Establishing the Relationship between Grind Size and Flotation Recovery using Modelling Techniques

Kym Runge, Jaclyn McMaster, Mariska Ijsselstijn and Andre Vien

Submitted for Publication:
Flotation '07 Cape Town, South Africa, 6 – 21 November 2007

Contact:
Kym Runge
Manager – Flotation Process Technology
Metso Minerals Process Technology – Asia Pacific
Metso Minerals
Unit 1 8-10 Chapman Place
Eagle Farm Queensland Australia 4009
Phone 07 3623 2965
Mobile 0418 950 043
Fax 07 3623 2998
kym.runge@metso.com
Establishing the Relationship between Grind Size and Flotation Recovery using Modelling Techniques

Kym Runge  
Manager – Flotation Process Technology  
Metso Minerals Process Technology Group – Asia Pacific  
Metso Minerals  
Unit 1 8-10 Chapman Place  
Eagle Farm Queensland Australia 4009  
kym.runge@metso.com

Jaclyn McMaster  
Graduate Process Technology Engineer  
Metso Minerals Process Technology Group – Asia Pacific  
Metso Minerals  
Unit 1 8-10 Chapman Place  
Eagle Farm Queensland Australia 4009  
jaclyn.mcmaster@metso.com

Mariska Ijsselstijn  
Process Engineer  
Rio Tinto’s Northparkes Mine  
Bogan Road  
Parkes NSW 2870  
Mariska.Ijsselstijn@riotinto.com

Andre Vien  
Senior Research Engineer  
Metso Minerals Process Technology  
2281 Hunter Road  
Kelowna, B.C., Canada  
andre.vien@metso.com
Establishing the Relationship Between Grind Size and Flotation Recovery using Modelling Techniques


* Metso Minerals Process Technology – Asia Pacific, Brisbane, Australia
# Rio Tinto’s Northparkes Mine, Bogan Road, Parkes, Australia
** Metso Minerals Process Technology, Kelowna, Canada

ABSTRACT

Flotation recovery is strongly affected by the liberation of the mineral particles in the flotation feed. Thus, to optimise flotation grade and recovery it is usually best to grind to a fine size. This requirement needs to be balanced by the cost of grinding finer and the revenue that results from an increase in grinding throughput, which often results in a coarsening of the flotation feed.

A relationship can be developed between grind size and flotation recovery using modelling techniques. There are different models that can be used to perform this task, differing in their complexity and the amount of data required for model development. Obviously, the simpler techniques have the advantage that they can be performed routinely with a minimum of information and thus cost. However there are doubts about the accuracy of the prediction.

In this paper, three different techniques that establish a relationship between grind size and flotation recovery are reviewed. The models are developed using data collected from an industrial flotation circuit. The predicted grind size versus flotation recovery relationship produced from the different techniques has then been compared.

Keywords
Froth flotation, particle size, modelling, simulation
INTRODUCTION

Flotation is a process which is strongly affected by the particle size produced during grinding. Particles of different size and degree of liberation float at different rates. Flotation recovery is often optimal for particles of an intermediate size (Figure 1) with coarser particles exhibiting slow flotation kinetics because of their size and poor liberation and fine particles exhibiting slow flotation kinetics because of poor flotation collision efficiency.

![Figure 1 – Classical recovery versus size relationship](image)

But there is a cost related to grinding to a fine size. Power usage increases as the grind size decreases and the tonnage able to be processed by a grinding circuit can be limited by a grind size target.

The overall objective should be to optimise profitability. To calculate conditions (in this case grind size) which optimise profitability, models are required to relate grinding circuit and flotation circuit performance. Grinding circuit modelling is relatively mature and there are models which can be used with confidence to predict grind size as a function of the grinding circuit configuration and operating conditions. There are a number of flotation circuit methodologies emerging to predict flotation performance as a function of grind size. The objective of this paper is to present and review three of these techniques and test them using an experimental data set collected from an industrial grinding/flotation circuit.
REVIEW OF FLOTATION MODELLING TECHNIQUES

Three different modelling techniques that predict flotation recovery as a function of grind size will be reviewed in this paper – the Bazin technique, a bank based sized flotation model and a floatability component by size model. These techniques differ in terms of their complexity and the amount of data required for their development. The objective of testing these models is to determine whether the simpler models produce realistic results and under what circumstances the more complex models should be used.

