UNIVERSITÉ DU QUÉBEC EN ABITIBI-TÉMISCAMINGUE

ENHANCING FINE PARTICLE RECOVERY IN FLOTATION AND ITS POTENTIAL APPLICATION TO THE ENVIRONMENTAL DESULPHURIZATION PROCESS

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Summary

Environmental desulphurization was recently proposed as a possible method to reduce the volume and mass of acid-generating tailings stored in tailings ponds. It consists of separating the sulphide minerals (acid-generating) from the tailing stream before its disposal in the tailings pond using a bulk sulphide flotation process. Studies have shown lower recoveries for fine sulphide particles compared to coarse and middle-sized particles, a problem also observed in other flotation applications. This document addresses the following question: How can we improve fine pyrite and pyrrhotite recovery in the environmental desulphurization process by flotation?

Two main approaches are used to improve the fine particle recovery in flotation; to increase the particle size, or to decrease the bubble size. Particle size is increased through several aggregation methods and the aggregated particles are then floated, a process known as floc flotation. Aggregation can be induced by polymer flocculants, hydrophobic interactions and micro-organisms.

Fine particles are best floated with small bubbles, and hydrodynamic conditions inside a mechanical flotation cell are not suited for fine particle flotation. Flotation columns of various types yield better fine particle recovery because of more favourable conditions and smaller bubbles. The small bubbles can be generated via dissolved-air flotation, induced-air flotation, hydrodynamic cavitation, electroflotation and microbubble generators.

Most of the methods are irrelevant of the mineral type and surface state as long as the mineral is hydrophobic, either naturally or made hydrophobic by surfactants. Therefore, theoretically, they could be applied to sulphides (pyrite and pyrrhotite) in environmental desulphurization. Suggestions are presented for future testwork in environmental desulphurization, such as the use of flotation columns, contact cells or Jameson cells, hydrophobic flocculation without oil enhancement, the use of fine bubble generators and microbially-induced flocculation. However, polymer flocculation, electroflotation, turbulent microflotation and the use of centrifugal force field were not suggested for fine sulphide particle recovery in environmental desulphurization.

Keywords: fine particle flotation, floc flotation, bubble size, environmental desulphurization

Introduction

The problem of acid mine drainage is a major concern in the mining industry (ex. Ritcey, 1989; Ripley, Redmann and Crowder, 1996). The environmental impacts of acid mine drainage are well known, and the mine operators now include a remediation plan for the potentially acid-generating areas in the design phase of the mine (Aubertin, Bussière and Bernier, 2002). The traditional method to reduce the environmental impacts of acid mine drainage consists of the treatment of the effluent by neutralization with an alkali for over 100 years (ex. Ritcey, 1989). More recently, the trend became towards the prevention of acid mine drainage formation, thus treatment is no longer required for extended periods of time. Different techniques were developed to prevent the formation of acid mine drainage, such as covers made of soil or water or synthetic materials (Aubertin, Bussière and Bernier, 2002). In the case of tailing impoundments, a new approach consists of separating the sulphide minerals (acid-generating) from the tailing stream before its disposal in the tailings pond. This technique called "environmental desulphurization" was tested in the laboratory and on pilot scale, and the results are promising (ex. Bussière *et al.*, 1998).

The separation of sulphides in the environmental desulphurization process is performed via a bulk sulphide flotation. However, the recovery of fine particles is known as being low in flotation processes (ex. Trahar and Warren, 1976). Therefore, the following question was given:

> *How can we improve fine pyrite and pyrrhotite particle recovery in the environmental desulphurization process by flotation?*

The present document will suggest answers by means of a literature review. A first section briefly discusses the environmental desulphurization process. A second section covers the problem of fine particle recovery in flotation. Sections 3 and 4 present different methods reviewed to improve fine particle recovery in flotation. The document ends with conclusions and a discussion on the applicability of the various methods presented to the flotation of pyrite and pyrrhotite in the context of environmental desulphurization.

Please note that the document covers options to improve fine particle recovery assuming that the fine particles are present in the circuit feed and their removal before the flotation process is not desirable. From a technical point of view, gravity separation of fines from the flotation feed stream would eliminate the issue of fine particle flotation. However, the focus of the document is the question given previously which implies that fine particles are present in the process and solutions should be proposed to improve their flotation recovery.

1 Environmental desulphurization

1.1 Principles and testwork

Acid mine drainage is produced when sulphide minerals are oxidized by contact with oxygen and water. The purpose of environmental desulphurization is to remove enough sulphides to prevent oxidation and / or acid mine drainage formation (ex. Bussière *et al.*, 1998; Benzaazoua *et al.*, 2000). A tailing stream contains a variety of minerals including sulphides and neutralizing minerals (ex. carbonates). The objective of desulphurization is to remove enough sulphides to lower the ratio of acid-producing to acid-neutralizing minerals to a point where the tailings are non acid-generating (Bussière *et al.*, 1998; Benzaazoua *et al.*, 2000). To achieve this, several gravity concentration methods and flotation were evaluated as possible routes, with bulk sulphide flotation being the most efficient (Humber, 1995; Bussière *et al.*, 1998).

Several authors succeeded in producing a potentially non acid-generating tailing using the environmental desulphurization process. McLaughlin and Stuparyk (1994) were able to reach 0.40% S with the Clarabelle mill tailing on a test circuit, at an operating pH of 6-7, up to 10 g/t xanthate addition and 10-15 minutes of retention time. Leppinen, Salonsaari and Palosaari (1997) also produced a potentially non acid-generating tailing from the Pyhäsalmi tailing stream on a laboratory scale. They operated at natural pH (5.5 to 6.5) with 100 to 150 g/t SIBX addition. Other laboratory tests performed on four different mill tailings (Benzaazoua *et al.*, 2000) were able to produce potentially non acid-generating tailings, and emphasized the choice of collector depending on the pulp: cyanide-free pulps can be desulphurized using xanthates while cyanided pulps require an amine collector.

The main challenge is to desulphurize the tailings after the ore treatment which usually depressed pyrite to recover valuable minerals (ex. sphalerite, chalcopyrite). However, the object of this document not being a literature review on desulphurization, it is sufficient to mention that the process is successful in producing a non acid-generating tailing and a sulphide concentrate from a concentrator tailing stream using flotation.

1.2 Effect of particle size

Although Leppinen, Salonsaari and Palosaari (1997) were able to desulphurize the Pyhäsalmi tailings, a low limit on sulphur was observed at 0.48%. A possible explanation came from the size by size sulphide distribution of the desulphurized tailings, presented in Table 1. More than half of the sulphides appear in the two finest fractions, -63 μm. The small particle size could explain the low sulphide recovery, since most of the sulphides are present as fines.

Table 1 : Distribution of sulphides after flotation treatment of the tailing sample (Leppinen, Salonsaari and Palonsaari, 1997)

Size fraction (μm)	Content of sulphides (%)	Distribution of sulphides (%)
+250	0.39	8.70
125 / 250	0.32	8.79
90 / 125	0.76	11.92
63 / 90	0.95	13.36
20 / 63	1.22	31.06
-20	1.28	26.17
Total		100.00

Benzaazoua, Kongolo and Bussière (2000) observed a decrease in pyrite recovery and a lower flotation rate constant for fine particles. They suggested that the optimal size range for pyrite flotation in desulphurization is between 35 μm and 120 μm. This is consistent with earlier observations on pyrite flotation reported by Trahar and Warren

(1976) which define the size range for maximum pyrite recovery between 50 μm and 150 μm.

The environmental desulphurization process could become more efficient if the recovery of fine sulphides was enhanced. However, lower finer particle recovery is not a problem specific to desulphurization. To answer the question "How can we improve fine pyrite and pyrrhotite particle recovery in the environmental desulphurization process by flotation", it is necessary to look into work published on the flotation of various minerals to find possible solutions.

2 Fine particle recovery in flotation

2.1 Reasons for low recovery

The size at which a particle becomes a "fine particle" is defined as the size finer than the minimum for optimal flotation recovery (Subrahmanyam and Forssberg, 1990). In the papers reviewed for this document the range of fine particles was usually from -20 μm to colloidal sizes. Figure 1 illustrates that sulphide recovery is low for the fine particle size for two ores (bulk sulphide flotation). Similar curves are obtained with a variety of minerals (ex. Trahar and Warren, 1976).

