Flotation of copper sulphides assisted by high intensity conditioning (HIC) and concentrate recirculation

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A B S T R A C T

This paper describes the effect of the partial concentrate (rougner floated product) recirculation to rougher flotation feed, here named concentrate recirculation flotation – CRF, at laboratory scale. The main parameters used to evaluate this alternative approach were flotation rate and recovery of fine (“F” 40–13 μm) and ultrafine (“UF” <13 μm) copper sulphide particles. Also, the comparative effect of high intensity conditioning (HIC), as a pre-flotation stage for the rougher flotation, was studied alone or combined with CRF. Results were evaluated through separation parameters, grade-recovery and flotation rates, especially in the fine and ultrafine fractions, a very old problem of processing by flotation. Results showed that the floated concentrate recirculation enhanced the metallurgical recovery, grade and rate flotation of copper sulphides. The best results were obtained with concentrate recirculation flotation combined with high intensity conditioning (CRF–HIC). The kinetics rate values doubled, the Cu recovery increased 17%, the Cu grade increased 3.6% and the flotation rates were 2.4 times faster. These were accompanied by improving 32% the “true” flotation values equivalent to 2.4 times lower the amount of entrained copper particles. These results were explained and proved to proceed by particle aggregation (among others) improving 32% the “true” flotation values equivalent to 2.4 times lower the amount of entrained copper particles. These results were explained and proved to proceed by particle aggregation (among others) occurring after HIC, assisted by the recycled floatable particles. This “artificial” increase in valuable mineral grade (by the CR) resulted in higher collision probability between hydrophobic particles acting as “seeds” or “carrier”.

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1. Introduction

The poor flotation recoveries of fine (“F” 40–13 μm) and ultrafines (“UF” <13 μm) mineral particles is very well known and many studies, projects and processes have been proposed, but this major technical and economic problem still continues (Collins and Read, 1971; Fuerstenau et al., 1979; Trahar, 1981; Sivamohan, 1990; Song and Lu, 2000; Rubio et al., 2003, 2006). The mineral fines formation or generation is diverse, from many sources: some particles are necessarily produced in the grinding process for adequate liberation of the particles; another fraction comes from re-grinding and a small quantity from particles shearing and smearing (in conditioners for example). In some cases, certain mineral ores must be ground to very fine particles because of the decreasing size in the grade and the increase of complex ores. Under actual conditions, a great deal of fine minerals is discarded as slimes prior to flotation for lack of effective recovery methods.

Many alternatives for solving this problem have been tried, but unfortunately most of them have not found practical applications (Rubio, 2002, 2006; Song and Lu, 2000). A good example is the case of selective flocculation with polymers that, despite more than 30 years of extensive studies and a few (mainly iron) plants using this technique, still remain a “promising” process (Fuerstenau et al., 1979; Subrahmanayam and Forsberg, 1990; Sivamohan, 1990; Mathur et al., 2000).

Main problems associated to the fines flotation are caused by their intrinsic properties, among others, small mass (low momentum), high interfacial free energy. Thus, main flotation drawbacks are related to the low probability of bubble–particle collision and adhesion, mechanical entrainment and entrapment, “slime coating”, higher flotation reagent adsorption, formation of dense froths and low process kinetics. It is well known that ultrafine and coarse particles float more slowly than particles in a mid-size range. The size limits are not precise but for many mineral systems, the ultrafines are those below 5–10 μm approximately. The flotation rate tends to decrease with decreasing particle size, because of the increasing importance of viscous and electrostatic forces, which are less important with larger particles. The rate of bubble–particle collision is, for the coarser particles higher than for the ultrafines, but the hydrodynamic forces associated with turbulence and shear in the pulp, lead to an increase in the rate of detachment. Thus as the particle size increases, it becomes increasingly difficult to maintain the capillary
bonds between the particles and the bubbles and to withstand shearing, while being transported into the froth.

Most selective aggregation using chemicals has not proven to be a unique and general solution, the conventional or column type flotation machines do not generate the bubbles size distribution required for an efficient capture of the very fines particles, neither the actual “conditioners” (merely pulp distributors) allow aggregation of the fines particles, F–UF, themselves (Rubio et al., 2003, 2006).