Bazin Technique

Bazin et al (1994) presented a technique which uses simple relationships, calibrated using data routinely measured by an operation, to predict flotation recovery as a function of grind size. This technique will be referred to as the Bazin technique throughout this paper.

The technique consists of three steps – (1) prediction of a grind size distribution, (2) prediction of the metal distribution within this size distribution and (3) use of this metal distribution in combination with the circuit size recovery relationship to predict overall flotation recovery.

Grind size distribution is an output of the grinding model being applied in a particular situation. The various options available to predict this size distribution are discussed in more detail in a later section of this paper.

Bazin et al (1994) presented a novel technique for predicting the distribution of metal within a particular grind size distribution. They observed a single relationship between the cumulative percent passing size distribution and cumulative percent passing metal distribution in both laboratory and industrial scale data (Figure 2). This assumption was also observed by Edwards and Vien (1999) in another laboratory and plant experimental study.
By fitting a cubic or polynomial function to this relationship, the cumulative metal distribution can be predicted for any grind size distribution. Fundamentally, this relationship accounts for both preferential breakage and classification segregation effects.

The third step of the procedure involves calculating the overall recovery using the size and metal distribution functions. This is performed using the size recovery relationship which can be obtained by mass balancing the products of a circuit, an example of which is depicted in Figure 1. The size and metal distribution functions enable the proportion of a metal in each size class in the feed to be determined. Overall metal recovery \( R_{\text{overall}} \) is the sum of the metal recovery of each size fraction \( R_i \) weighted by the proportion of metal in that size fraction in the feed \( m_i \) (Eq(1)).

\[
R_{\text{overall}} = \sum_i m_i R_i
\]  

This overall recovery can be plotted as a function of grind size to determine the point of maximum flotation recovery and the rate of loss in flotation recovery as grind size is increased.

This technique has the advantage that it can be performed using information routinely measured by a processing plant. For metallurgical accounting purposes, feed, concentrate and tailing samples are often collected, sized and assayed. The only additional information...
required to determine the relationships to use this technique are the assays on each size fraction.

This technique relies on two assumptions – that the cumulative metal versus cumulative size and the particle size versus recovery relationships are independent of grind size. More work is required to determine the extent to which these assumptions are true in a flotation circuit.

As the relationships that form the basis of this technique must be derived from operating plant data, the technique cannot be used for prediction for ores yet to be processed through the plant. There are also no rules as to how the recovery versus size relationship will change with a change in circuit operation or circuit feed rate, parameters one might want to optimise in a flotation grinding optimisation study. This limits the extent to which the model can be used in a study of this kind.

**Rates for each size class in each Flotation Bank**

It is not unusual for surveys of a flotation circuit to be performed every one or two years and that the samples from these surveys are sized and the size fractions assayed. This data is suitable for the development of a bank based flotation circuit model. Apling and Osborne (1986) and Kawatra et al (1982) have developed size distributed models of this type to simulate industrial circuit data.

Flotation is often considered a first order process, with the recovery of a particle in a flotation cell being a function of its flotation rate constant ($k$) and the cell residence time ($\tau$). Particles also report to concentrate due to the entrainment mechanism and Johnson (2005) states that this recovery mechanism is a function of the concentrate water recovery ($R_w$) and the degree of entrainment (Ent). Flotation recovery, incorporating both mechanisms, can be calculated using an Equation developed by Savassi (1998) (Eq(2)).

$$R = \frac{k \tau \left(1 - R_w\right) + \text{Ent} R_w}{(1 + k \tau \left(1 - R_w\right) + \text{Ent} R_w)}$$

(2)

