Evaluation, modelling and simulation of the MIM flotation circuit to increase revenue

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Abstract

The recovery and concentrate grade, used simultaneously, are the most widely accepted measures of assessing metallurgical performance. Meanwhile, new challenges are being found with more complex ores around the world, a new methodology to evaluate flotation circuits more accurately is used to keep the Mineral Industry afloat in a changing world by increasing revenues.

The flotation process is described by means of approaches of floatability components. The rate constant, k, is determined as a function of the dynamics of flotation cells as well as the properties of minerals. The new methodology is used for evaluation of a copper flotation circuit. The circuit evaluation shows that the particles being recovered in the scavenger bank are increased by 1.7% due to an increase in the collector. Meanwhile, the collectorless flotation of an initially hydrophilic chalcopyrite sample, cyclone underflow (80% recovery at pH 7.7), was decreased to 40% after regrinding at the same range of pH. Simulation of the copper circuit shows that an additional chalcopyrite recovery of 1.0% in the entire circuit is achieved by increasing the scavenger capacity. However, the concentrate grade decreases 2.7%, due to an increase of the recycle stream in cleaners. The medium and slow floating fractions are the main components in this recycle, which are normally associated with composite particles.

Keywords: Flotation; rate constant; floatability components; evaluation; simulation

1 Introduction

Evaluation and optimisation of flotation circuits has been the focus on most metallurgists since the first plant started hundreds of years ago. In the 70’s, computer based techniques were developed to evaluate flotation process performance and to develop mathematics model which describe behaviour of minerals across a flotation cell/bank or circuit. Such models were developed to simulate flotation circuit behaviour (King, 1973; Whiten, 1972).

Kinetics and liberation data were more successful in the evaluation of flotation process (Cameron, 1971; Frew, 1982; Lynch, 1981). These types of data are very useful to describe the behaviour of mineral in the entire flotation process or individual flotation cells. However, the most interesting approach for evaluation of flotation
processes is that in which the analysis can be used to quantify the gangue component reporting to the concentrate due to each of these three mechanisms: poor liberation, entrainment and imparted hydrophobicity or true flotation (Davila, 1994; Johnson, 1987; Runge, 1997).

Within this paper an evaluation, based on floatability classes, of the Mt Isa copper circuit is presented. The floatability component approach is used to investigate how the gangue mineral is reported in the concentrate by analysing the non-floataing fraction in the entire flotation process. At this stage only the overall mineral behaviour is presented, but this methodology can be extended to size, mineral and liberation analysis.

2 Experimental

2.1. Sampling

Flotation surveys and cell characterisations were carried out at the Mt Isa Copper Concentrator (Figure 1) in June and August 2000. Samples of the feed, concentrate and the tailings of each flotation cell were collected for each survey. Samples were cumulated over a period of time of two hours (one cut every 30 minutes). All samples of feed, concentrate and tailings were weighed, filtered and weighed again to calculate percent of solids. These samples were analysed for lead, zinc, iron and silica at the Mt Isa laboratory. Backup samples were stored in case further analysis was required.

![](image_url)

**Figure 1 – Copper flotation circuit at Mt Isa Mines (June 2000)**
2.2. Batch Flotation Tests

Batch flotation tests were conducted in conjunction with each plant survey to determine the floatability parameters more accurately. Each of these tests was conducted under the same standard procedure using a JKMRC laboratory cell. Mill water, dosed with drops of methyl-isobutylcarbinol (MIBC), was used to make up the volume of the laboratory flotation cell to 4.4 litres. The air rate was adjusted to 13 litres/min and the impeller speed to 1000 rpm. Concentrate particles were collected after cumulative flotation times of 0.33, 0.67, 1.0, 2.0, 4.0 and 8.0 minutes. All samples were weighed, filtered and weighed again to calculate the percent of solids. Then, dried samples were assayed by lead, zinc iron and silica at the Mt Isa laboratory to construct the recovery-time curves of each stream.

2.3. Flotation Models

The flotation kinetic model assumes that flotation is a first-order process in which the rate of removal of a flotation component is proportional to its concentration in the flotation unit:

\[ -\frac{dC}{dt} = kC \]  

(1)

where \( C \) is the concentration of particles in the cell at time \( t \); 
\( k \) is the first-order rate constant of the particles in the cell.