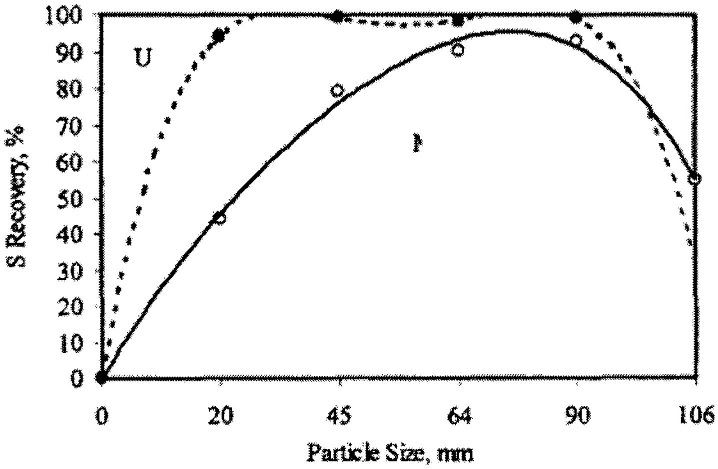


Figure 1 : Variation in sulphur recovery with particle size fraction for Merensky (M) and UG 2 (U) ores. (from Feng and Aldrich, 1999)

The reasons for the low performance of flotation on fine particles are usually explained by the three distinct characteristics related to their small size: small mass, high specific surface area and high surface energy (Soto and Barbery, 1988; Subrahmanyam and Forssberg, 1990). Figure 2 presents the implications of the three characteristics on the grade and recovery behaviour in flotation. The small mass of the particles causes a low flotation rate because of the lower probability of particle-bubble collision and adhesion (Soto and Barbery, 1988; Lange, Skinner and Smart, 1997; Feng and Aldrich, 1999; Song *et al.*, 2001; Liu *et al.*, 2002; Pyke, Fornaserio and Ralston, 2003). The high surface energy and high surface area result in high non specific reagent consumption, enhanced surface oxidation and solubility, which decrease selectivity (Soto and Barbery, 1988; Gorman and Smith, 1991; Lange, Skinner and Smart, 1997; Feng and Aldrich, 1999; Song *et al.*, 2001). The term "high surface energy" is used rather loosely in the literature and its significance is not clear.

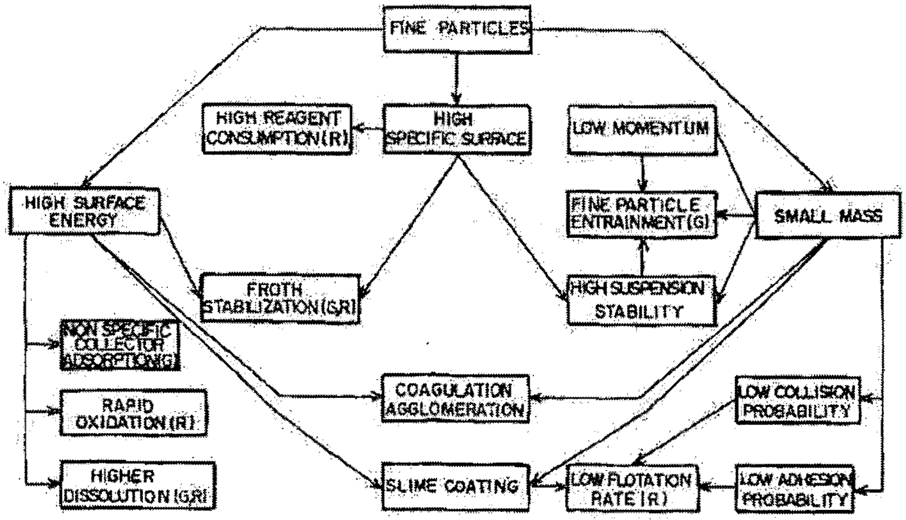


Figure 2 : Relationship between the physical and chemical properties of fine particles and their behaviour in flotation (from Subrahmanyam and Forssberg, 1990)

2.2 Methods to improve fine particle recovery

Two main approaches were used to improve the fine particle recovery in the flotation process (ex. Song *et al.*, 2001):

- Increase the particle size;
- Decrease the bubble size.

The various methods to achieve these two points are discussed in sections 3 and 4.

3 Floc flotation

3.1 Selective polymer flocculation

The aggregation of fine particles into bigger particles followed by the aggregate flotation is often called “floc flotation” (Trahar and Warren, 1976). A method used to produce flocs is by selective polymer flocculation. Soto and Barbery (1988) used polyethylenoxide and polyacrylamides to flocculate selectively calcite, talc, kaolinite and alumite, as single minerals and as two-mineral mixtures. The particle size tested was below 20 μm. They observed good flocculation in single mineral tests but the selectivity and recovery dropped when two minerals were combined, possibly because of heterocoagulation and entrapment (Soto and Barbery, 1988). However, floc flotation experiments of talc in a laboratory column gave better recovery than without polymers, or using a mechanical cell. Cellulose xanthate was shown to be selective towards base metal sulphides, the polymer gave high sphalerite and galena recoveries but low grades (Mandre and Panigrahi, 1997).

An improvement on this process is made when a surfactant is used, especially for hydrophilic minerals. Zhong, Chen and Chen (1988) used polyacrylamides and several collector surfactants to flocculate hematite, cassiterite, calcite, quartz, ilmenite and feldspar, and found that the adsorption of the polymer on the mineral surface was higher in the presence of a collector. A collector with a higher selectivity produced a more effective and selective flocculation (Zhong, Chen and Chen, 1988).

Sadowski and Polowczyk (2004) used commercial flocculants (MAGNAFLOC 1440, E-10, 351) in combination with flotation collectors to float ZnO and MgO. They identified two steps in the process: 1) the adsorption of collector on the mineral surface, 2) creation of a polymer-surfactant complex by flocculant addition. It was observed that the presence of excess collector in the solution when the polymer is added resulted in a second layer of adsorbed surfactant on the polymer, making the mineral particle hydrophobic. In that case, the recovery by flotation was greater than when no collector layer is formed on the polymer and the particle remains hydrophilic (Sadowski and Polowczyk, 2004).

Wightman, Grano and Ralston (2000) proposed a method to reduce the loss of selectivity by entrapment of the gangue in the flocs. Figure 3 shows the procedure that involves separation of coarse and fine fractions, followed by selective flocculation of the coarse particles before the fines are re-mixed in and attracted to the coarse flocculated particles. An advantage of this method is the greater availability of the mineral particle surfaces for collector adsorption, since fine particles are not covered with flocculant.

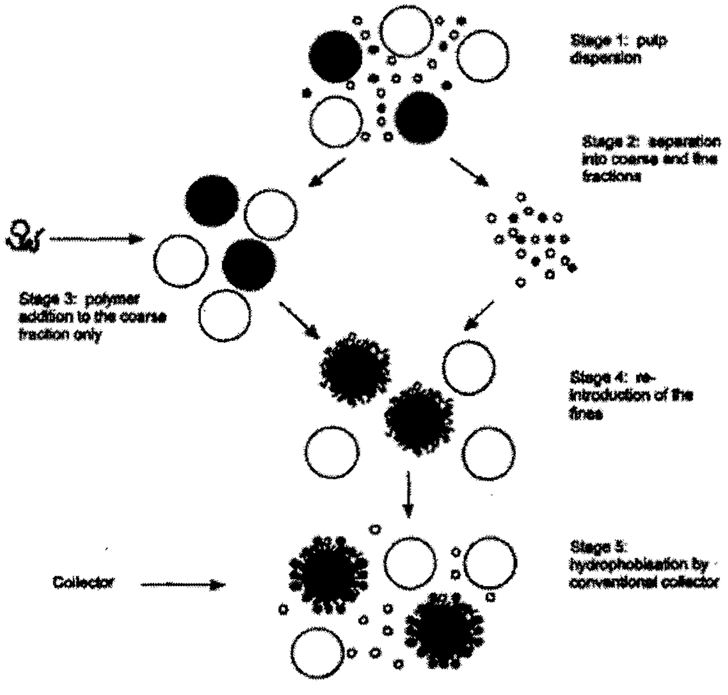


Figure 3 : The procedure to obtain coarse-fine galena aggregates in a mixed mineral system (open circles are quartz, dark circles are galena) from Wightman, Grano and Ralston (2000)

Experiments performed on a galena-quartz system using cellulose xanthate as the flocculant and KEX as the collector showed that the “split method” gave higher galena selectivity but lower recovery than without the polymer, and a slight improvement of the ultrafine (-5 µm) rate constant. Performance of the split system was higher than the conventional flocculant addition method (Wightman, Grano and Ralston, 2000).

The difficulties encountered with polymer flocculation, i.e. flocculant selectivity and hydrophilicity, limit the applicability of this method (Rubio and Hoberg, 1993).

3.2 Hydrophobic aggregation

Aggregation of particles can be induced in many ways, flocculation being one of them and coagulation another one. Coagulation occurs when the repulsive force between the particles are reduced, either by charge neutralization or reduction of the electrical double layer (Mirnezami, Hashemi and Finch, 2004). Surface charge is measured by the zeta-potential. Vergouw *et al.* (1998a, 1998b) observed that the maximum aggregation of particles of pyrite and galena occurred at a zeta-potential close to 0, called the iso-electric point (iep), corresponding to the phenomenon of homocoagulation. Similar results were obtained by Mirnezami, Hashemi and Finch (2004). This type of aggregation follows the DLVO (Derjaguin, Landau, Verwey, Overbeek) theory (Lu and Dai, 1988), which implies that the potential energy of particles (V_T) is the sum of the repulsive electrostatic interaction energy (V_E) and the attractive London – van der Waals dispersion energy (V_D): $V_T = V_E + V_D$ (Honaker, Luttrell and Yoon, 2004).