The concept of shear flocculation has been extending to the conditioning stage ahead of flotation (Bulatovic and Salter, 1989; Stassen, 1991). Many authors claim that energy transferred in the conditioning stage, often expressed as conditioning time, has a pronounced effect on the concentrate recovery, grade and flotation rate. The high intensity conditioning process (HIC) enhanced the flotation recovery of the very fine particles of copper sulphides (Bulatovic and Salter, 1989), gold, uranium oxide and pyrite fines (Stassen 1991; Valderrama and Rubio, 1998), oxidized copper ores and copper and molybdenum sulphides (Rubio et al., 2003) and nickel ore (Chen et al., 1999). It is appear that this process is now being applied industrial using special designed conditioners.

However, there is a situation, not much explored, viz. staged or multifeed flotation, whereby differ from conventional flotation in that two or more feed inputs are used and improvements in both

**Table 1**

Multifeed circuits reported on the literature.

<table>
<thead>
<tr>
<th>Author</th>
<th>Country/Mineral system</th>
<th>Comments</th>
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<tbody>
<tr>
<td>Firth et al. (1979); Firth et al. (1978);</td>
<td>Australia/coal</td>
<td>Investigation of feed size distribution on the staged flotation of poorly floating coals. These circuits (staged flotation) were two-stage reagent addition, re flotation of classified tailings and split feed flotation. The re flotation of classified tailings circuits not only gave the best metallurgical performance, it also was the least affected by pulp density variations and imperfect size classification</td>
</tr>
<tr>
<td>Tsai (1988)</td>
<td>USA/coal</td>
<td>Staged froth flotation for deep cleaning of fine coal. In this staged flotation, the slowly floatable, mineral-rich particles are first rejected, and the resulting overflow is recl amed at lower flotation rates through careful control of frother concentration. As a result, a clean coal product of lower ash and lower pyritic sulfur contents has been produced at a higher yield as compared with the single-stage flotation and other reported multistage flotation circuits. The selectivity of this staged flotation is dictated by the mineral size and its distribution in the feed coal. The mineral particles that remain in the clean coal have a volume mean diameter of 6 μm, with 50 per cent being smaller than 13 μm</td>
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<tr>
<td>Fuestenau (1988)</td>
<td>USA/ molybdenite</td>
<td>The possibility of improving the flotation efficiency of Turkish bituminous coal fines by a split feed flotation scheme was investigated in comparison with conventional single-stage flotation. Coal samples collected from the &lt;0.6 mm fines treatment circuit of an operating washery were used in laboratory batch tests simulating single-stage and split feed flotation circuits. Collector (kerosene) and frother (iso-octanol) additions were used as major test variables. Split feed flotation appeared to be a better alternative scheme from the metallurgical point of view</td>
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<tr>
<td>Hosten and Muratoglu (1996)</td>
<td>Turkey/ bituminous coal</td>
<td>A brief discussion on the potential of different kind of circuit, namely the multifeed circuits, was presented. Multifeed circuits differ from conventional circuits in that two or more feed inputs are used. Improvements in both mineral recovery and grade (through autogenous carrier effects) and conservation of flotation reagents. Multifeed circuits offer potential for large plants processing several ores of varying composition</td>
</tr>
<tr>
<td>Fuestenau et al. (2000)</td>
<td>USA/coal and molybdenite</td>
<td>Results of an experimental application of this technique to the flotation of a molybdenite ore was presented to illustrate the potential benefit from multifeed circuit. Molybdenite multifeed circuit flotation roughers significantly improved the grade of optimized single-stage roughers: from 14 to 34 per cent MoS2 while maintaining comparable molybdenite recovery and reducing Syntex (emulsifying agent, that is used to enhance molybdenite flotation) and frother consumption by one-half</td>
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<td>Rodrigues et al. (2005)</td>
<td>Brazil/copper</td>
<td>The high intensity conditioning process (HIC) enhanced the flotation recovery of the very fine particles of copper sulphides</td>
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<td>Proyecto Flotación de finos y ultrafinos-Codelco-Chuquicamata – IM2 (2002), Tabosa (2007), Tabosa et al. (2009).</td>
<td>Brazil/chile/copper</td>
<td>Studies of coal and molybdenite flotation circuits were carried out experimentally, using semi-batch flotation methods. Investigation of a new multifeed flotation circuits, in which flotation feed is split into two fractions with the concentrate from one stage being fed with the other half of the feed into a second stage. These circuits would permit reduction in reagent additions and higher grade rougher concentrate was produced with the flotation of a molybdenite ore</td>
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<td>Laboratory experimentation showed that multifeed rougher circuits can provides can yield enhanced results in the flotation of naturally hydrophobic minerals, such as coal and molybdenite</td>
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<td>For coal, studies showed that open multifeed rougher circuits can provide yield and product ash contents comparable to single-stage rougher. Closed multifeed rougher circuits provided much higher yields than conventional single-stage rougher circuits at lower reagent consumption</td>
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<td>For molybdenite flotation, open two-stage multifeed rougher circuits appears to be far superior to the single-stage rougher for producing a much higher grade rougher concentrate at lower reagent consumption. This appear to be due to the effects of conditioning fresh feed with reagent-coated rougher concentrate</td>
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<td>Concentrate recirculation flotation (CRF) was proposed aiming to modify flotation circuits design. CRF includes one conventional flotation stage and a second stage in which its feed is compound of “first” (higher flotation kinetics particles) concentrates and fresh feed. On this stage (second stage) occurs an artificial increase of higher flotation kinetics particles. Those particles are suitable for aggregates formation, serving as “seeds” for these aggregates, transporting lower flotation kinetics fine and ultrafine particles. Besides, aggregates formation can increase using high intensity conditioning (HIC) combined with this technique</td>
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<td>Flotation of sulphide fines results, using CRF technique and HIC combined (CRF–HIC), were compared with a “standardized” mill laboratory procedure. Concentrate copper recoveries were of the order of 7–18 per cent higher and their process kinetics values were higher, 1.5 min⁻¹ for CRF and 2.9 min⁻¹ for CRF–HIC, compared with 1.2 min⁻¹ for standardized mill laboratory procedure</td>
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</table>
mineral recovery and grade (through autogenous carrier effects) and conservation of flotation reagents were reported. There are some industrial cases whereby concentrate (a fraction) is, occasionally, being recycled but is not a common practice worldwide. Several authors (Hu and Huang, 1977; Firth et al., 1978, 1979; Tsai, 1988; Rubio, 1988; Fuerstenau, 1988; Fuerstenau et al., 2000; Hosten and Muratoglu, 1996; Rodrigues et al., 2005; Proyecto Flotación de finos y ultrafinos – Codelco – Chuquicamata – IM2, 2002; Tabosa, 2007; Tabosa et al., 2009) – see Table 1 – on the analysis of some coal, copper and molybdenum sulphide flotation circuits reported the multifeed circuit as a potential alternative circuit. According to them, several factors may be responsible to these improvements. Split feed flotation on multifeed circuits, whereby the concentrate of part of a rougher circuit is mixed with fresh feed onto another rougher circuit, reducing reagents consumption and improving fine mineral particles recovery. Also, flotation rate in-