If the recovery is measured in a flotation cell, Eq(2) can be re-arranged to calculate the flotation rate constant achieved in the bank (Eq(3)). To enable this calculation to be
performed, cell residence time can be estimated using the cell volume \( V_{\text{cell}} \) and tailings flowrate \( Q_{\text{tailings}} \) (Eq(4)) and water recovery is an outcome of the circuit survey mass balance. Degree of entrainment can be estimated using typical sizing entrainment parameters (Johnson, (2005)).

\[
k = \frac{\text{ENT} \ R_w \ (1 - R) - R \ (1 - R_w)}{R \ (1 - R_w) - \tau \ (1 - R_w)}
\]

\[
\tau = \frac{V_{\text{cell}}}{Q_{\text{tailings}}}
\]

Using the information derived from a sized mass balance of the circuit, the overall rate constant of each mineral in each size can be calculated for each cell or bank of the process. These parameters form the basis of the flotation bank model. During flotation simulation, the flow of the sized mineral components is tracked around the circuit. At each flotation bank, Equation 2 is used to calculate the recovery to concentrate of each sized mineral component using the rates measured during the calibration survey. Water flow is usually assumed to be proportional to solids flow. Cell residence time and water recovery required in the flotation bank recovery equation are outcomes of the simulation. The simulation must therefore be performed iteratively, with each unit calculation performed in sequence repeatedly until the flow in all streams converges to a constant value. Grinding circuit models can be used to change the size distribution feeding this flotation circuit simulation.

This model structure has the advantage that it can be developed from circuit survey data alone. It is essentially assuming, however, that all mineral particles of a particular size always float in a bank with the same flotation rate, regardless of its other properties (i.e. coarse liberated particles reporting to a bank will float the same way as coarse composite mineral particles). Researchers have obtained a deviation from simple first order kinetics within an individual mineral size class and to overcome these problems have described the floatability of each size by floatability components (e.g. Sutherland, 1977) – the technique described in the following section.
Floatability Component by Size Models

This modeling approach involves dividing each mineral by size class in the feed into discrete floatability components (Figure 3). The classical two component model (Kelsall, 1961) is an example of this method.

Figure 3 – Physical representation of the mineral by size by floatability component model (after Harris, 1998)

The recovery achieved in a flotation cell using this model is a function of the proportion of each component in the stream ($m_i$) and the rate constant of each component ($k_i$). This rate constant being a function of both the component’s ore properties and the cell operating conditions (e.g. air rate, froth depth, froth transportation distances).

Harris (1998) developed a non-linear parameter estimation technique for determining the parameters of this model using circuit survey information performed in conjunction with batch laboratory flotation tests. The laboratory flotation tests are all performed using the same operating conditions and are used to determine the ore’s floatability characteristics (i.e. flotation rate constant of each floatability component and the proportion of these components in every stream of the circuit). The circuit survey data being used to determine parameters associated with the effect of cell operation on flotation rate. These techniques have been used to develop models of many industrial flotation circuits (Runge et al, 1996; Alexander and
Wigley, 2003; Schwarz and Kilgariff, 2005) and are beginning to be used to develop models including the effect of size (Coleman et al, 2007).

Recovery in a flotation cell when using this model is a function of the proportion of each component in the feed ($m_i$), the laboratory batch test flotation rate constants ($k_i^0$), degree of entrainment and water recovery (Eq(5)). It is also a function of the cell scale up constant ($C$) which is introduced to account for the different operating conditions in the full scale cells compared to the laboratory test.

$$R = \sum_i m_i \frac{k_i^0 C \tau (1 - R_w) + \text{Ent} R_w}{(1 + k_i^0 C \tau) (1 - R_w) + \text{Ent} R_w}$$  \hspace{1cm} (5)$$

Simulation, using this model, involves tracking the flow of the different sized floatability components in the different streams of the flotation circuit. At each flotation unit, the recovery of each floatability component to concentrate is calculated using Equation 5. It is assumed that the cell scale up number for a particular bank will remain constant during the simulation. Simulation is performed in a similar way to the bank based model – iteratively as the cell residence time and water recovery values are outcomes of the simulation which at first are unknown.