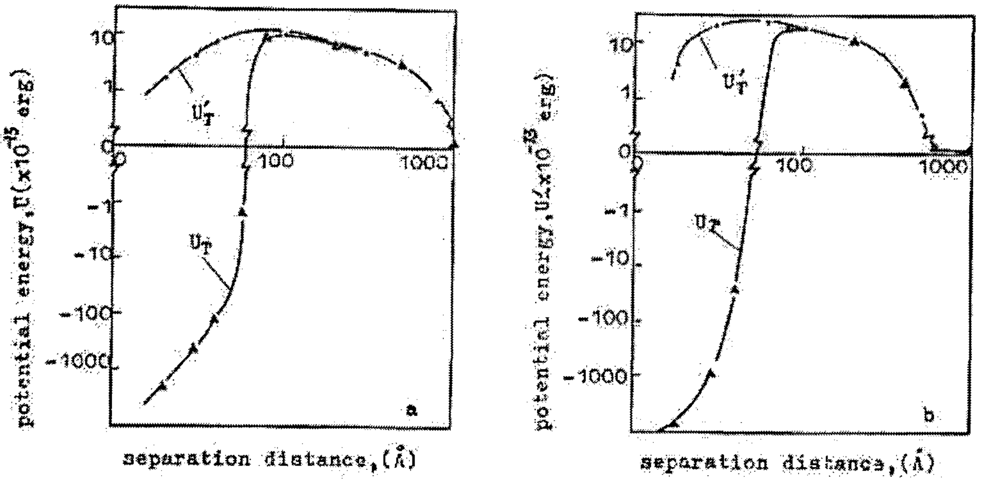
This equation is valid for moderately hydrophilic and weakly hydrophobic particle suspensions of colloidal size. When the particles are more strongly hydrophobic, aggregation is possible at high zeta-potentials (ex. Fuesternau, Li and Hanson, 1988; Lu and Dai, 1988; Akdemir, 1997; Song *et al.*, 2000) and the classical DLVO theory is no longer valid. Another term V_H was added to the total energy equation to account for the hydrophobic attraction energy, giving the following equation as the extended DLVO theory: $V_T = V_E + V_D + V_H$ (Honaker, Luttrell and Yoon, 2004). This version of the DLVO theory was verified experimentally using -5 μm coal and graphite particles by Hu and Qiu (2003) and by Honaker, Luttrell and Yoon (2004) using 1.3 μm alumina. The hydrophobic energy term (V_H) is several hundred times higher than V_E and V_D and becomes a significant factor in aggregation of hydrophobic particles (Lu and Dai, 1988, Subrahmanyam and Forssberg, 1990).

The origin of the hydrophobic attraction between particles is still under debate. A recent theory suggests that the presence of very small bubbles (nanobubbles) on the particle surfaces promotes bridging and attraction (Attard, 2003; Nguyen *et al.*, 2003). However, convincing evidence is still missing to confirm the theory of nanobbubles (Ralston, Fornasiero and Mishchuck, 2001; Evans, Craig and Senden, 2004).

3.2.1 Shear flocculation

Particles in a suspension are not automatically aggregated because there exists a potential energy barrier that keeps them dispersed. The potential energy varies with the distance between two particles, as shown in Figure 4, in which hydrophobic particles would aggregate when within 60-70 Å of each other (Lu and Dai, 1988).

For particles to aggregate when they are dispersed, the corresponding potential energy barrier must be overcome by applying extra energy to the particles, usually by intense stirring (Lu and Dai, 1988; Fuerstenau, Li and Hanson, 1988; Subrahmanyam and Forssberg, 1990; Song *et al.*, 2000, 2001; Xu *et al.*, 2004). The process is called "shear flocculation" or "high intensity conditioning". Two conditions are necessary for shear flocculation to take place: 1) particles must be hydrophobic, and 2) energy must be added to overcome the potential energy barrier (Subrahmanyam and Forssberg, 1990; Song *et al.*, 2000, 2001; Xu *et al.*, 2004). The use of a non polar oil was also investigated by a number of authors to enhance the shear flocculation process (Lu and Dai, 1988; Song *et al.*, 2000, 2001).



a--- rhodochrosite, equivalent radius being 8.99×10^{-5} cm;

b--- quartz, equivalent radius being 1.43×10^{-4} cm;

U_T' --- calculated in terms of DLVO theory;

U_T --- calculated in terms of hydrophobic interaction theory.

Figure 4 : Potential energy curves between hydrophobic particles (from Lu and Dai, 1988)

3.2.1.1 Agitation speed

Fuerstenau, Li and Hanson (1988) worked on the shear flocculation of hematite particles less than 20 μm. Their results are presented in Figures 5 and 6. In Figure 5 a shift in the size distribution is observed, with the lowest amount of fines obtained with an agitation rate of 1200 rpm (tested in a 200 mL baffled vessel). However, agitation of 2000 rpm was detrimental to the aggregation of the fines as observed by the high peak at 10 µm. This phenomenon of possible floc breakage at high shear rates was also observed by other authors (Lu and Dai, 1988; Song *et al.*, 2000, 2001; Xu *et al.*, 2004). Flotation tests, presented in Figure 6, show the improvement in recovery with intense agitation prior to flotation (1200 rpm in this case). The advantage of shear flocculation is particularly apparent for short flotation times and a net gain on ultimate recovery.

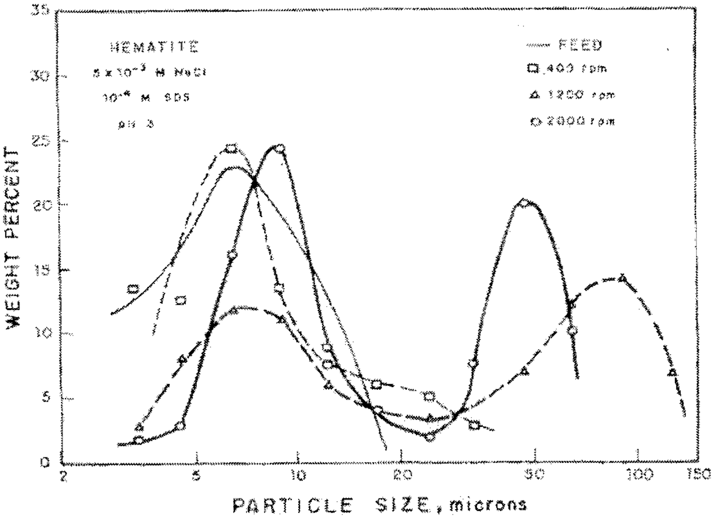


Figure 5 : Size distribution of hematite particles and aggregates produced at different agitation speeds (from Fuerstenau, Li and Hanson, 1988)

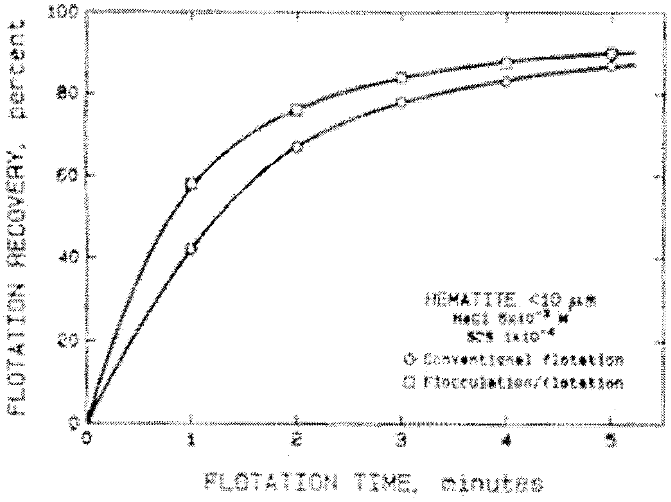


Figure 6 : Flotation recovery of hematite as a function of time, with and without flocculation (from Fuerstenau, Li and Hanson, 1988)

Akdemir (1997) also worked on shear flocculation of hematite but found no significant effect of agitation speed between 100 rpm and 2000 rpm. The collectors used were a petroleum sulfonate (Aero 801) and dodecylamine acetate (DDA) whereas Fuerstenau, Li and Hanson used sodium dodecyl sulphate. Collector concentration had an important influence on flocculation and flotation recovery, implying that hydrophobicity is a major factor in hydrophobic aggregation, perhaps more significant than energy input (Akdemir, 1997). However, not all experimental results agree on this point (see Subrahmanyam and Forssberg, 1990; Song *et al.*, 2000, 2001; Xu *et al.*, 2004).