![Fig. 1. Schematic representation of the configuration of various conventional and multifeed flotation circuits (after Fuerstenau et al. (2000)).](image)

![Fig. 2. Copper and mass size distribution in sample.](image)
creases due to an “artificial” increase of valuable mineral grade (faster flotation rate), where hydrophobic particles may act as “seeds” (carrier) to fine particles. Besides this, lower reagents consumption, due to the effects of conditioning fresh feed with reagent-coated rougher concentrate.

Fuerstenau et al. (2000) show that multifeed rougher circuits can provide enhanced results in the flotation of naturally hydrophobic minerals, such as coal and molybdenite. For coal, it was shown that open multifeed rougher circuits (Fig. 1) can provide yield and product ash contents comparable to single-stage rougher. Closed multifeed rougher circuits provided much higher yields than conventional single-stage rougher circuits at lower reagent consumption. For molybdenite flotation, an open two-stage multifeed rougher circuit appears to be far superior to the single-stage rougher for producing a much higher grade rougher concentrate at lower reagent consumption.

Firth et al. (1978) suggest a staged (multifeed) flotation circuits for an Australian poorly floating coal. These circuits were two-stage reagent addition, reflotation of classified tailings and split feed flotation. According to them, the key to successful flotation of poorly floating coals is the equitable distribution of collectors to all size fractions.

