AMINPRO'S METHODOLOGY FOR APPLYING CURRENT FLOTATION MODELING STARTING FROM BATCH KINETICS (PART 2 OF 2)

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ABSTRACT

For many years flotation equipment sizing and circuit design were based entirely on pilot plant trials using a bulk sample of the ore in question. As ore grades have dropped and the scale and capital costs of process operations have increased, it became necessary to consider the economic impact of ore variability during the design stage. Engineers began to supplement the pilot plant data with batch tests performed on point samples collected throughout the ore deposit. Empirical scale-up factors, developed through personal experience, were then applied to the batch test data to predict plant performance. Recent research has led to a better understanding of the mechanisms of flotation at both the laboratory and the plant scale, allowing scale-up procedures based on science and economics to begin to replace those based on personal experience. In Part 1 of this two part series, test interpretation models were explained and a comparison made between the traditional and newer methodologies in the execution and interpretation of kinetic test work. In this paper, the authors describe how the results of a properly executed laboratory test can be used to create a plant-scale model that will reliably predict flotation circuit performance under a wide range of operating conditions, allowing the design engineer to perform what-if scenarios, create grade-recovery curves, and size the flotation circuit such that the recovery, grade, and costs are optimized for the entire ore body.

KEYWORDS: mechanisms of flotation; ore variability; economic impact; equipment sizing; reliable flotation circuit performance prediction; what if scenarios.

1. INTRODUCTION

1.1. Background

In part one of this series a short review was presented of relevant developments in the practice of flotation modeling (Amelunxen & Amelunxen, 2013). A methodology was described for interpreting batch test kinetic data in the context of the flotation compartmental model, yielding collection zone flotation recovery and rate constants that are independent of froth zone effects. These collection zone kinetic parameters are then applied to the industrial process to calculate the collection zone recovery in the full scale machine. This collection zone recovery is subsequently adjusted for froth zone effects and hydraulic entrainment, yielding a corrected recovery estimate for the full scale perfect mixer. The methodology is illustrated conceptually in Figure 1.

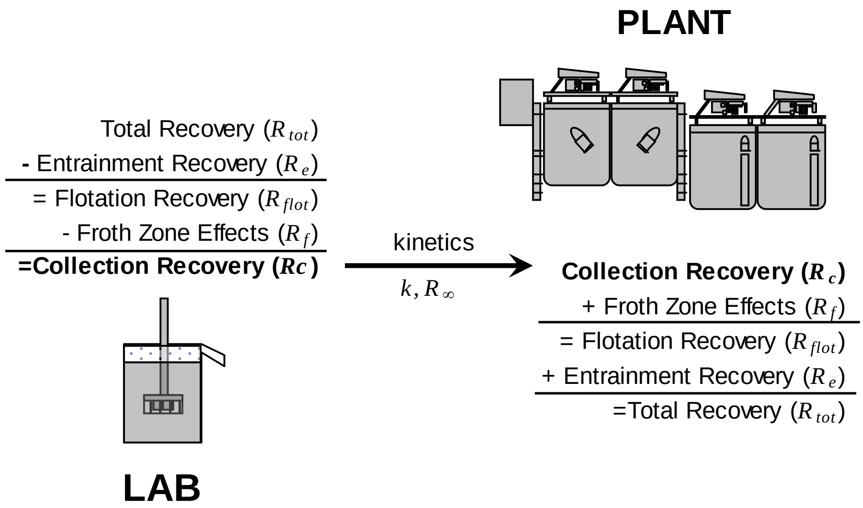

Figure 1. Aminpro scale-up methodology.

In theory the method illustrated by figure 1 applies only to a single perfect mixer. For a full scale plant model, which consists of many perfect mixers that can be arranged in any number of configurations, many more features are required of a functional plant model. These are summarized below.

1.2. Features of a Plant Model

Any plant flotation model must contain the following basic features.

1.2.1. The ability to track kinetics

The R_∞ and k parameters that are measured in the laboratory, whether on an averaged or size-by-size basis, are only valid for the first cell in a rougher bank. The simple action of removing floatable material via the concentrate of the first cell reduces the kinetics of the feed to the second cell. The same must be true in reverse: when combining streams with different kinetics, the resultant kinetics must reflect that of the blend of streams.