Song *et al.* (2000, 2001) worked with galena and sphalerite fines at agitation speeds up to 1400 rpm. The floc size increased up to a maximum and then decreased, the corresponding effect on flotability is presented in Figure 7. A certain amount of energy is required for hydrophobic flocculation to take place, but too high energy levels rupture the flocs and the mean floc size decreases (Song *et al.*, 2000). Stirring duration was evaluated using galena fines at a fixed agitation speed. The floc size increased with stirring time up to a constant value, suggesting that once sufficient energy is provided to the particles, additional agitation at the same speed does not break or grow flocs (Song *et al.*, 2000).

The impeller type was found to have an influence on shear flocculation by optimizing the energy dispersion pattern (Xu et al., 2004). Indeed, at the same energy input (power draw) of 7,5 kWh/m^3, the impeller type was found to affect the aggregate size, the impeller causing bigger flocs having the higher energy dissipation rate.

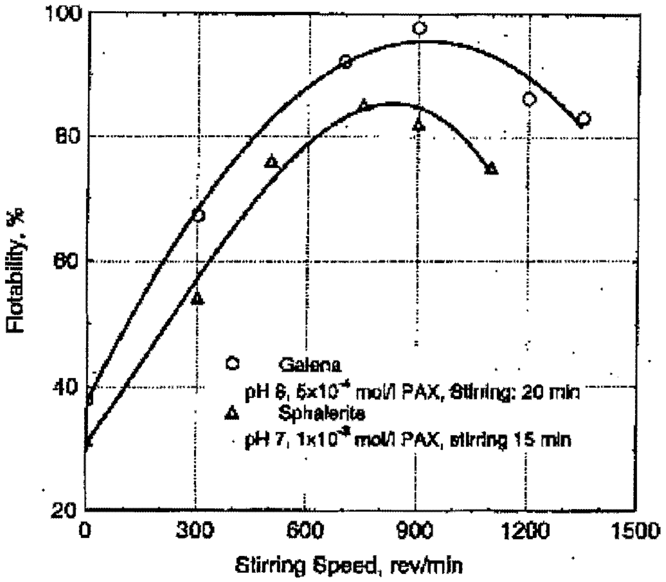


Figure 7 :Flotability of flocculated galena and sphalerite fines induced by PAX as a function of stirring speed in the hydrophobic flocculation step (from Song *et al.*, 2001)

3.2.1.2 Particle size

Results by Warren reported by Subrahmanyam and Forssberg (1990) illustrate the influence of particle size and agitation speed on the aggregation of particles, expressed in Table 2. The particle size effect was also demonstrated by Song *et al.*

(2000) using galena fines and potassium amyl xanthate, 10 minutes of stirring at 500 to 700 rpm. The strongest aggregation was obtained with -2μm galena, producing flocs 10 times larger than the original particles (95% between 10 and 30 μm). Figure 8 presents the aggregation degree, which is related to the number of particles in the aggregate, for three size classes (Song *et al.*, 2000). Shear flocculation appears to be more effective on very fine particles, the reasons are currently being investigated.

Table 2 : Effect of particle size and stirring speed on aggregation of particles in shear flocculation of sheelite (Subrahmanyam and Forssberg, 1990)

Particle size (μm)	Stirring speed (rpm)	Observation
Single size fractions		
1 (0.75-1.4)	1000	Aggregation
	850	Negligible aggregation after 90 min stirring in 10^{-4} M Na oleate
7.6-11.1	850	Particles already aggregated
16.6-23.6	850	Slight aggregation
23.6-31.2	850	Negligible aggregation
38.0-44.0	850	No aggregation
1.0-12.0	1700	Aggregation
Mixtures of particles		
0.75-1.4 and 16.6-23.6	850	Ultrafines did not aggregate themselves but adhered to coarse particles ("slime coating")
0.75-1.4 and 7.6-11.1	1700	Three types of aggregates, i.e. fine-fine, coarse-fine and coarse-coarse

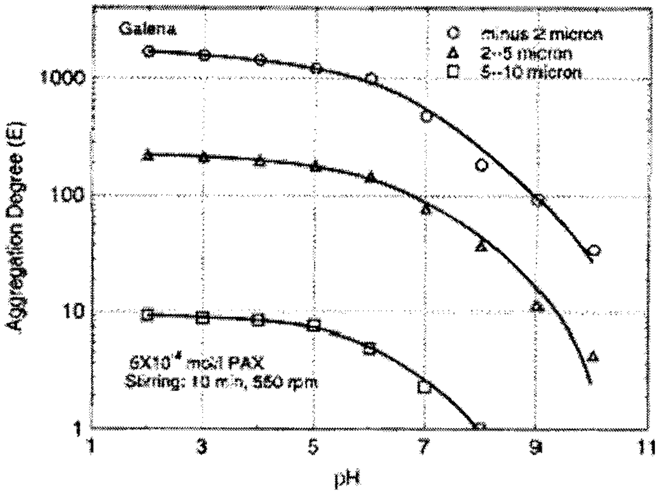


Figure 8 : Aggregation degree of the hydrophobic flocculation of the three galena samples induced by PAX as a function of pH (from Song *et al.*, 2000)

High intensity agitation was also related to surface cleaning and improvement of flotation behaviour for particles larger than 10 µm (Xu *et al.*, 2004). A process called "sulphidisation" was used to improve the recovery of fines (-20 µm) at the OK Tedi plant. Sulphidisation involves the cleaning of oxidized surfaces by sulphide ions (NaHS) during conditioning. The result was an average increase of 3,4% in copper recovery for porphyry sulphide ore blends (Orwe, Grano and Ralston, 1998).

The combination of high intensity conditioning and split conditioning was investigated by Small *et al.* (1997), with the size separation made by conventional hydrocyclone. The fine fraction had a P_{80} of 10 µm while the coarse fraction had a P_{80} of 80 µm. The galena selectivity was improved in the overflow stream however it was lower in the underflow stream. High intensity conditioning improved galena flotation rate but separate conditioning of the size fractions did not improve overall flotation performance due to the decreased selectivity (Small *et al.*, 1997).

3.2.1.3 Collector dosage

The degree of hydrophobicity of the fine particles is an important parameter in hydrophobic flocculation (Subrahmanyam and Forssberg, 1990; Akdemir, 1997). Song *et al.* (2000, 2001) found that the flotability of galena and sphalerite fines increased with increasing collector concentration (see Figure 9). The higher flotation response was linked to enhanced floc hydrophobicity and larger flocs, the adsorption of collector on the surfaces increased the hydrophobic attraction and promoted flocculation (Song *et al.*, 2000, 2001).

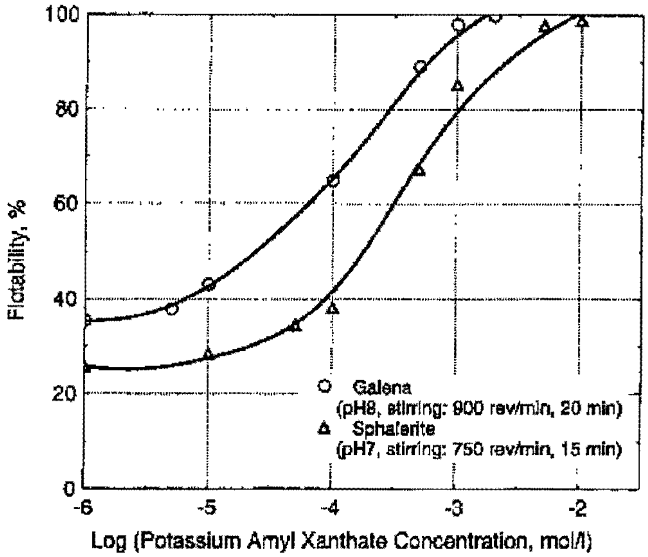


Figure 9 : Flotability of flocculated galena and sphalerite fines induced by PAX as a function of PAX concentration (from Song *et al.*, 2001)

3.2.1.4 Non polar oil addition

The addition of non polar oil during shear flocculation enhances the flotability of the flocs, as observed by Song *et al.* (2000, 2001). Kerosene addition also reduced the collector requirements, thus it can replace a significant amount of collector (2x10^{-3}

mol/L PAX vs. 1x10^{-4} mol/L PAX with 200 mg/L kerosene). Non polar oil makes larger and denser flocs, and increases their hydrophobicity (Lu and Dai, 1988; Song et al., 2000, 2001). Oil droplets adhere to solid particles and form films, increasing particle hydrophobicity and promoting the formation of oil bridges (Lu and Dai, 1988; Song et al., 2000, 2001). The oil bridges make the flocs stronger and more compact, which in turn make them more resistant to breakage (Song *et al.*, 2000, 2001).

3.2.2 Carrier flotation

Carrier flotation is an approach based on the use of slime coating integrated into shear flocculation to improve fine particle recovery (Fuerstenau, Li and Hanson, 1988). Autogenous carrier flotation is the process by which fine mineral particles are aggregated onto coarse particles of the same mineral, with the coarse particles acting as seed (Valderrama and Rubio, 1998). Fuerstenau, Li and Hanson (1988) showed an increase in grade and recovery of hematite from quartz compared with standard shear flocculation conditioning. Three requirements were identified for carrier flotation: 1) coarse particles present in the pulp, 2) high intensity agitation, and 3) coarse and fine particles must be hydrophobic (Subrahmanyam and Forssberg, 1990).