The mineral and size distribution feeding this flotation simulation model is established by the grinding model. Each mineral size class is broken up into different floatability components, the rate and proportion of which was established during the calibration survey. This approach intrinsically assumes that a mineral in a particular size will float with the same floatability characteristics regardless of the degree of grinding that has occurred. If the ore floatability properties of a size class is a strong function of its degree of mineral liberation, then what this assumption implies is that when a coarse sized particle breaks into finer sized particles, these finer particles have mineralogical properties similar to other particles already existing in that size. Unless an ore exhibits significant preferential breakage and/or classification segregation due to density effects, this assumption should hold true for a flotation circuit feed stream which is the product of a grinding circuit.

Flotation recovery in this model is a function of the ore and the operating conditions used in the different cells of the process. It would therefore be expected to be more accurate in its
predictions than that of the previous models presented. Model validation performed at pilot scale indicates that the modelling methodology can successfully predict the change in flow of mineral components with a change in circuit configuration (Coleman et al, 2007).

Unfortunately the parameters of this model cannot be derived from circuit survey data alone. As multiple stream batch testing has not traditionally been performed during a circuit survey, the technique cannot be used to model historical circuit survey data. To determine these parameters as a function of size, the circuit and batch laboratory information also needs to have been sized and the size fractions assayed. It can be an expensive and time consuming process to develop this type of model.

EXPERIMENTAL DATA COLLECTION

Data collected during a circuit survey performed of Northparkes Module 1 copper flotation circuit was suitable for derivation of all three flotation modelling approaches outlined in the previous section.

The Northparkes Module 1 circuit at the time of the survey consisted of both roughing and cleaning stages of flotation (Figure 4). Roughing was performed by a tank cell and rougher and scavenger banks of Dorr Oliver 600 conventional square cells. Cleaning was predominantly performed by two Jameson cells. Conventional Dorr Oliver 300 flotation cells were used for cleaner scavenging. The operating objective of this circuit was to maximise the recovery of copper and produce a concentrate of acceptable copper grade for downstream processing. Copper in the feed was present in a number of copper bearing minerals (bornite, chalcopyrite and tetrahedrite) and the gangue was predominantly silicates. Pyrite was the predominant sulphide gangue. At the time of the testwork, the solids in the feed stream to the tank cell had a P80 of 150 microns.
The circuit survey involved collection of sub-samples of each of the streams in the circuit (Figure 4) at 30 minute intervals over a two hour period. The circuit operated very steadily during this sampling campaign.

Four batch laboratory flotation tests were performed using samples of different streams of the process. These samples were collected at the same time as the circuit survey. Streams floated were the cleaner and recleaner concentrate streams, the combined rougher concentrate and scavenger tailing. The tests were performed in a bottom driven Agitair style 5 litre flotation cell using an air rate of 18 litres/min and an impeller speed of 1000 rpm. Concentrate was removed at a constant rate using a fixed depth paddle, designed to ensure that the entire froth phase would be removed during each scrap. No reagent was added to the tests other than frother which was dosed into the plant water used to maintain cell level. Five or six concentrates and a tailing sample were produced from each test.

All samples collected during the test program were weighed wet, filtered, dried and weighed again. Sub-samples were analysed by x-ray fluorescence spectroscopy to determine their copper, gold, iron, sulphur and silica assays. Samples were also sized and size fractions assayed for copper.
This data was used to perform a mass balance to estimate unknown stream flows and produce consistent size by assay information for the circuit. Recovery of copper and gangue was also calculated as a function of size in each laboratory batch flotation test.

**CALIBRATION OF THE FLOTATION MODELS**

The data from the testwork was used to determine the parameters of each of the three flotation models. All models were derived for both copper and “not copper”. Not copper is a term that refers to everything that is not copper in the ore stream (other minerals and other elements within the copper mineral matrices). The not copper assay is equal to 100 minus the copper assay. By modelling both copper and not copper, the copper recovery and grade can be established by the flotation models for a particular simulation.

**Bazin Technique Calibration**

The Bazin technique only requires the recovery versus size to be established for the circuit. Figure 5 shows the copper and not copper recovery versus size relationship calculated using the mass balanced size by assay survey data.