1.2.2. The ability to predict concentrate grades

The model must accurately predict stage recovery and concentrate grades for all minerals without using adjustment factors. If the mineral kinetic parameters, froth recoveries, and entrainment models are correctly configured, no correction factors should be necessary.

1.2.3. The ability to consider mechanical constraints

Flotation machines have mechanical limits as well as metallurgical limits. The flotation model must consider such things as lip loading constraints, carrying rate limits, and froth crowding effects.

1.2.4. The ability to mimic human logic (if used for geometallurgy)

When using a geometallurgical approach to plant design, it is common to change the process configuration assumptions based on the ore type being processed. For example, it is often necessary to change the regrind circuit specific energy based on the mineralogy and liberation characteristics of the ore type. This capability should be conserved in the model. Because modern flotation circuit design is an economic optimization exercise across a mine sequence plan, the process model should be flexible enough to incorporate human logic in the model configuration.

2. THE AMINPRO PLANT MODEL

The Aminpro plant model uses the same compartmental approach described in part one of this series. It is comprised of a collection zone model, a froth zone model, and an entrainment model.

2.1. Perfect Mixer Model

2.1.1. Collection Zone Model

The collection zone model is a first order rate equation after Arbiter & Harris (1962):

$$R_c = \frac{k\tau}{1 + k\tau} \qquad (1)$$

Where R_c is the collection zone recovery as a function of the mean residence time τ and k is the first order rate constant. R_∞ and k are measured in the lab and the values for each mineral, including gangue, are input as average values (in the case of the SKT test) or as a function of grind (in the case of the FKT test).

2.1.2. Froth Recovery

The effect of froth zone recovery is incorporated using the compartmental model described in the first part of this series:

$$R_{flot} = \frac{R_c R_f}{R_c R_f + 1 - R_c} R_\infty \qquad (2)$$

Where R_∞ is the maximum recovery at infinite time, R_f is the froth recovery and R_{flot} is the flotation recovery of the mineral specie after accounting for froth recovery. The value of froth recovery is generally a function of the froth depth and distance to the lip; i.e. it is a measure of how hard the operator is “pulling” the cell. It is the primary control parameter for the model.

2.1.3. Entrainment

Entrainment is defined using the Savassi (2005) form in which the solids recovery by entrainment is proportional to the water recovery. The degree of entrainment is:

$$D.E. = \frac{R_s}{R_w} \tag{3}$$

Difficulties have been reported in fitting a single model equation to the curve of D.E. versus particle size for all operating conditions (Zheng *et al.*, 2006). The authors have found that most of the error can be mitigated by using a modified form of the Swebrec function:

$$DE_x = \frac{f(x)}{1+f(x)} \tag{4}$$

Where

$$f(x) = m_x^{\left(\frac{\ln(0.25)}{\ln(m_{80})}\right)}, \quad m_x = \frac{\ln\left(\frac{X_0}{x}\right)}{\ln\left(\frac{X_0}{X_{50}}\right)} \text{ and } m_{80} = \frac{\ln\left(\frac{X_0}{X_{20}}\right)}{\ln\left(\frac{X_0}{X_{50}}\right)} \qquad \text{(4a, 4b, and 4c)}$$

and X_0, X_{20}, and X_{50} are the particle sizes at which the degree of entrainment is 0.0, 0.2, and 0.5, respectively. Setting the X_0 to a coarse size, say 1700 microns, and fitting the X_{20} and X_{50} parameters to the data published by Zheng *et al.* yields an empirical equation for X_{20} as a function of froth height and superficial gas velocity (*Jg* can be estimated from data available in Wood (2002)) . The X_{50} can then be determined from the X_{20} by fitting a power function with a linear decay term. The regression scatter plots are shown in Figure 2.