True flotation and entrainment are responsible for the transport of particles from the pulp to the concentrate. Recovery by entrainment is proportional to the feed water recovery in the concentrate and the degree of entrainment \((ENT_i)\) is a strong function of particle size (Johnson, 1972).

\[ ENT_i = \frac{mass\ transfer\ of\ entrained\ particles\ to\ the\ concentrate}{mass\ transfer\ of\ water\ to\ the\ concentrate} \]

(2)

Thus, the recovery by entrainment is then calculated as:

\[ R_{ent} = \frac{(1 - R_{tracer}) \cdot ENT_i \cdot R_w}{(1 - R_w)} \]

(3)

where \( R_{tracer} \) is the overall recovery of the gangue mineral assuming fully liberated particles; 
\( R_w \) is the water recovery.

Recovery by true flotation is induced by hydrophobic particle. Thus, the overall recovery \((R)\) of particles in a flotation unit is calculated by:

\[ R = \frac{k\tau}{1 + k\tau} \quad \text{perfect mixing} \]

(4)
\[ R = 1 - \exp(-kt) \text{ plug flow} \] (5)

where \( \tau \) is the mean residence time using a perfectly mixed model;
\( t \) is the mean residence time using a plug flow model.

The kinetic rate constant, \( k \), is described by three parameters (Gorain, 1997):

\[ k = S_b R_f P \] (6)

where \( S_b \) is the bubble surface area flux, sec\(^{-1}\);
\( R_f \) is the froth recovery, a characteristic of the froth phase in a flotation unit;
and
\( P \) is the ore floatability.

The bubble surface area flux is a parameter related to cell hydrodynamics:

\[ S_b = \frac{6 \cdot J_g}{d_{32}} \] (7)

where \( J_g \) is the superficial gas velocity, cm/sec;
\( d_{32} \) is the Sauter mean diameter, cm.

Several techniques have been developed to calculate the froth recovery factor \( (R_f) \). However, the most widely accepted is that developed for batch flotation tests (Feteris, 1987). \( R_f \) is calculated using the Equation 8:

\[ R_f = \frac{k}{k_c} \] (8)

where \( k_c \) is the first-order rate constant in the collection zone.

A flotation process can be characterised by a distribution of the ore floatability in a flotation stream. The characterisation can be conducted by (1) mineral classes; (2) size by mineral classes; and (3) size by mineral by liberation classes.

Figure 2 shows a schematic representation of the ore floatability in a flotation unit. Bubbles form aggregates with the hydrophobic particles as seen in Figure 2. Aggregates bubble-particles with very high hydrophobicity (black circles) will have a fast flotation response compared with those particles with low hydrophobicity or hydrophilic particles (white circles). Thus, a flotation response in a flotation unit can be represented by a distribution of floatability, i.e. fast, medium, slow and non-floating.
Figure 2 – Schematic representation of the floatability components in a flotation unit

When \( R_f \) or \( S_b \) is not measured during the plant survey, a new constant is introduced in the flotation model:

\[
C_{scale-up} = k_{cell}^{batch} = \frac{k_{cell}}{k_{batch}} = \frac{S_b^{cell}}{S_b^{batch}} \cdot \frac{R_f^{cell}}{R_f^{batch}} \cdot P^{cell}
\]

In Equation 9, it is assumed that the ore floatability in the cell, \( P^{cell} \), and in the batch flotation test, \( P^{batch} \), is the same. Thus, the cell scale-up number is a contribution of:

\[
C_{scale-up} = \frac{S_b^{cell}}{S_b^{batch}} \cdot \frac{R_f^{cell}}{R_f^{batch}}
\]

### 2.4. Mass Balancing

Mass balancing is the most useful method of data assessment. JKSimMet mass balancing sub-system is used to assess data efficiently. The balancing process produces best-fit estimates of flowrates and a set of adjusted lead, zinc, iron, silica and percent solids which are consistent with those flowrates. Overall, JKSimMet mass balancing technique minimises the sum of weighed squared errors (Jktech, 1989), that is:

\[
SSQ = \sum_{j=1}^{N} \sum_{i=1}^{L} \left( \frac{X_{ij} - x_{ij}}{\sigma_{ij}} \right)^2 + \sum_{i=1}^{L} \left( \frac{A_i - a_i}{\sigma_i} \right)^2
\]

where \( N \) is the number of measurements,

\( L \) is the number of streams,
\( X \) is a measurement, \\
\( x \) is the adjusted measurement, \\
\( A \) is the measured flow, \\
\( a \) is the adjusted flow, and \\
\( \sigma_f \) and \( \sigma_t \) are the weights or standard deviations for the measurements and flows respectively.

