weight of the samples in both the fluidised-bed flotation and the mechanically agitated cell. The frother concentrations were, however, different, i.e., 1 g/t for the test in the fluidised-bed flotation cell, and 35 g/t for the tests in the Denver flotation cell. In Flowsheet 1, the coarser split portion ( $+150 \mu\text{m}$ ) was first conditioned in a 2.5 litre Denver flotation cell at 45% solids by weight. The pH was adjusted to, and maintained at  $10.5 \pm 0.04$ . Frother was added to the process together with the fluidisation air and water coming into the cell through an in-line sparging system. A quartz background of similar size distribution as the feed material (150–250  $\mu\text{m}$ , 250–425  $\mu\text{m}$ , 425–600  $\mu\text{m}$  and 600–850  $\mu\text{m}$ ) was used to build the fluidised-bed. The quartz particles simulate the non-float particles that, in the continuum process, autogenously generate the fluidized bed. The conditioned pulp was then transferred slowly into the suspended fluidised-bed separation zone with the aid of an overhead feeder. The concentrate collected at the end of a 14 minute flotation period was dried and sieved into the appropriate size fractions (150–250  $\mu\text{m}$ , 250–425  $\mu\text{m}$ , 425–600  $\mu\text{m}$  and 600–850  $\mu\text{m}$ ) and samples were taken and prepared for chemical assay. A

**Table 1**  
Quantitative XRD data showing the mineralogical composition (wt.%) of the ore used.

Mineral	Formula	– 63 $\mu\text{m}$	63–150 $\mu\text{m}$	150–425 $\mu\text{m}$	425–850 $\mu\text{m}$
Amorphous	Undefined	0	1.3	0.2	0
Quartz	$\text{SiO}_2$	39.8	39.9	48.2	53
Muscovite	$\text{K}_2\text{Al}_4[\text{Si}_6\text{Al}_2\text{O}_{20}](\text{OH},\text{F})_4$	0.4	0.5	0.8	0
Chlorite	$(\text{Mg}, \text{Fe}^{2+}, \text{Fe}^{3+}, \text{Mn}, \text{Al})_{12}[(\text{Si}, \text{Al})_8\text{O}_{20}](\text{OH})_{16}$	9.2	10.4	8.3	9.5
Dolomite	$\text{CaMg}(\text{CO}_3)_2$	27.3	27.6	24.6	21.5
Pyrite	$\text{FeS}_2$	7.5	6.6	6.9	6.9
Chalcopyrite	$\text{CuFeS}_2$	15.8	13.5	11	9
Total		100	99.8	100	99.9

Download English Version:

<https://daneshyari.com/en/article/235662>

Download Persian Version:

<https://daneshyari.com/article/235662>

[Daneshyari.com](https://daneshyari.com)