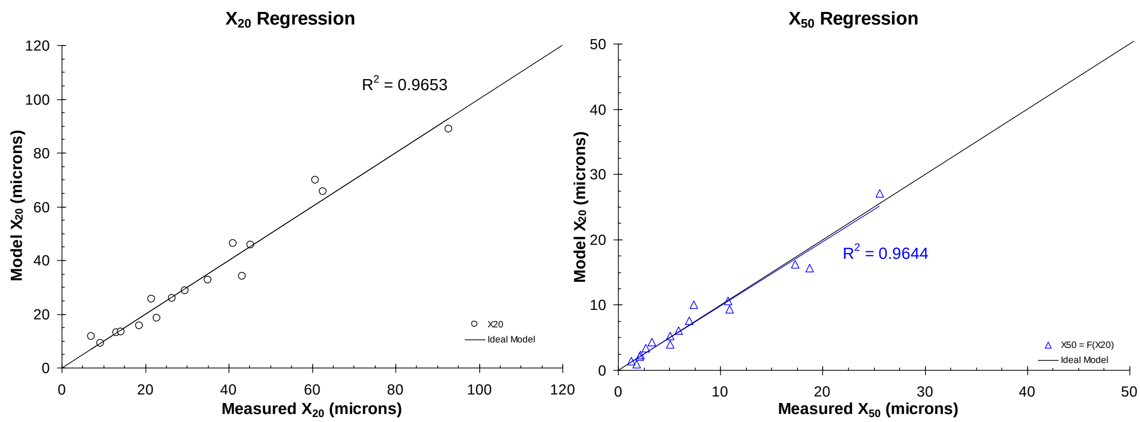

Figure 2. X_{20} (left) and X_{50} (right) regression scatterplots for modeling degree of entrainment versus particle size.

Figure 3 shows the modeled entrainment curves compared to the raw data predicted by Zheng. While it is true that the entrainment rates may be system-specific and/or functions of other variables such as froth viscosity, froth apparent viscosity, etc., the above model provides a simple approximation for modeling the effect on entrainment rate of changing froth depths and air rates.

The Aminpro FKT model tracks each particle size class separately, and entrainment is applied to both floatable and non-floatable minerals remaining in the pulp phase after collection. In this manner, the fractions of each size class recovered by collection and entrainment are accounted for separately. This is a prerequisite for calculating the kinetics of the concentrate and tailings streams.

For the simplified SKT model is it necessary to assume a P_{80} and P_{50} to estimate an average D.E. for each reactor. Thereafter the model calculations are identical to those for the FKT test, but the stream is treated as if it were a single size class.