Based on experimental results and personal judgement, the standard deviations are assigned to the data as follows:

- For assays greater than 10\%, a standard deviation of 1.0\% is assumed.
- For assays less than 1\%, a standard deviation of 0.1\% is assumed.
- For assays between 1\% and 10\%, a standard deviation of 0.1\% plus one tenth of the fraction is assumed.

### 3 Results and Discussion

#### 3.1. Mass Balancing

To mass balance the flotation circuit data, the plant feed is assumed to be accurate (i.e. contain zero error). It should be noted that the mass balancing program tells how the experimental data (including both the stream data and sizing data) meet the mass conservation criterion. It does not tell which data are accurate or inaccurate. Neither can it give the true values. The accuracy of the experimental data depends entirely on the experimental design and handling.

Mass balancing results are summarised in Figure 3 and Table 1. The adjusted data are very similar to the experimental values.

![Figure 3 – Comparison of the adjusted assays and the experimental assays in the Mt Isa copper circuit](image)
Table 1 – Comparison of the overall mineral recoveries using assays and balanced data of the Mt Isa copper circuit.

<table>
<thead>
<tr>
<th>Mineral Recovery (assay data)</th>
<th>CuFeS₂</th>
<th>FeS</th>
<th>NSG</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mineral Recovery (balanced data)</td>
<td>94.89</td>
<td>17.27</td>
<td>3.36</td>
</tr>
<tr>
<td>Mean</td>
<td>95.22</td>
<td>14.83</td>
<td>3.56</td>
</tr>
<tr>
<td>$\sigma_{n-1}$</td>
<td>0.47</td>
<td>3.44</td>
<td>0.28</td>
</tr>
</tbody>
</table>

3.2. Reproducibility of Batch Flotation Tests

Laboratory batch flotation tests are useful in the determination of the ore floatability parameters, such as the ore floatability ($P$) and the fraction ($m$) of that particular floatability in the feed stream. Replicate laboratory batch flotation experiments were conducted in the retreatment tailings during the first survey with the regrinding in operation. Figure 4 shows the recovery-time and grade-time curves of two laboratory tests. As shown in these curves, a good level of reproducibility is observed.

![Figure 4 – Reproducibility of batch flotation tests performed in the retreatment tailings](image-url)

3.3. Nodal Analysis

The nodal analysis technique evaluates the ore floatability behaviour around a single separator or mixer in a flotation process using the mineral recovery-time curves (Runge, 1997). This technique is a back calculation of procedure in which the flotation responses of separator products are combined and compared with that of the...
feed (Figure 5a). In the case of a mixer (Figure 5b), the flotation responses of the various feed streams are combined and compared with that of the product stream.

![Separator and Mixer Diagram](image)

Figure 5 – Typical separator and mixer in a flotation circuit. (a) Combined stream separated into two streams; (b) a number of feed streams produce one combine stream.

### 3.3.1. Evaluating Reagent Addition

Reagent addition has significant effects on the ore floatability (Klimpel, 1995). Samples were taken from the feed, concentrate and tailings of the scavenger bank. It is important to note that the feed of the scavenger bank is the rougher tailings which was taken before the collector *sodium butyl xanthate* (BX) and the frother (MIBC) were added. Figure 6 shows the sampling points in this particular separator.

![Sampling Points Diagram](image)

Figure 6 – Locations of samples for batch flotation tests around the scavenger bank. The symbols are associated with the recovery-time curves in Figure 7.