The collision rate between coarse and fine particles is greater than between fine and fine particles, and fine particles adhere well on the coarse particle surfaces (Wang, Qu and Hu, 1988; Valderrama and Rubio, 1988; Lange, Skinner and Smart, 1997; Hu, Qiu and Miller, 2003). Figure 10 presents the interactions of coarse and fine particles with increasing energy input during conditioning. The highest improvement in recovery occurs at the third section of energy input where coarse particles are being coated by fines and also fine particles form themselves aggregates. The authors correlated these mechanisms to experimental results in which recovery of gold fines was increased by 24%, gold concentrate grade by 50% and the flotation rates were 2 to 3 times faster than without high intensity conditioning (Valderrama and Rubio, 1998). An energy input of 3 to 4 kWh/m^3 of pulp yielded the best results, and it was suggested to recirculate the first concentrate as seeds during conditioning (Valderrama and Rubio, 1998).

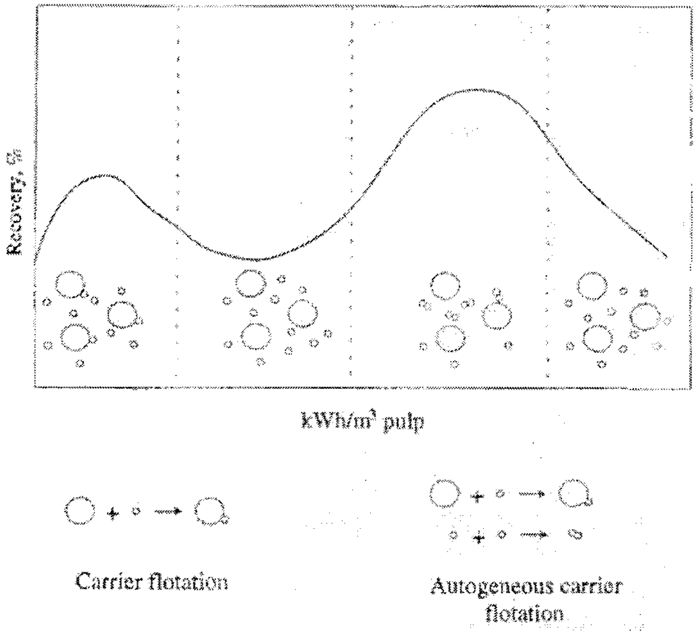


Figure 10 : Particle aggregation phenomena during the conditioning stage and mechanisms of flotation (from Valderrama and Rubio, 1998)

However, the amount of coarse versus fine particles has an influence on the flotation response, as demonstrated by Lange, Skinner and Smart (1997) using sphalerite. They observed that higher coarse : fine surface area ratios give better results, and that a coarse : fine ratio of less than 25% caused significant depression of coarse sphalerite. In this study, fines were <20 μm and coarse particles between 38 and 75 μm.

Particles have a boundary layer between themselves and the fluid medium. The state of the boundary layer, laminar or turbulent, is given by the Reynolds number (Re) which is a function of the particle size. Coarse particles and large aggregates frequently have high Re, therefore vortices and eddies are produced behind the moving particle (Wang, Qu and Hu, 1988; Hu, Qiu and Miller, 2003). These small eddies are the site of fine particle aggregation and result in the growth of middle size flocs (Hu, Qiu and Miller, 2003).

The previous discussion emphasized on carrier flotation with the same mineral as "carrier" and "carried". Work was conducted using different material as carrier, such as Rubio and Hoberg (1993) who worked with polypropylene spheres. The carrier was made hydrophobic with oleic acid. Results are concordant with other shear flocculation / carrier flotation studies in terms of energy input, hydrophobicity and flotation recovery. Furthermore, the duration of conditioning and flotation were reduced and very high selectivity was obtained (Rubio and Hoberg, 1993). However, the polypropylene spheres must be separated after flotation by stirring in alkaline media (Rubio and Hoberg, 1993).

3.2.3 Oil agglomeration

Oil agglomeration, also called spherical agglomeration, is an extension of shear flocculation. It already has practical use in fine coal recovery but recent laboratory work was performed on other minerals (House and Veal, 1989; Laskowski and Lopez-Valdiviesco, 2004; Wu, Monhemius and Gochin, 2005). The process consists in the formation of agglomerated by mixing hydrophobic solids with immiscible oil (House and Veal, 1989; Wu, Monhemius and Gochin, 2005). The requirements are hydrophobic particles, immiscible bridging oil and high intensity agitation (Sadowski and Smith, 1988; Laskowski and Lopez-Valdiviesco, 2004). Oil consumption is high historically, in the range of 10% by weight of solids, recent work is in the 0.5 to 2% range (Laskowski and Lopez-Valdiviesco, 2004). Due to its natural hydrophobicity, coal is the choice material for oil agglomeration, but other minerals can become suitable candidates provided they are rendered hydrophobic by surfactant addition

(Laskowski and Lopez-Valdiviesco, 2004). The agglomerates are usually recovered by screening.

The oil agglomeration process was found to be efficient with calcite and barite using sodium oleate as the surfactant and n-heptane as the oil phase (Sadowski and Smith, 1988). House and Veal (1989) conducted testwork on the agglomeration of chalcopyrite and pyrite using n-tetradecane and gas oil with xanthates. Copper recovey reached 85% after 10 minutes (stirring at 1000 rpm) and 90% after 60 minutes, with a grade of 30% (95% chalcopyrite), the sieve size being 250 μm. Pyrite was agglomerated faster; 92% recovery after 20 minutes. The optimal agitation speed was between 750 and 1000 rpm. They also recovered copper and zinc selectively from a mixed sulphide / siliceous gangue mixture (d_{80} = 23 μm) (House and Veal, 1989).

The properties of the oil are important for the success of the agglomeration process. Indeed, medium surface tension and medium viscosity are more effective (in the case of coal), furthermore a compatibility factor involving polarity between oil and solid must be taken into account (Wu, Monhemius and Gochin, 2005).

A variation of this process is the "liquid-liquid extraction" process in which fine solid particles are displaced from a water phase to an oil phase by mixing and phase separation (Kusaka *et al.*, 1994). The hydrocarbon coats the solid hydrophobic particles during agitation and then the two phases are separated. The hydrophobic particles are present in the oil phase and the hydrophilic particles remain in the water phase. Kusaka *et al.* (1994) worked on collectorless liquid-liquid extraction of galena from quartz using several hydrocarbons as the oil phase. They obtained high galena recovery and selectivity with many oil types at a oil / water ratio of 10/50 and 0.99 wt% aqueous particle concentration (Kusaka *et al.*, 1994).

A new approach involving flotation and oil was introduced by Liu *et al.* (2002). Air bubbles coated by a thin film of oil containing a collector, called "reactive oily bubbles", are more suitable to fine particle recovery because of the formation of compact gas/oil/solid agglomerates (Liu *et al.* 2002). Testwork was performed with fine silica (< 40 μm) and fine sphalerite and galena (<75μm), which is coarser than most experimental results presented in this document. It was observed that the type of collector had an influence on the flotation recovery and its dosage was minimal compared with standard flotation. The flotation recovery and selectivity was high and could be optimized using standard flotation modifiers such as activators and depressants (Liu *et al.* 2002).

To summarize the section on hydrophobic aggregation, the following points are emphasized:

- Aggregation of hydrophobic particles occurs when the potential energy barrier is overcome;
- Agitation and hydrophobicity are key parameters in shear flocculation;
- Coarse particles can be used as carriers to enhance shear flocculation;
- Addition of oil can be beneficial.

3.3 Microbially-induced flocculation

The use of micro-organisms in flotation and flocculation was recently investigated. It was observed that interactions between bacteria and mineral surfaces change the hydrophobicity of the mineral (Somasundaran, Ren and Rao, 1998; Patra and Natarajan, 2003, 2004). Two types of interactions were identified:

1. direct: bacteria adhere to mineral surfaces and change the surface properties;
2. indirect: biological products, such as secreted metabolites, induce surface properties modifications.

Flotation and flocculation are two processes that can benefit from the use of micro-organisms. Work conducted using *Thiobacillus ferrooxidans* highlighted the bacteria ability to depress pyrite in flotation, suggesting a possible use in coal beneficiation (Somasundaran, Ren and Rao, 1998). Moreover, *Mycobacterium phlei* induced selective coal flocculation from pyrite and ash (Somasundaran, Ren and Rao, 1998).

Interesting work was performed by Patra and Natarajan (2003) on bioflocculation and flotation of pyrite from oxide gangue in the context of environmental desulphurization. For the flocculation tests, pyrite and quartz with an average particle size of 5 µm, and 3 µm calcite were interacted with *Bacillus polymyxa* and its extracellular protein. It was found that pyrite had a higher density of adsorbed bacteria, making selective flocculation of pyrite from a mixture of quartz and calcite effective (Patra and Natarajan, 2003). The flotation experiments were performed on coarser material (-103 +65 µm). Pyrite was depressed by bacterial cells and quartz was made more hydrophobic (Patra and Natarajan, 2003).