![Figure 5 – Recovery of copper and what is not copper as a function of particle size in the Northparkes Module 1 flotation circuit](image)

Recovery at a different grind size is the weighted sum of these recoveries based on the metal and size distribution produced from the grinding simulations.
Bank Based model Calibration

The bank based flotation model requires the flotation rate constant of copper and not copper in each size class to be established for each of the flotation banks in the circuit. Figure 6 shows the flotation rate constant of copper in different size classes derived for the conventional flotation cells and the Jameson flotation cells using the circuit survey data. Estimates were made of cell volume and the degree of entrainment for each size class to enable this calculation to be performed. It should also be noted that the Jameson cell recovery during modelling was assumed to be independent of feed flow, not dependent on cell residence time and thus only a function of the cell flotation rate calibrated from the circuit survey.

Almost all of the cells exhibited a maximum flotation rate constant for the intermediate sizes, with drops in rate observed for the coarse and fine size fractions.

Floatability Component Model Calibration

To calibrate the floatability component by size model the mass balanced survey and batch laboratory data was used to determine the mass and batch test rates of the components required to represent the ore floatability in the circuit. Cell scale-up numbers for each component in each flotation unit of the process were also determined. It was found that both the copper and not copper floatability in each size class were best described by three components – a fast component, a slow component and a non-floating component. Table 1
and 2 show the batch flotation rate constants and the proportion in each of the copper and not copper components in the different size classes in the circuit feed stream.

<table>
<thead>
<tr>
<th>Size</th>
<th>Floatation Rates</th>
<th>Proportion in each Rate Class</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Cu Fast</td>
<td>Cu Slow</td>
</tr>
<tr>
<td>+150</td>
<td>3.8</td>
<td>0.40</td>
</tr>
<tr>
<td>+106</td>
<td>5.0</td>
<td>0.45</td>
</tr>
<tr>
<td>+75</td>
<td>5.5</td>
<td>0.63</td>
</tr>
<tr>
<td>+53</td>
<td>5.6</td>
<td>0.33</td>
</tr>
<tr>
<td>+38</td>
<td>5.7</td>
<td>0.14</td>
</tr>
<tr>
<td>+20</td>
<td>5.2</td>
<td>0.56</td>
</tr>
<tr>
<td>+10</td>
<td>4.0</td>
<td>0.03</td>
</tr>
<tr>
<td>-10</td>
<td>1.9</td>
<td>0.05</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Size</th>
<th>Floatation Rates</th>
<th>Proportion in each Rate Class</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Not Cu Fast</td>
<td>Not Cu Slow</td>
</tr>
<tr>
<td>+150</td>
<td>2.9</td>
<td>0.36</td>
</tr>
<tr>
<td>+106</td>
<td>3.3</td>
<td>0.25</td>
</tr>
<tr>
<td>+75</td>
<td>3.3</td>
<td>0.25</td>
</tr>
<tr>
<td>+53</td>
<td>3.8</td>
<td>0.13</td>
</tr>
<tr>
<td>+38</td>
<td>4.3</td>
<td>0.09</td>
</tr>
<tr>
<td>+20</td>
<td>4.0</td>
<td>0.04</td>
</tr>
<tr>
<td>+10</td>
<td>3.4</td>
<td>0.03</td>
</tr>
<tr>
<td>-10</td>
<td>1.8</td>
<td>0.04</td>
</tr>
</tbody>
</table>

* The greyed floatability class was fixed during fitting so as to fit the trend of the other results

Overall, the derived parameters followed expected trends. Copper flotation rates were optimum for the intermediate size classes. The proportion of non floating copper increased as particle size increased. Liberation is known to deteriorate and for copper to be locked in composites in the coarser size fractions. Thus it would be expected that the proportion of non-floating copper would increase with size.

The not copper results will largely be associated with the non-floating gangue. As such the proportion of floating not copper would be expected to decrease as particle size decreases as observed due to improved liberation. The other trends (i.e. decrease in non-floating in fine sizes and the increase in the slow floating class) are not as expected and are thought to be due to compensation for problems with the entrainment estimates. Entrainment will not be the same in all cells as assumed during model development and this fact will often result in the
derivation of very slow gangue (rather than non-floating gangue) which is actually being recovered by entrainment. This is more prevalent in the non-copper model because a larger proportion of recovery is due to entrainment. Simulation results are not overly affected by this phenomenon.