Figure 7 shows the chalcopyrite recovery-time curves around the scavenger bank (Figure 6). The concentrate stream shows a very fast flotation response in which 84% is recovered in the first 2 minutes of flotation. Meanwhile, the chalcopyrite response in the tailings is slow, may be due to the coarse particles or poor liberation.
Figure 7 – Chalcopyrite recovery-time curves around the scavenger bank. Combine product is calculated using the scavenger concentrate and tailings

The most important aspect of Figure 7 is that the reconstructed recovery-time curve (combine product) is greater than that of the rougher tailings (scavenger feed) prior to reagent addition. By adding more collector (BX) in the scavenger feed, the ultimate recovery of chalcopyrite increases 20% which means an increase of 1.7% of chalcopyrite recovery in the scavenger feed.

When no collector is added across a separator or mixer in a flotation process, the ore floatability must be conserved as in the case of the cleaner bank (Figure 8). The combine feed is calculated based on the rougher concentrate, column tailings, retreatment concentrate and recleaner tailings and the combine product is calculated based on the cleaner concentrate and tailings.

Figure 8 – Locations of samples for batch flotation tests around the cleaner bank

Figure 9 shows the chalcopyrite recovery-time curves of the combine feed and combine product (Figure 8). As shown in Figure 9, both recovery-time curves are
very similar. It is an indication that the ore floatability is conserved when no collector 
or changes on the physical properties of minerals occurs.

Figure 9 – Chalcopyrite recovery-time curves around the cleaner bank. Combine 
product is calculated using the cleaner concentrate and tailings

3.3.2. Evaluating Regrinding

Regrinding composite particles is expected to decrease the rate of flotation due to a 
decrease in the particle size and the need for additional collector on the new surface 
available. In this section, comparing the recovery-time curves before and after 
regrinding assesses the regrinding on chalcopyrite. Figure 10 shows the sampling 
points around the regrinding circuit.

The combine feed involves the scavenger concentrate and cleaner tailings. Thus, a 
comparison between the recovery-time curves of both the combine feed and the
cyclone overflow is investigated (Figure 11). It is important to note that only cyanide (CN) is added in the cyclone overflow.

Figure 11 – Chalcopyrite recovery-time curves around regrinding. Combine feed is calculated using the scavenger concentrate and cleaner tailings

Figure 11 shows that the cyclone underflow response is slightly slower compared with the combine feed in the first four minutes of flotation. This behaviour is due to changes on surface chemistry due to regrinding as observed in Figure 12. This figure shows that the collectorless flotation of an initially hydrophilic chalcopyrite sample, cyclone underflow (80% recovery at pH 7.7), was decreased to 40% after regrinding at the same range of pH (see Table 2). The chalcopyrite flotation response immediately after regrinding is severely retarded. However, more investigation is required to determine which of these two mechanisms are taking place on the chalcopyrite surface:

- Adsorption or precipitation of iron oxyhydroxide species as an overlayer on the iron-deficient chalcopyrite;
- Ferric iron oxidative leaching of the chalcopyrite surface.

<table>
<thead>
<tr>
<th>Stream</th>
<th>PH Start</th>
<th>PH End</th>
<th>Eh (SHE), mV Start</th>
<th>Eh (SHE), mV End</th>
<th>Temp (ºC) Start</th>
<th>Temp (ºC) End</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cyclone Underflow</td>
<td>7.55</td>
<td>7.85</td>
<td>17</td>
<td>51</td>
<td>25.0</td>
<td>26.0</td>
</tr>
<tr>
<td>Ball Mill Discharge</td>
<td>7.50</td>
<td>7.70</td>
<td>-132</td>
<td>37</td>
<td>25.0</td>
<td>25.0</td>
</tr>
</tbody>
</table>
It should be noted that some oxidation products such as ferric or ferrous ionic species originating from pyrite and/or pyrrhotite particles may exist in the pulp and produce an acidic oxidation environment. Researchers (Subrahmanyan, 1993; Trahar, 1994) have published that the following equation has a dominant role in determining the pulp potential level in flotation:

\[ Fe^{3+} + e^- \leftrightarrow Fe^{2+} \]

Consequently, the pulp potential becomes lower according to Equation (13). This could be the reason why regrinding leads to more negative pulp potential as reported in Table 2.

\[ E = 0.771 - 0.05916 \log \left( \frac{Fe^{2+}}{Fe^{3+}} \right) \]

(13)

However, the retardation effect showed in Figure 12 is restored once the ball mill discharge is mixed with the scavenger concentrate and cleaner tailings which passing through the cyclone unit (Figure 11). It is assumed that the centrifugal action of the cyclone is another important factor restoring the chalcopyrite hydrophobic flotation. Therefore, additional pre-conditioning time should increase the recovery of chalcopyrite after regrinding.