Similar work was conducted with *Paenibacillus polymyxa* and pyrite and sphalerite in which the two minerals were separated by selective flocculation and flotation (flotation was again performed on coarser particles) (Patra and Natarajan, 2004). Pyrite was flocculated while sphalerite was dispersed by bacteria and protein at pH 8-9. Furthermore, bacterial interaction made pyrite hydrophilic, even in the presence of xanthate. Sphalerite flotation was improved with bacteria and xanthate, therefore sphalerite and pyrite were effectively separated by flotation with *P. polymyxa* (Patra and Natarajan, 2004).

4 Mechanical aspects

4.1 Bubble size – influence on fine particle flotation

Collection of a particle requires three steps: the particle and the bubble must approach each other, then the liquid film between the bubble and the particle gets thinner, and finally the liquid film ruptures and a three phase contact is established (Tortorelli *et al.*, 1997; Pyke, Fornasiero and Ralston, 2003). The collection efficiency of a bubble and a particle is described as: $E_{coll} = E_c E_a E_s$, where E_c is the collision efficiency, E_a is the attachment efficiency and E_s is the stability efficiency. Figure 11 shows that E_c increases with particle size whereas E_a and E_s decrease with particle size, the combination of the three curves resulting in the flotation rate constant k being low for fine and coarse particle size (Pyke, Fornasiero and Ralston, 2003).

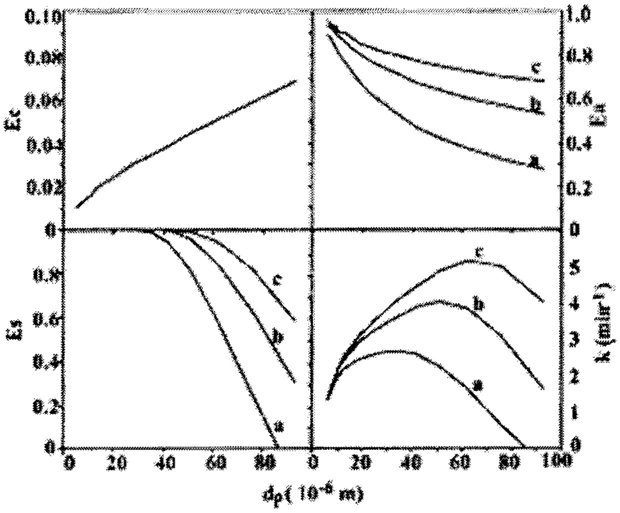


Figure 11 : Calculated collision (E_c), attachment (E_a), and stability (E_s) efficiencies and flotation rate constant k as a function of quartz particle diameter and induction time (from Pyke, Fornasiero and Ralston, 2003)

It is generally accepted that the main reason for the low flotation response of fine particles is the decrease in the probability of collision between particles and bubbles as the particle size is decreased (Yoon and Luttrell, 1989; Matis, Gallios and Kydros, 1993; Yoon, 1993; Koh and Schwarz, 2003). Yoon and Luttrell (1989) expressed the collision probability for intermediate bubble Reynolds ($\mathrm{Re} \approx 0$) number as:

$$P_c = \left(1.5 + \frac{4}{15}\mathrm{Re}_b^{\;0.72}\right)\frac{d_p^{\;2}}{d_b^{\;2}}$$

This equation is valid for particles $d_p < 100$ µm and bubbles $d_b < 1$ mm. The collision probability is therefore a function of the bubble diameter to the power -2 (Yoon, 1993; Koh and Schwarz, 2003). To improve the flotation recovery of fine particles, the use of small bubbles is warranted since it will increase the probability of collision between the particles and the bubbles (Ahmed and Jameson, 1985; Yoon and Luttrell, 1989; Yoon, 1993).

Fine particles have low inertia and will tend to follow the liquid streamlines around the bubble (Yoon and Luttrell, 1989; Matis, Gallios and Kydros, 1993; Ityokumbul *et al.*, 2000; Mileva and Nikolov, 2003). A mineral particle will follow the liquid streamlines for a given bubble size if:

$$\frac{\rho_p d_p^{\;2} U_b}{9\mu d_b} << 1 \text{ (Ityokumbul } et\ al.\text{, 2000).}$$

where ρ_p = particle density; d_p = particle diameter; U_b = bubble rise velocity; μ = slurry viscosity; and d_b = bubble diameter. Therefore, fine particles will be able to collide with small bubbles only.

4.2 Bubble generation

Several methods exist to generate bubbles in a flotation machine. Some of these methods that can produce small bubbles are presented.

4.2.1 Dispersed air flotation

The conventional method of generating bubbles is by blowing air close to an impeller. To improve the probability of collision between fine particles and bubbles, the energy input to the flotation process is higher than that required for coarser particles. A higher power input given a specific air flow rate generates more bubbles (Schubert, 1999). However, the size of the bubbles and their number has an influence on fine particle recovery, as seen in the previous section. Research is conducted to develop a system that generates very fine bubbles in a standard flotation cell. Such a system called "colloidal gas aphrons" (CGA) was presented by Cilliers and Bradshaw (1996). CGA are generated by a spinning disk impeller and reached a size of 50 μm, compared to 500 μm using a standard impeller (Cilliers and Bradshaw, 1996). Tests were performed with -38 μm pyrite in 3-litre cells at a pH of 4. The CGA impeller resulted in higher froth grades at equivalent recoveries, overall recovery was similar for CGA and conventional systems, but the flotation rate was reduced with CGA. The reason could be the lower buoyancy of the very small bubbles so they need time to coalesce to float (Cilliers and Bradshaw, 1996).

4.2.2 Dissolved air and induced air flotation

Dissolved air flotation (DAF) is commonly used in wastewater treatment because of the very fine bubbles (< 10 μm) produced and its relative non-selectivity (Jameson, 1999). A stream of pressurized air-saturated water is introduced into a tank in which the pressure is reduced and air bubbles nucleate. Nucleation often occurs on solid surfaces, sometimes bubbles are entrapped in flocs, therefore any particle can be recovered (Matis, Gallios and Kydros, 1993; Jameson, 1999). DAF requires long residence times and large cells, and is not commonly used in mineral flotation (Jameson, 1999).

A more applicable approach is induced air flotation (IAF). In this process, the pulp is fed at very high speed and air is contacted with the pulp by induction (Evans, Atkinson and Jameson, 1996). A popular flotation machine using this principle is the Jameson cell, shown in Figure 12, encountered in mineral and coal flotation plants and wastewater treatment plants (Evans, Atkinson and Jameson, 1996; Jameson, 1999). The bubble size is larger in IAF than DAF but the amount of air that can be dissolved in DAF stream is much lower than the air flow reachable in IAF (Jameson, 1999). Furthermore, the zone of contact between bubbles and solid particles is under high shear, which increases the probability of collision of fine particles with bubbles, while the separation zone is quiescent (Evans, Atkinson and Jameson, 1996).

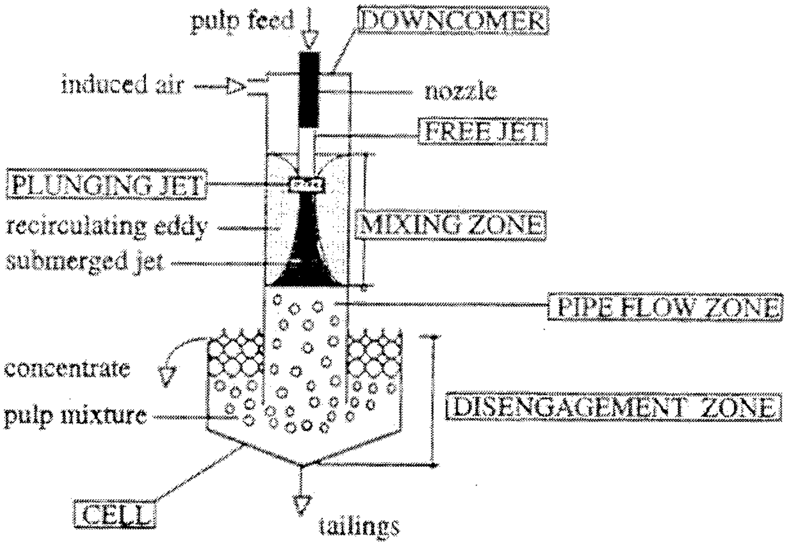


Figure 12 : Schematic of Jameson cell showing hydrodynamic regions (from Evans, Atkinson and Jameson, 1996)

Another machine using IAF is the Fastfloat cell, a prototype improved fine (-9 μm) marmatite recovery at Pasminco Mining, Broken Hill. Air is drawn from the atmosphere by thin liquid jets and very small bubbles are produced when the slower pulp comes in contact with the high speed induced air (Chudacek *et al.*, 1997).