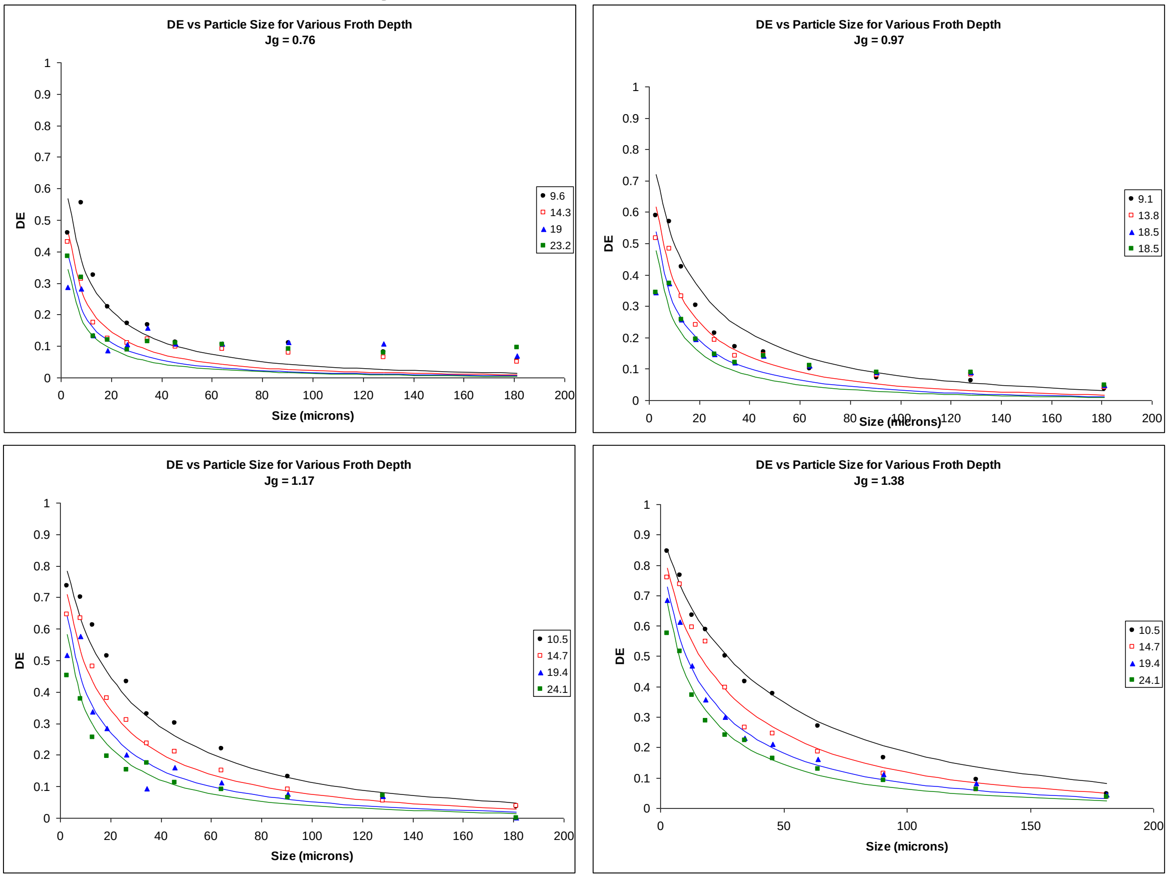

Figure 3. Measured vs Predicted DE vs Particle Size for different Jg and froth depths.

2.2. Kinetics of the Product Streams

The mineral particles recovered to the concentrate of a flotation machine are always comprised of three fractions:

1. *Hydrophobic* or floatable material recovered by true flotation;
2. *Hydrophobic* material recovered by mechanical entrainment with water;
3. *Hydrophilic* or non-floatable material recovered by mechanical entrainment.

Because by definition the R_∞ for a hydrophobic mineral is 100% and that of hydrophilic or non-floatable material is 0%, the R_∞ of the concentrate stream is simply 100 times the fraction of floatable to the total weight of material recovered to the concentrate. The rate constants of the size classes are unchanged.

2.3. Kinetics of combined streams

When two streams with significantly different kinetic parameters are combined, it is necessary to calculate the kinetics of the resulting blend (without having to collect a sample and perform a laboratory test). This often occurs in copper cleaning circuits when rougher concentrates are combined with higher pH or reground cleaner concentrates, or in molybdenum separation circuits where streams with significantly different pH and grind characteristics are circulating in the cleaner section.

The Aminpro model computes the R_∞ of the blend by simple weighted average and the rate constant by logarithmic average. Because for reaction equations the additive parameter is the recovery axis rather than the time axis, rate constants must be transformed into a parameter that is linearly proportional to recovery. It follows from the from the first order rate equation for plug flow reactors that:

$$R_c \propto 1 - e^{-kt} \tag{5}$$

and therefore the weighted average rate constant (k_{ave}) of a blend of n streams (or size classes of streams, in the case of the FKT model) with rate constants k_i can be calculated as follows:

$$k_{ave} = -\ln\left(\frac{\sum_{i=1}^{n} m_i e^{-k_i}}{\sum_{i=1}^{n} m_i}\right) \tag{6}$$

Where m_i is the mineral's recoverable mass flow in stream i. The method is illustrated graphically in Figure 4. The graph on the left shows the calculated flotation recovery of a 50% blend of a fast floating mineral (R_∞ = 95%; k = 2.5) and a slow floating mineral (R_∞ = 75%; k = 0.5). The graph on the right shows the fit of the first order rate equation using a logarithmic average rate constant (k = 1.16, bold line) and simple average rate constant (k = 1.50, simple line). While neither method yields a perfect fit, particularly near the origin of the time scale, the logarithmic method yields a lower rate constant and somewhat better fit (a least-squares fit of k yields 1.15 for the blend, supporting the log-average method). The fact that neither method yields a perfect fit merits further study, as the issue of additivity is relevant not just for process modeling but for correct application of the geostatistical methods used to populate the block model with the kinetic parameters.