COMPARISON OF THE FLOTATION MODEL PREDICTIONS

Each of these models was linked to a grinding model to enable the circuit recovery and grade to be estimated as a function of grind size. The aim is to compare and evaluate the differences between the predictions obtained from the different flotation modelling methodologies.

A grinding model is required to produce the input to the flotation models – the size and metal distribution at the different grind sizes. It is best that these grind size distributions be predicted using population based grinding models, the methodologies of which are described by Napier-Munn et al (2005). These models can be calibrated to produce the expected grind size and metal distributions achievable using the existing grinding circuit.

The grinding predictions performed in this paper, however, were performed using a technique more suited to a preliminary evaluation. This technique was used by Edwards and Vien (1999) in their application of the Bazin methodology. The feed size distribution is fitted to a Rosin-Rammler model (with the size expressed as percent passing a given size) (Eq(6)).

\[
y_i = 100 \left(1 - e^{-\left(\ln\left(\frac{P}{100}\right)\frac{d_i}{R}\right)^\alpha}\right)
\]

The percentage in each size fraction \(y_i\) is a function of the size \(d_i\), percentage passing \(P\) of a chosen reference size \(R\) and a parameter associated with the shape of the size distribution \(\alpha\). This model assumes that the shape of the product size distribution is independent of fineness of grind (i.e. \(\alpha\) constant). Inputting different values of \(P\), results in different grind size distributions. This model was fitted to the flotation feed data from the Northparkes circuit (Figure 7). The fit was adequate and the prediction of the shape of the grind size distributions of the feed to the various preceding grinding stages was also adequate.
Using the Rossin-Rammler models, different size distributions to feed the three flotation models were produced (Figure 8).

The metal distribution in this size distribution was estimated using a cubic polynomial derived by comparing the cumulative metal versus cumulative percent passing mass relationship in the Northparkes flotation feed and tertiary grinding feed streams (Figure 9).
Figure 9 – Metal versus mass cumulative distributions derived using a cubic polynomial fitted to the Northparkes tank cell and tertiary milling feed streams

For each of the grind size distributions shown on Figure 8, the final concentrate grade and recovery was simulated using the three different flotation models. Figures 10 and 11 show the final concentrate copper grade and recovery estimated as a function of grind size derived from this analysis.

Figure 10 – Copper recovery as a function of feed P80 estimated using the three flotation models
The prediction of circuit copper recovery with a change in grind size is identical for the three models tested. These models indicate that a P80 of 65 micron results in optimal circuit recovery. Grinding finer than 65 micron results in a drop in recovery as this is the point where the proportion of the size distribution residing in the poorly recovered -10 micron fraction outweighs any benefits associated with grinding the poorly floating coarser particles.

It is predicted that copper grade will increase as the grind size becomes finer. This is a consequence of gangue recovery decreasing as particle size decreases. The improved liberation in the finer sizes results in a drop in recovery and counteracts any increased recovery due to entrainment. The Bazin approach and the floatability component approach result in very similar grade predictions. The sized bank model grade predictions are lower than the other two models at fine grind sizes.

The models indicate that there is benefit in grinding finer (in terms of copper recovery and grade) but these benefits would need to be balanced by the costs associated with the higher power requirements and the lower throughput that might result from a smaller grind size. Grind size should not, however, be decreased beyond 60 micron as recovery at sizes finer than this value is predicted to decrease.

All models produce similar predictions. Essentially this means that the more complex models result in the same recovery versus size relationship at the different grind sizes. Changes in circuit performance are not significant enough at the different grind sizes to change this relationship and thus change the circuit grade and recovery predictions. Thus to establish the
link between grind size and recovery one need only use the Bazin Technique. It has produced the same trade off between grind size and circuit recovery. It is easy to calibrate and can be performed routinely as different ores are processed through a plant, enabling the establishment of grinding targets for different ores.