4.2.3 Hydrodynamic cavitation

Hydrodynamic cavitation is a process by which bubbles are formed when the local pressure is reduced to a point that breaks a liquid-liquid or solid-liquid interface in a flowing film (Zhou *et al.*, 1997). Cavitation is susceptible to occur when the fluid flow is accelerated by going from a large pipe into a smaller section, as presented in Figure 13 (Zhou *et al.*, 1997). The result is fine bubbles that nucleate on the solid particles or in the fluid stream. Tests performed with fine silica (-5 µm) and zinc precipitates showed a definite increase in flotation recovery when a cavitation tube was inserted in the flotation cell (Zhou *et al.*, 1997). Hydrodynamic cavitation could be present in Jameson cells, contact cells and microcells (Zhou *et al.*, 1997).

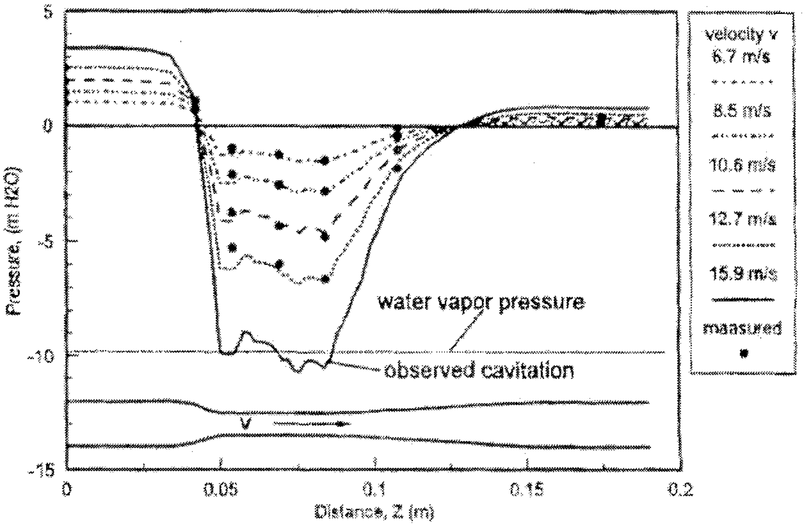


Figure 13 : Comparison of predicted and measured pressure changes along the cavitation tube (no chemicals, no air added) (from Zhou *et al.*, 1997)

4.2.4 Electroflotation

In electroflotation, bubbles are generated by the electrolytic breakdown of water from which oxygen and hydrogen are produced (Ledesma and Guzmán, 1988):

Anodic reaction: $4\ OH^- \rightarrow 2\ H_2O + O_{2(g)} + 4\ e^-$

Cathodic reaction: $4\ H^+ + 4\ e^- \rightarrow 2\ H_{2(g)}$

The bubbles produced are smaller than those from standard bubble generators, the size being a function of current density and electrode material, among others (Bhaskar Raju and Khangaonkar, 1982; Ledesma and Guzmán, 1988). The experiments performed using electroflotation were conducted in modified Hallimond tubes.

Work on chalcopyrite (Bhaskar Raju and Khangaonkar, 1982) and chalcocite (Ledesma and Guzmán, 1988) fines showed an increase in recovery using electrolytic oxygen bubbles. Testwork was also performed on pyrite of larger particle size (-45 +25 μm) and a maximum recovery of 45% was achieved (Matis, Gallios and Kydros, 1993). An interesting comparison of the performance of electroflotation versus dispersed-air flotation of magnesite and dolomite is presented in Figure 14.

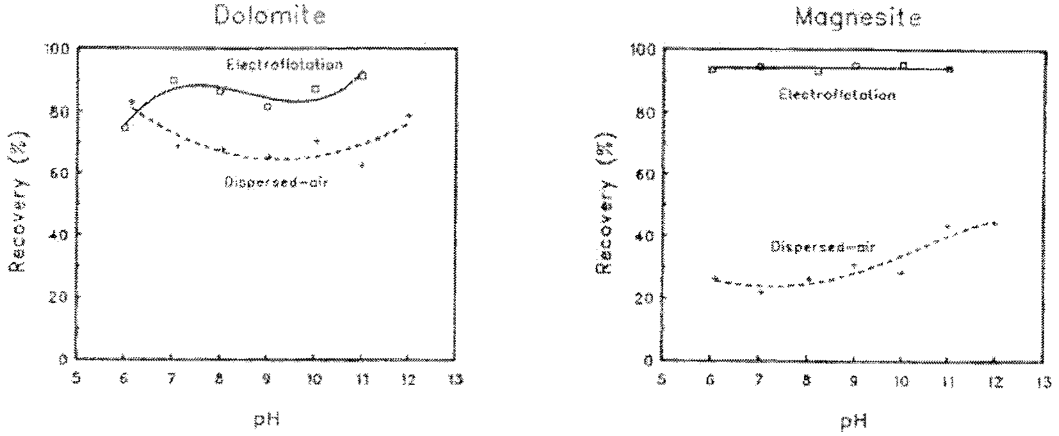


Figure 14 : Comparison of electrolytic flotation of dolomite and magnesite fines with dispersed-air flotation (both of Hallimond type): effect of pH (from Matis, Gallios and Kydros, 1993)

Collector adsorption on the metal sulphide surface was identified as being an electrochemical process during which oxygen is reduced at the particle surface creating anodic sites where xanthate is oxidized and adsorbed (Bhaskar Raju and

Khangaonkar, 1982; Matis, Gallios and Kydros, 1993). The anodic potential applied for electrolytic production of oxygen bubbles is able to create additional anodic sites and further increase collector adsorption and particle hydrophobicity. Results by Bhaskar Raju and Khangaonkar (1982) and Ledesma and Guzmán (1988) support this phenomenon.

4.3 Flotation machines

4.3.1 Columns

The poor recovery in fine particle sizes with conventional mechanical flotation cells is attributed to the turbulent conditions inside the cell (Yoon and Luttell, 1989). Collision between bubbles and particles are relatively random (Keyser *et al.*, 1996). Flotation columns are a type of machine that evolved from these limitations. A conventional column cell has a deep froth zone and wash water sprays that allow achieving a high concentrate grade by reducing entrainment (Finch and Dobby, 1990; Harris *et al.*, 1992). The feed is added in the top third of the column and the particles settle among rising bubbles in a quiescent state, providing good conditions for increased fine particle – bubble collision probability (Finch and Dobby, 1990; Harris *et al.*, 1992; Keyser *et al.*, 1996). Fine particles benefit specifically from this countercurrent interaction since their settling rate is lower and their residence time in the column is higher, which enhances the probability of particle – bubble collision (Finch and Dobby, 1990; Keyser *et al.*, 1996). Many plants replaced mechanical cells by columns in the cleaning stage and obtained higher recovery of fine particles (ex. Hall and Averiss, 1988; Salim, 1996; Salas *et al.*, 1996). The bubble size, usually smaller in flotation columns, is a function of the gas rate (Castillo, Dobby and Finch, 1988; Finch and Dobby, 1990; Harvey, Jameson and Nguyen, 2003) and the type of sparger used (Yoon, 1993).

4.3.1.1 External contactor

The Jameson cell, already presented in section 4.2.2, is a modification of the conventional column cell with respect to the mechanism of air introduction into the column. Its advantages are mainly the small bubble size and improved bubble-particle collisions (see section 4.2.2).

The contact cell is similar to the Jameson cell in terms of the contact between air bubbles and the slurry being made outside the column itself. The contact cell uses an external contactor in which the slurry stream is passed through a porous tube where gas is injected (see Figure 15) (Tortorelli *et al.*, 1997; AMinpro, 2003). This provides a turbulent zone where bubbles and particles collide, and actual separation is done in the column. Research showed that the contact cell is effective in recovering fine particles because of the external contactor while the column part can handle larger particles, making the overall process a good candidate for a wide particle size range (Tortorelli *et al.*, 1997).

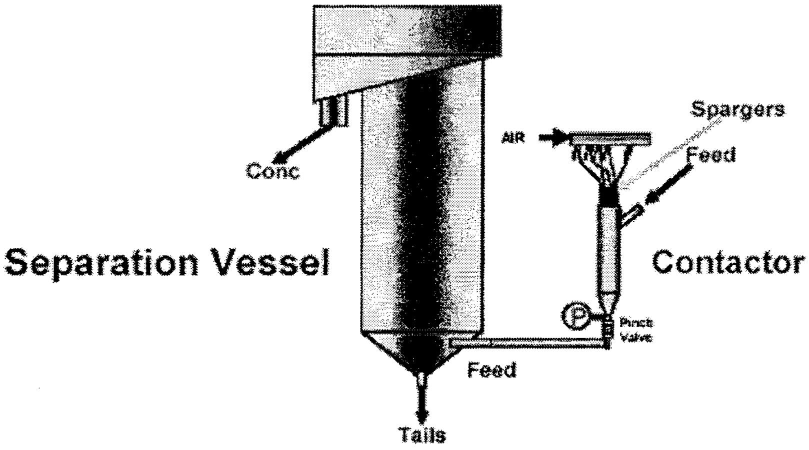


Figure 15 : A contact cell (from AMinpro, 2003)

4.3.1.2 Agitated columns

Harris *et al.* (1992) designed a laboratory agitated column cell in which an agitated zone with an impeller is added below a quiescent zone. The operation of the agitated column was compared to a batch mechanical cell and a conventional column for coal flotation. It was observed that the agitated column recovered selectively both fine and coarse particles. However, the fines selectivity was lower than with the conventional column. The authors suggested that optimization of the length of the quiescent zone should improve fine particle selectivity (Harris *et al.*, 1992).