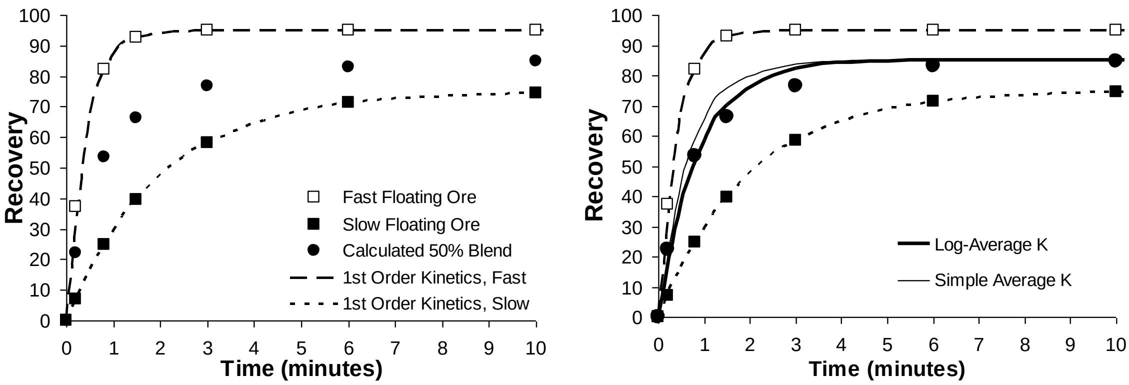

Figure 4. Calculated blend of 50% fast floating ore, 50% slow floating ore (left) and fit calculated first order kinetics (right).

2.4. Iterative Methods

Once the models for each bank of flotation cells have been configured as described above, the mass flow rates and kinetics of the product streams are calculated. Because most flotation circuits contain numerous flotation stages with the products from later stages returning as feed to prior stages, the flotation circuit simulator must iterate until the kinetics and mineral flows in each size class converge on the steady state values. For simplicity, flexibility, and transparency the platform used is Microsoft Excel and the iterative algorithms are coded in Visual Basic for Applications

(VBA). The authors feel that the primary drawback of Excel relative to other platforms (namely, computational speed) is no longer a significant factor given the complexity of the models and the processing speed of publicly available CPUs.

3. MECHANICAL CONSTRAINTS

The steady state flow rates must conform to established process mechanical constraints such as carrying capacity and lip loading constraints.

3.1. Carrying capacity

Column and mechanical cell carrying capacity limitations have been studied extensively since the publication of the first model by Espinoza-Gomez *et al.* (1988). Most recently Patwardhan and Honaker (2000) have proposed a model based on studies on coal flotation that significantly revises the carrying constraints upwards. Industrial cell carrying rates (*not* carrying capacities) as high as 4.0 tonnes/hr-m2 and have been observed in large diameter base metal column cells (Figure 5 shows some of the authors' industrial plant data relative to the Patwardhan model).

The issue of carrying capacity should not be understated: in the authors' experience some recent high tonnage concentrators have been designed for maximum carrying capacities that are less than half of what the recent models and industrial data suggest they should be, thereby shifting the design objective from a residence time optimization exercise to a simple cross-sectional area allowance (and resulting in significantly higher, and unnecessary, capital expenditure for the project owners).

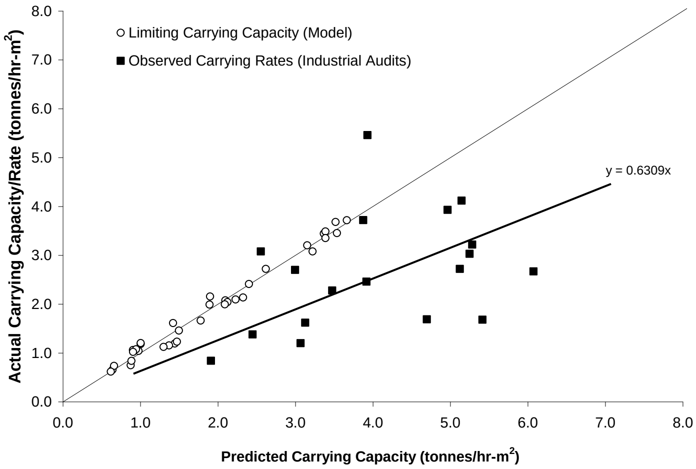

Figure 5. Flotation cell carrying capacity model (open circles, after Patwardhan and Honaker, 2000) and observed carrying rates from industrial column cells (black squares, from Aminpro database).