The Bazin Technique, however, cannot be used to simulate and predict circuit performance with a change in circuit operation or throughput, parameters one might also want to include in a float grind optimisation. To perform these types of simulations, one needs to employ a more complex flotation model, examples of which are the bank based model and floatability component models presented in this paper. As a demonstration of the kinds of relationships that can be established, Figure 12 shows the results of simulations performed using the floatability component model for different feed throughputs at different grind sizes.

![Figure 12 – Final Concentrate grade and recovery simulated for different feed grind sizes and different feed tonnage rates](image)

To compare the simulation predictions produced from the bank based and floatability component models, both were simulated under a range of different circuit operating conditions. 100 simulations were performed using both models in which the banks of the circuit were either pulled more quickly or slowly by adjusting the bank kinetic rates, or in the case of the floatability component model, the cell scale up parameters. The magnitude of the adjustment was a randomly created deviation normally distributed with a standard deviation of 15% (calculated using an algorithm outlined in Numerical Recipes (Press, 1992)). The final concentrate copper grade and recovery produced from these various simulations have been plotted on Figure 13.
The floatability component model predicts that a change in circuit operation will move the circuit along a grade versus recovery relationship. Concentrate recovery cannot exceed 93% as this is the percentage of copper that is floatable in the circuit at the baseline grinding condition.

The magnitude of change in the bank based model predictions far exceeds that of the floatability component model. Recovery is not limited in the bank based model and for circuits with sufficient residence time, recovery will approach 100%. Ore floatability is not tracked within the bank based model and this results in greater variation in circuit performance. Slow floating particles are recovered more quickly than they should when diverted to banks with fast kinetics. Fast floating particles float more slower than they should when diverted to banks with slow kinetics.

Because floatability component models differentiate between ore and cell effects and track ore floatability during simulation, it is assumed to produce more accurate flotation simulation results. There are large differences between the simulation predictions performed using the bank based and floatability component models. It is therefore recommended that whenever possible, the data required for the development of floatability component by size models be collected when planning a grinding/flotation circuit optimisation simulation study.
If the data available for a circuit optimisation study only enables derivation of a bank based model, then it should be kept in mind that the magnitude of any change in circuit performance is larger than one would expect in reality.

**CONCLUSIONS**

Flotation modelling methodologies have been developed which can be linked with grinding circuit models to simulate and optimise total operation performance. These techniques offer an opportunity for the benefits and costs of grinding to different sizes to be established.

Three different flotation models which relate feed grind size to flotation circuit performance were developed in this paper using data collected from an industrial flotation circuit. These models differed in terms of their complexity of derivation and the amount of data required for model calibration. All models gave similar relationships between flotation circuit grade and recovery and feed grind size. The models predict quite different metallurgical results, however, when feed tonnage rates, circuit configuration or cell operation is changed from that employed in the baseline data set.

The Bazin technique was the simplest of the techniques tested and it can be developed from samples routinely collected for metallurgical accounting of flotation circuit operation. This technique offers operations a means of regularly determining the optimum grinding targets for the different ores processed through its plant.

The floatability component by size modeling approach would seem the only technique amenable to a comprehensive grinding and optimisation study. It not only enables the optimum grind size to be established at a particular plant tonnage rate but enables the effect of throughput and circuit operation to be included in the grind/float optimisation exercise. The bank based model also tested in this paper could also, theoretically, be used for these types of simulations but it has been shown to over predict the magnitude of the change in metallurgical performance of a flotation circuit.

All techniques assume that the mineralogical (and thus flotation properties) of a particular size class does not change as grind size decreases. There is some evidence that it is valid to
assume this in the feed to a flotation circuit. Future work will investigate how to best predict liberation as a function of grind size in a grinding model.

The next step in the evolution of flotation modelling is for the parameters to be explicitly linked to particle liberation. With the increase in availability of mineralogical information for a circuit, it will not be long before the relationships between liberation and flotation are established and able to be incorporated into the flotation modelling algorithm. There will always be a place, however, for simpler techniques that can be developed with the minimum of information for certain types of flotation circuit evaluation.

REFERENCES