In more recent testwork, a flotation column was equipped with static mixers et several points on its height to provide energy to promote fine particles-bubbles collisions (Ityokumbul *et al.*, 2000). Pyrite grade and recovery was evaluated as a function of agitation rate. The optimum agitation rate was identified as being 400 rpm, which translates (using air velocity) into a power input of 0.3 kW/m^3 (Ityokumbul *et al.*, 2000). Conventional mechanical cells have an energy rate of order of 1 kW/m^3, whereas 3 to 4 kW/m^3 was required for shear flocculation (Valderrama and Rubio, 1998; Ityokumbul *et al.*, 2000). The agitated column increased pyrite recovery by 13% at comparable grades, the reasons highlighted being the improved collision probability caused by moderate agitation (Ityokumbul *et al.*, 2000).

4.3.1.3 Microbubble flotation

In a quiescent flotation column, the use of small bubbles is beneficial to fine particle flotation. Several methods were developed to generate microbubbles in the range of 100 to 400 μm, most of those generators using high shear rates of the fluid where bubbles nucleate (Yoon, 1993). These microbubble generators are best used with flotation columns than mechanical cells, the benefits of very small bubbles being emphasized in relatively quiescent conditions (Yoon, 1993). The Microcel is a sparger device recently developed that gains popularity for use with flotation columns (Yoon, 1993).

4.3.2 Centrifugal force field flotation cells

The application of a centrifugal force field to the flotation process has several advantages: 1) bubbles are smaller; 2) particle-particle interaction is reduced; 3) froth production is reduced (Tils and Tels, 1992). The result is increased selectivity and faster flotation. Tils and Tels (1992) presented experimental results using a flotation cyclone and a flotation centrifuge to separate quartz and activated carbon, and gold from gangue. The centrifugal forces applied were between 50 and 200 times the ordinary gravitational force. They obtained fast flotation and good selectivity, at solid contents of 0.13% and 1.3%, definitely lower than industrial flotation practice.

4.3.3 Turbulent microflotation

Turbulent microflotation is a new concept by which flotation of very fine particles (≤ 1 µm) is achieved in a narrow tube (a pipeline) with small bubbles with an initial size ≤ 40 µm. The process is composed of four separate steps (Rulyov, 2001):

1. Coagulation or flocculation of the particles to reach a minimum size of 7 µm using reagents and agitation;
2. Heterocoagulation of particle aggregates and bubbles;
3. Aggregation and coalescence of particle-loaded bubbles and formation of froth;
4. Separation of froth from water by sedimentation.

Steps 1 to 3 must be performed under turbulent flow conditions. The process was tested on pilot scale using a pipeline with a cross-sectional area of 2 cm^2, 100 ppm of aluminum hydroxide precipitates to be floated, and hydrogen and oxygen bubbles generated by electrolytic decomposition of water. It was found that 98% of the fine aluminum hydroxide precipitates were recovered in 1.5 minute and bubble coalescence was an important parameter in flotation recovery (Rulyov, 2001).

5 Conclusion and recommendations

5.1 Summary

The objective of the document was to provide an answer to the question: "How can we improve fine pyrite and pyrrhotite recovery in the environmental desulphurization process by flotation?". Since work on environmental desulphurization is limited, information was found for generalized fine particle flotation irrelevant of the mineral treated. The methods found to increase fine particle recovery fell into two categories: 1) increasing particle size (1 in table), and 2) decreasing bubble size (2 in table). Table 3 presents the methods reviewed in this document.

Table 3 : Methods to improve fine particle recovery

Method	*Mechanism*
Selective polymer flocculation (1)	Polymer-induced flocs
Shear flocculation (1)	Hydrophobic flocculation induced by energy input
Carrier flotation (1)	Hydrophobic flocculation enhanced with coarse seed particles
Oil agglomeration (1)	Hydrophobic flocculation with immiscible oil
Microbially-induced flocculation (1)	Surface modifications by micro-organisms
Induced-air flotation (2)	High shear slurry-air contact zone, quiescent separation zone
Electroflotation (2)	O_2 and H_2 bubbles produced from water breakdown
Flotation columns (2)	Small rising bubbles in quiescent slurry
Microbubble generators (2)	High shear air-slurry contact, column spargers
Centrifugal force field flotation (2)	Application of centrifugal force to flotation
Turbulent microflotation (1, 2)	Very fine particle flotation in a tube using electrolytic bubbles

5.2 Application to environmental desulphurization

Most of the methods are irrelevant of the mineral type and surface state as long as the mineral is hydrophobic, either naturally or made hydrophobic by surfactants. Therefore, theoretically they could be applied to sulphide gangue (pyrite and pyrrhotite) in environmental desulphurization. Only the use of bacteria in flocculation and flotation was tested specifically for environmental desulfurization (Patra and Natarajan, 2003). The methods discussed previously are separated into two categories in terms of their applicability to environmental desulphurization: suggested methods, i.e. worth looking into when considering future testwork; and not suggested, i.e. not readily applicable.

5.2.1 Suggested methods

1. Column flotation, contact cell, Jameson cell

The first suggested modification to the standard desulphurization plant is to include column, contact or Jameson cells, by themselves or in combination with mechanical cells. The quiescent separation zone and smaller bubble size could increase fine pyrite and pyrrhotite recovery, with or without parameter modifications (pH, chemicals dosage).

2. Hydrophobic flocculation

Shear flocculation could be used to improve the recovery of fine pyrite and pyrrhotite. A high intensity conditioning unit is required, and in the case of carrier flotation, a coarse concentrate bleed can be recirculated to the conditioner. However, the use of oil to enhance shear flocculation or in oil agglomeration should be investigated further in terms of environmental impacts of oil-contaminated tailings before being pursued for environmental desulphurization.

3. Small bubble generators

Fine bubble spargers can be used in column cells, such as the Microcel. The optimization of the conditions for hydrodynamic cavitation in the flotation cells could improve fine particle recovery.

4. Microbially-induced flocculation

Patra and Natarajan are currently involved in research on micro-organisms and environmental desulphurization. It is suggested to keep up to date with their future publications and conferences on their work.

5.2.2 Methods not suggested

1. Polymer flocculation

The use of polymers was shown to be not very selective and the flocs produced were mostly hydrophilic. These factors, combined with the complex chemical composition of the slurry entering the environmental desulphurization circuit, limit the applicability of this method.

2. Electroflotation, centrifugal force field flotation and turbulent microflotation

Since these methods were mostly tested with conditions very different from the environmental desulphurization process, they are not readily applicable. It is not suggested to emphasize on these methods which are probably very costly.

5.3 Final comments

A suggested protocol for testwork to improve fine particle recovery in environmental desulphurization using hydrophobic flocculation is the following:

1. Determine design of experiment parameters for tests (eg. agitation speed and duration of agitation in conditioner).
2. Sample a tailing stream and perform a size-by-size analysis to obtain the sulphide distribution.
3. Perform a baseline flotation test in a Denver cell.

4. Perform tests according to design of experiment, using flotation parameters from baseline.
5. Repeat design of experiment.
6. Analyse the size fractions of the flotation concentrate and tail to obtain the sulphide distribution, for baseline and design of experiment tests.

Testwork could include agitation speed and duration experiments, impeller type and baffle arrangements in the conditioner, independent size class conditioning and flotation, and the effect of coarse "seed" particle addition.

The same test sequence can be performed in alternate flotation machines, such as a laboratory flotation column or contact cell. In such cases, parameters to consider in the testwork include gas rate and froth depth (for columns) and gas rate and pressure (for contact cells).

Flotation is a surface-based process for any particle size. A better understanding of the surface properties particular to fine particles through fundamental research could improve the way optimization of fine particle flotation is conducted. Indeed, the standard optimization starting points of any flotation process such as reagent addition (dosage and distribution) and air flow rate could be manipulated to enhance fine particle response in flotation circuits.

Another aspect that was not found in the literature that could have an impact on fine particle flotation is pulp temperature. Temperature has an effect on flotation performance and has been identified at several operations (Levanaho *et al.*, 2005). The heat (energy) may be responsible for collector adsorption strength, but no published work relating seasonal temperature differences specifically to fine particle recovery was found. Fine particles, accountable for a large portion of total mineral surface area in pulp, are prime candidates for possible temperature effects. This hypothesis warrants investigation.

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