3.2. Lip Loading

Lip loading is defined in terms of tonnes/hr of pulp recovered to the concentrate per linear meter of overflow lip. Values as high as 20 tonnes/hr-m have been observed for large scale commercial columns (Amelunxen, 1991).

4. PROCESS PLANT BENCHMARKING

Model validation is performed by collecting audit data from industrial plants and measuring the flotation kinetics of the feed in the laboratory using the methodology discussed in Part 1. The kinetic information is then used to model the flotation circuit and the results are compared to the balanced audit data. The model is considered valid if:

1. The plant assays match those of the model, for all minerals including gangue;
2. The plant recoveries for all minerals mach those predicted by the model;
3. In the case of the FKT model, the concentrate size distributions in the plant should also be accurately forecasted by the model.

The two "tuning parameters" used to calibrate each stage in the model are froth recovery and water recovery. Froth recovery is tuned such that the recovery of the primary economic mineral matches that achieved by the plant. If the measured kinetic parameters are correct, all other mineral recoveries should also match. Next, the water recovery is tuned such that the percent solids of the concentrate streams match those of the plant. If the measured kinetics, froth recovery, and entrainment models are correct, then the mass flow of not just the important minerals, but *all minerals in the concentrate* should also match those measured in the plant.

Figure 6 and Figure 7 show the predicted versus actual metal recoveries and concentrate assays for a series of pilot plant runs performed on a South American copper/silver/gold ore. The kinetic parameters were measured on the feed to the pilot plant and the model was calibrated using the methodology described above.

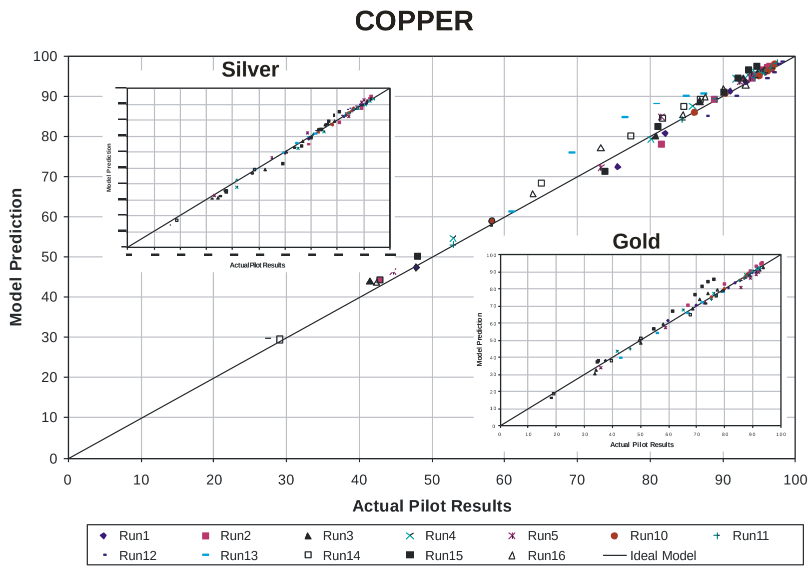

Figure 6. Predicted vs actual recoveries for copper, silver, and gold pilot plant.