Figure 12 – Chalcopyrite recovery-time curves before and after regrinding
3.4. Simulation

3.4.1 Flotation Circuit Parameter Estimation

Balanced data and recovery-time relationship from laboratory flotation tests were used to determine the ore floatability parameters \((P_i's \text{ and } m_i's)\) and cell scale-up numbers \((C_{\text{scale-up}})\) which describes the mineral behaviour in the copper circuit (Figure 13). The parameters were derived using an Excel spreadsheet with the aid of the SOLVER algorithm. Four floatability components (fast, medium, slow and non-floating) were calculated for each of the chalcopyrite, pyrite and non-sulphide gangue minerals.

Floatability parameters in Figure 13 were determined based on the following constrains:

- The entrainment mechanism is not taken into account in the development of this model.
- Cell scale-up numbers (i.e. bubble surface area flux \((S_b)\) and froth recovery \((R_f)\)) do not change when the circuit configuration is altered (simulation).
- The ore floatability is determined in the rougher and scavenger feed due to addition of collector.
- The ore floatability does not change significantly after regrinding composite particles.

Table 3 and 4 show the derived floatability parameters for chalcopyrite, pyrite and non-sulphide gangue under the conditions of the plant surveys.
Table 3 – Derived floatability parameters ($P$’s and $m$’s) in the Mt Isa copper circuit

<table>
<thead>
<tr>
<th>Stream</th>
<th>Mass Fraction in Feed Stream</th>
<th>Ore Floatability</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>$m_{fast}$</td>
<td>$m_{medium}$</td>
</tr>
<tr>
<td>Rougher Feed</td>
<td>CuFeS$_2$</td>
<td>0.7236</td>
</tr>
<tr>
<td></td>
<td>FeS</td>
<td>0.0000</td>
</tr>
<tr>
<td></td>
<td>NSG</td>
<td>0.0000</td>
</tr>
<tr>
<td>Scavenger</td>
<td>CuFeS$_2$</td>
<td>0.0200</td>
</tr>
<tr>
<td>Feed NSG</td>
<td>FeS</td>
<td>0.0000</td>
</tr>
<tr>
<td></td>
<td>NSG</td>
<td>0.0000</td>
</tr>
</tbody>
</table>

Table 4 – Cell scale-up numbers in the Mt Isa copper circuit

<table>
<thead>
<tr>
<th>Flotation Bank</th>
<th>Mean Residence Time (min)</th>
<th>Cell Scale-up Numbers</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rougher</td>
<td>12.07</td>
<td>CuFeS$_2$</td>
</tr>
<tr>
<td>Scavenger</td>
<td>9.17</td>
<td>FeS</td>
</tr>
<tr>
<td>Retreatment</td>
<td>11.75</td>
<td>NSG</td>
</tr>
<tr>
<td>Column</td>
<td>80.49</td>
<td>CuFeS$_2$</td>
</tr>
<tr>
<td>Cleaner</td>
<td>13.70</td>
<td>FeS</td>
</tr>
<tr>
<td>Recleaner</td>
<td>35.18</td>
<td>NSG</td>
</tr>
</tbody>
</table>

As may be observed in Table 4, the cell scale-up number in the column is slower compared with the other flotation units. It may be due to the low turbulence in the column which produces a very low $S_b$ value.

Model prediction of the Mt Isa copper circuit is shown in Figure 14. The derived parameters represent the mineral behaviour fairly well. The scattering points in Figure 14 are mainly from the batch flotation tests, specially, across the column cells, where the separation process is completely different as in the conventional cells.
3.4.2. Simulation

The floatability parameters \((P's\) and \(m's\)) and the cell scale-up numbers were entered into the JKSimFloat simulator. The following simulations were conducted to investigate the effect of increasing circuit capacity:

- Simulation 1 – decreases in circuit tonnage
- Simulation 2 – increasing retreatment capacity
- Simulation 3 – bank residence time assessment

The results of these simulations are shown in Table 5.