The aim of this work is to compare, at laboratory scale, standardized rougher flotation tests with partial concentrate (routher floated product) recirculation to rougher flotation feed (concentrate recirculation flotation – CRF). Also, the comparative effect of high intensity conditioning (HIC), as a pre-flotation stage for the rougher flotation, is studied alone or combined with CRF. Compar-
isons were measured through the separation parameters: grade, recovery (global and true flotation) and flotation rate constants included size-by-size results to assess the flotation behaviour of the fine fractions of copper sulphides.

2. Experimental

2.1. Materials

2.1.1. Ore
Copper porphyry samples, containing 0.98–1.10% Cu were supplied by Codelco – Chile. About 1/4 of the mass and about 40% of the total copper content is in the fines and ultrafines fractions (<35 µm). Fig. 2 shows feed size-by-size mass and copper distribution.

2.1.2. Reagents
Lime, as pH regulator to pH 10.5 was added at the conditioning stage together with SF-113 (sodium isopropyl xanthate) and SF-506 (Promoter) at 30 and 8 g t⁻¹ concentration, respectively. The frother was a propylene glycol, DF-250, at 25 g t⁻¹.

2.2. Methods

2.2.1. Standard mill test
The standard mill tests were carried out in a laboratory D12 Denver flotation machine and in a square flotation cell with 3 L of capacity, endowed with a manual scraper to remove the froth. The pulp with 38% in weight and with pH in 10.5 (monitored with an AnalionTM pH meter model PM 608), was conditioned with the collectors SF-113 (25 g t⁻¹), SF 506 (8 g t⁻¹) and the frother (25 g t⁻¹) for one minute to allow collector and frother adsorption at 1000 rpm in the cell. After the conditioning time, the flotation started with the injection of air in the flotation cell, monitored with a flow meter. Data were obtained from concentrates taken at 1, 2, 3, 5 and 9 min flotation, keeping the volume in the cell constant with water injection at pH 10.5. Fig. 3 shows a schematic representation and some pictures of the standard mill test configuration.

2.2.2. Concentrate recirculation flotation (CRF)
The CRF studies were divided into two-stages. The first stage was carried out a standard mill test, and the first minute concentrate (higher flotation probability) was collected. The volume of

Fig. 5. Details of the: (a) 6-blade Rushton impeller and (b) baffles added to the flotation cell at the high intensity conditioning stage.

Fig. 6. Schematic representation of the configuration of a high intensity conditioning (HIC) mill test.
this concentrate was measured and added to the second stage feed. At this second stage, a new standard mill test was carried out, being its feed composed by a new feed plus the first concentrate of the STD test (first stage), whilst the pulp volume was kept constant. Fig. 4 shows a schematic representation of the concentrate recirculation flotation (CRF) tests.

2.2.3. HIC (high intensity conditioning)

The HIC studies were carried out similarly to the standard mill test, with the exception that the stage of the conditioning of the pulp with the reagents (collectors and frother), was substituted for a high turbulent stage, obtained with the introduction of four baffles in a flotation cell equal to the mill flotation test and a Rush-

<table>
<thead>
<tr>
<th>Tests</th>
<th>Mass recovery (%)</th>
<th>1st minute copper metallurgical recovery (%)</th>
<th>Copper metallurgical recovery (%)</th>
<th>Copper grade (%)</th>
<th>True flotation recovery (%)</th>
<th>Entrainment grade</th>
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<tr>
<td>STD</td>
<td>5.8</td>
<td>25</td>
<td>60.5</td>
<td>1.1</td>
<td>11.2</td>
<td>0.4</td>
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<tr>
<td>CRF</td>
<td>9.6</td>
<td>67.0</td>
<td>1.3</td>
<td>9.1</td>
<td>0.5</td>
<td>38</td>
</tr>
<tr>
<td>HIC</td>
<td>6.4</td>
<td>69.0</td>
<td>1.1</td>
<td>12.0</td>
<td>0.4</td>
<td>51</td>
</tr>
<tr>
<td>CRF–HIC</td>
<td>8.6</td>
<td>43</td>
<td>78.0</td>
<td>1.6</td>
<td>15.0</td>
<td>0.4</td>
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Fig. 7. Flotation kinetic curves of a copper sulphide ore, comparative results (STD, CRF, HIC, and CRF–HIC). Tests conditions: [SF-113] = 30 g t⁻¹; [SF-506] = 8 g t⁻¹; [DF250] = 25 g t⁻¹; pH = 10.5; 38% solids by weight; conditioning stage 1000 rpm (HIC and CRF–HIC: 1400 rpm - 3 min).