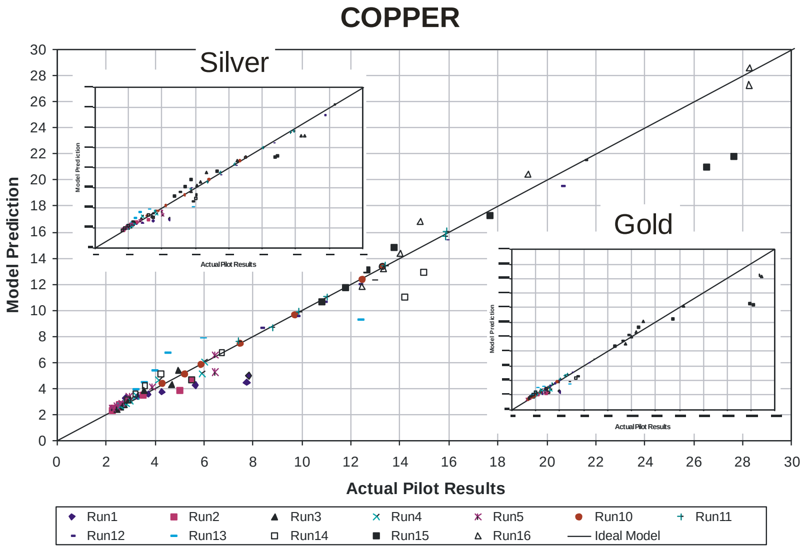

Figure 7. Predicted vs actual concentrate grades for copper, silver, and gold pilot plant.

Plant audits are useful for more than just validating flotation models and enriching design databases. They offer valuable insight into the behavior and sensitivity of a flotation circuit to alternate operating strategies. The Aminpro model has been used for auditing and optimization at more than fifteen industrial and pilot flotation circuits, including:

1. Six large scale copper or copper-moly porphyry plants (Africa, North and South America);
2. Four primary molybdenum concentrators (China);
3. Three copper/molybdenum separation circuits (South America);
4. A copper/gold/silver pilot plant circuit (South America);
5. A lead/zinc operation (South America);
6. A coal flotation circuit (Australia).

5. PROCESS PLANT DESIGN

The process plant design methodology is a three-step procedure as outlined below.

5.1. Select a starting point

The starting point configuration is generally chosen based on a database or experience, and the circuit is then modeled using the kinetics measured in the lab. The modeling procedure is identical to that described above, with the difference that there is no plant audit with which to calibrate the flotation circuit model. This is not a problem, however, because the model does not employ any adjustment factors that would require corroboration by a database—the only "tuning parameters" that need to be validated are concentrate pulp densities and froth recoveries. Indeed, empirical and/or phenomenological models for water and froth recoveries are emerging (for example, see Zheng *et al*., 2006 for a review of the former; Vera *et al.*, 2002 for the latter).

5.2. Determine the circuit performance and economic return

The circuit performance is measured on a cost-revenue basis. Revenues are calculated on a year-by-year basis using the tonnage, ore type or types, and head grades of the existing mine plan and the relevant metal prices and smelter contracts (including penalties for impurities). Capital costs are estimated using Aminpro's existing database of equipment quotations and agitator energy requirements. Because the tonnage and head grade for each year change, the circuit must be configured to generate the highest revenue per tonne for the given year. This involves an iterative procedure in which froth and water recoveries are varied for the various flotation stages until the revenue reaches a maximum. The procedure is the mathematical analog of the plant metallurgist's job of estimating the grade recovery curve (although this was often done with historical data rather than a kinetics-based processs model) and using the sales contract and metal prices to determine the optimum operating point. This is the "human logic" component of the model and it consists of iterative or inferential procedures coded in VBA.

5.3. Find the optimum circuit configuration

Once the preliminary circuit has been optimized, the ultimate design is determined by incrementing the rougher and cleaner flotation capacity (where appropriate) and calculating the new optimum economic performance as in Step 2. This procedure is repeated until the NPV is maximized; i.e. that either adding flotation capacity, or removing it, would result in reduced project value. Therefore, the design is optimal.

6. CONCLUSIONS

In Part 1 of this two-part series, a methodology was presented for determining the collection zone flotation kinetics parameters from a laboratory batch flotation test. This paper described Aminpro's methodology for integrating the collection zone kinetics parameters with a mathematical model of an industrial scale flotation circuit. The key components of any industrial scale model were listed and a method was provided for incorporating them into a mathematical model of an industrial scale flotation circuit. The issue of additivity of flotation kinetics parameters was discussed, and a methodology was shown that allows for tracking kinetics parameters when combining and splitting streams. An alternative and improved mathematical model was provided to describe the relationship between degree of entrainment and particle size, and mechanical constraints such as carrying capacity were also reviewed. The model effectiveness was demonstrated using pilot plant data from a copper/gold/silver project. Finally, with a configured model and updated capital and operating cost information, it was shown how iterative procedures could be used to derive an economically optimum flowsheet configuration.

7. REFERENCES

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