**Table 5 – Simulation case study results**

<table>
<thead>
<tr>
<th>Case Study</th>
<th>Final CuFeS(_2) Grade</th>
<th>Final CuFeS(_2) Recovery</th>
</tr>
</thead>
<tbody>
<tr>
<td>Base Case</td>
<td>25.78</td>
<td>96.25</td>
</tr>
<tr>
<td>Simulation 1</td>
<td>20.58</td>
<td>98.11</td>
</tr>
<tr>
<td>Simulation 2</td>
<td>24.97</td>
<td>96.59</td>
</tr>
<tr>
<td>Simulation 3</td>
<td>24.86</td>
<td>97.17</td>
</tr>
</tbody>
</table>

The results show that it should be possible to increase the overall chalcopyrite recovery by 1.8% by decreasing the circuit tonnage by half. It means that increasing the circuit capacity in some stages of the entire circuit would increase the overall recovery in the copper circuit. The overall chalcopyrite recovery would be slightly increased by increasing the retreatment capacity. However, by increasing the scavenger and retreatment capacity the overall chalcopyrite recovery is increased almost 1% unit.
Table 6 shows the effect of the recycle in the cleaner tailings by increasing the retreatment capacity. The recycle is defined as the ratio (in percentage) of the cleaner tailings divided by the rougher feed. An incremental solid flowrate is rejected in the cleaner tailings from 32.3% (base case) to 51.9% with an addition of eight mechanical flotation cells.

Table 6 – Recycle stream by increasing retreatment capacity

<table>
<thead>
<tr>
<th></th>
<th>Base Case</th>
<th>Simulation 2</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fast Fraction</td>
<td>0.03</td>
<td>0.25</td>
</tr>
<tr>
<td>Medium Fraction</td>
<td>4.73</td>
<td>13.07</td>
</tr>
<tr>
<td>Slow Fraction</td>
<td>25.83</td>
<td>39.57</td>
</tr>
</tbody>
</table>

As observed in Table 6, the slow fraction has an important contribution on the recycle stream. However, it should be noted that no model validation has been conducted and also that the flotation model assumes no entrainment. It means that the grade prediction is overestimated and any conclusion should be taken with caution.

4 Conclusions

Nodal analysis shows that the ultimate recovery of chalcopyrite increases 20%, which means an increase of 1.7% of chalcopyrite recovery in the scavenger feed, by adding more collector (BX) in the scavenger feed. The chalcopyrite flotation response immediately after regrinding is severely retarded due to the regrinding leads to more negative pulp potential. The collectorless flotation of an initially hydrophilic chalcopyrite sample, cyclone underflow (80% recovery at pH 7.7), was decreased to 40% after regrinding at the same range of pH. Derived floatability parameters show good prediction in the copper circuit. Simulation results show that decreasing the circuit tonnage by a factor of 50% an additional chalcopyrite recovery of 1.8% is achievable. Increasing the reteatment capacity by doubling the number of cells gives 0.3% additional chalcopyrite recovery, but the grade is severely decreased. An additional 1.0% chalcopyrite recovery is achieved by increasing the scavenger and retreament capacity. However, the final chalcopyrite grade is decreased because an increase in the medium and slow flotating fractions in the cleaner tailings which may increase the recovery of non-valuable minerals.

5 Acknowledgements

The author would like to express his appreciation to the following parties for their help and support to make this research work possible:

- AMIRA P9M Sponsors
- David Way and Virginia Lawson from Mt Isa Mines for their support during the testwork
- Kym Runge and Xiao-Feng Zheng for invaluable assistance in performing the testwork and data analysis
Consejo Nacional de Ciencia y Tecnologia (CONACyT) of Mexico for Juan L. Reyes-Bahena’s scholarship Grant No. 66764/137256.

References
JKTech (Ed.) 1989. JKSImMer user manual and suplementary information manual (Version 5, release 1). Brisbane, Australia.