Fig. 8. Copper grade-recovery curves, comparative results (STD, CRF, HIC, and CRF–HIC). Tests conditions: [SF-113] = 30 g t⁻¹; [SF-506] = 8 g t⁻¹; [DF250] = 25 g t⁻¹; pH = 10.5; 38% solids by weight; conditioning stage 1000 rpm (HIC and CRF–HIC: 1400 rpm - 3 min).

Fig. 9. Cumulative copper concentrate grades, comparative results (STD, CRF, HIC, and CRF–HIC). Tests conditions: [SF-113] = 30 g t⁻¹; [SF-506] = 8 g t⁻¹; [DF250] = 25 g t⁻¹; pH = 10.5; 38% solids by weight; conditioning stage 1000 rpm (HIC and CRF–HIC: 1400 rpm - 3 min).

Fig. 10. Mean particle size in the concentrate as a function of time, comparative results (STD, CRF, HIC, and CRF–HIC). Tests conditions: [SF-113] = 30 g t⁻¹; [SF-506] = 8 g t⁻¹; [DF250] = 25 g t⁻¹; pH = 10.5; 38% solids by weight; conditioning stage 1000 rpm (HIC and CRF–HIC: 1400 rpm - 3 min).
ton impeller (Fig. 5a). This high-shear radial impeller is excellent for particle dispersion.

The HIC were carried out in the same D12 Denver flotation machine, when the stator was taken out, and the rotor was substituted for a Rushton impeller with 1400 rpm speed (transferring 2 kWh m⁻³ of pulp), during 3 min, without air injection, and with 50% of solids by weight. After HIC, the solids content was adjusted to 38% solids by weight, the impeller speed was changed back to 1000 rpm and the flotation started with the injection of air in the flotation cell, monitored with a flow meter, as in the STD test. Figs. 5 and 6 show some details of the HIC system used.

Herein, the studied variable, conditioning time, was quantified in terms of energy transferred to pulp. This energy was measured with a wattmeter, as described by Valderrama (1997).

2.2.4. CRF combined with HIC (CRF–HIC)

The CRF–HIC studies were carried out as a HIC test (stage 1), as described previously, and the first minute concentrate (higher flotation probability) was collected. The volume of this concentrate was measured and added to the second stage feed (new HIC). At this second stage, a new HIC mill test was carried out, (stage 2) being its feed composed by a new feed plus the first concentrate of the HIC test (stage 1), whilst the pulp volume was kept constant.

All concentrates and tailings were filtered, dried, weighed and analyzed for copper by atomic adsorption (SpectrAA 110-Varian).

The process efficiency was measured by evaluating separation parameter recovery, “true” recovery and entrainment grade (as described by Warren, 1985, 1991). Cu grade and rate constants. The constant recoveries of the Klimpel model (best model for the data obtained) were obtained using a nonlinear regression method (least square fitting). In all the fittings the $R^2$ value was about 0.99.

Finally, to assess the performance of flotation in the F and UF fractions, size-by-size copper recoveries were measured in the fractions: >53 µm; 53–35 µm; 35–25 µm; 25–15 µm and <15 µm.

3. Results and discussion

Figs. 7 and 8 and Table 2 show comparative results of flotation of copper sulphides at lab scale. Fig. 7 and Table 2 shows that under standard flotation test conditions, less valuable particle are floated compared to the other techniques, reaching to recoveries of the order of 60% Cu with rate constants (Klimpel model) of about 1.2 min⁻¹ and a true recovery or recovery through bubble–particles interactions of 30%.

The concentrate recirculation flotation (CRF) test (1st min concentrate recirculation), compared with the mill standard test, increased the kinetics rate values, enhanced the Cu recovery by 6.5% and the recovery by true flotation values by 8%; followed by a decrease in entrainment grade and a slight reduction in copper final concentrate grade.

When the high intensity conditioning (HIC) precedes the STD flotation test, the kinetics rate values approximately doubled; the Cu recovery increased by 8.5% and the recovery by true flotation by 21%. Also, the entrainment grade reduced to 0.9 and the copper final concentrate grade slightly increased. Hence, results show that the high intensity conditioning, as a pre-flotation stage, might increase both copper grade and recovery revealing, like in other studies, high process selectivity. This specificity appear to be demonstrated by the enhanced true flotation values, which can only be explained in terms of the appearance of “new” valuable and hydrophobic particles in the concentrate. Thus, either particle aggregation or surface cleanliness or both, are the mechanisms involved.

Yet, best metallurgical and kinetic results were obtained with concentrate recirculation flotation combined with high intensity conditioning (CRF–HIC). Results were 17% greater Cu recovery and 3.6% improved Cu grade at a higher flotation rates (2.4 times faster). These were accompanied by an enhancement equivalent to 32.5% in the true flotation values and by a low amount of entrained copper particles (2.4 times smaller).

A comparative analysis of copper concentrate grade with flotation time (Fig. 9) shows similar results for both STD and HIC tests. Concentrate recirculation, however, affected each technique in a distinct manner. Whilst the concentrate recirculation flotation (CRF) decreases the copper concentrate grade the CRF assisted by the high intensity conditioning (CRF–HIC) increases the copper concentrate grade.

This general behaviour might be the result of an “artificial” increase in valuable values grade followed by an aggregation occurring during the HIC, improving overall flotation performance. The
recycled valuable mineral particles would be acting as “seeds” or carriers whereby other particles can be attached.

This combination of methods appears to be similar to the mechanisms proposed for the so-called autogenous flotation (Valderrama and Rubio, 1998).

In order to prove the aggregation hypothesis, the mean size of floatable particles (concentrate) were measured comparatively. Fig. 10 shows that bigger particles were formed in the very first minutes of flotation when that is assisted by the high intensity conditioning. This could therefore suggest the formation of some kind of particle aggregation.

Moreover, the particle size distribution of concentrate products for STD, CRF, HIC and CRF–HIC presented in Fig. 11 shows that when flotation is assisted by the high intensity conditioning (both HIC and CRF–HIC) coarser particles are recovered; which in turns could indicate that particles acted as carrier for particle aggregation of the fines themselves or with coarser particles.

Fig. 12 shows particle mass recoveries, in each fraction, compared to the mill standard. Results show that after HIC and CRF–HIC the recovery of fine (<15 μm) and intermediate (>53 μm) particle size was higher when compared to STD. Highest recovery of fine (<15 μm) particles was obtained with concentrate recirculation flotation, indicating its higher efficiency for the flotation of the finest particle fraction.

Finally, Fig. 13 shows size-by-size copper recoveries, obtained in all techniques studied compared to the mill standard. According to expectations (and mechanisms proposed), this figure shows clearly higher recoveries in practically all fractions, especially in the fine and ultrafine fractions (5% higher to CRF) and in the coarser (>35 μm) fraction, 10–15% higher for CRF–HIC. These results support the aggregation hypothesis of the fines themselves or with coarser particles (“seeds” or carrier). This may explain the results obtained, the higher recoveries and concentrate grades at higher process kinetics.

4. Conclusions

Results found showed that the concentrate recirculation, to a rougher flotation feed, enhanced the metallurgical recovery, grade and rate flotation of copper sulphides. The best results were obtained with high intensity conditioning (HIC), as a pre-flotation stage, combined with concentrate recirculation flotation (CRF), increasing the kinetics rate values, Cu recovery and concentrate grade. These results were accompanied by an enhancement in the “true” flotation and a low amount of entrained copper particles. Particle aggregation occurring after HIC, enhanced by the higher number of recycled floatable particles, might explain these results. This “artificial” increase in valuable mineral grade resulted in higher collision probability between hydrophobic particles acting as “seeds” or “carrier”. Size-by-size flotation recoveries appear to confirm this aggregation mechanism. Hence, it is believed that these results represent real advances and that this concentrate recirculation based technique may establish a new basis to improve the potential of HIC, as an efficient pre-flotation stage.

Acknowledgments

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References


![Fig. 13. Comparative size-by-size flotation copper recoveries: (STD, CRF, HIC, and CRF–HIC). Copper recoveries of concentrate products in >53, 35, 25, 15 and <15 μm fractions. Tests conditions: [SF-113] = 30 g t⁻¹; [SF-506] = 8 g t⁻¹; [DF250] = 25 g t⁻¹; pH = 10.5; 35% solids by weight; conditioning stage 1000 rpm (HIC and CRF–HIC: 1400 rpm – 3 min).](image-url)


