

FLOTATION PLANT DESIGN AND PRODUCTION PLANNING THROUGH GEOMETALLURGICAL MODELLING

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ABSTRACT

The key to successful flotation plant design, production planning and mine/mill optimisation is a solid understanding of the resource to be processed.

As advocated by the authors of this paper, the main components of geometallurgical modelling of an orebody and its associated flotation plant are:

- an ore sampling program and subsequent laboratory testing of these samples in order to extract process model parameters,
- geostatistical distribution throughout the resource model of the process parameters,
- calibration of the plant via benchmarking (for existing operations), and
- plant simulation using a system of process models and the distributed metallurgical parameters (from step b) as the data set.

It is important that the grinding and flotation models are linked. For example, a laboratory test conducted on a drill core sample, intended to represent a portion of the orebody, is conducted at a specific grind (represented by a P_{80}). However, when that ore is actually processed through the plant it may well be (and in many cases most likely will be) at another P_{80} . Additionally, the flotation plant residence time will often be determined solely by the 2 grinding circuit capacity and feed slurry density. In the case of SAG mill – ball mill circuits, the fluctuations in tonnage and grind are known to be high. The modelling approach described in this paper allows for changes in the measured flotation kinetic parameters in order to reflect the expected grind as determined by the comminution process.

A valuable aspect of the geometallurgical modelling approach is the ability to identify and quantify sources of error to obtain the precision of a design or

production forecast. The main sources of error are in the flotation test itself, the geostatistical error (related to sample density), and the model calibration error. The latter is determined through plant benchmarking.

INTRODUCTION

Many mineral extraction operations are challenged on a daily basis with feed ore variability that impacts the throughput of the grinding section and the subsequent concentrate grade and recovery that can be produced. A key to maximising the value of a mining and processing facility is to develop a prior understanding of the characteristics of the feed ore over the lifecycle of the operation from design and financial evaluation of the project, through to monthly and yearly production forecasts and on-going circuit troubleshooting and optimisation. This is achieved through the application of geometallurgical modelling of an ore body.

The main components of geometallurgical modelling are:

- an ore sampling program and subsequent laboratory testing of these samples in order to extract process model parameters,
- geostatistical distribution throughout the resource model of the process parameters,
- calibration of the plant via benchmarking (for existing operations), and
- plant simulation using a system of proven process models and the distributed metallurgical parameters (from step b) as the geometallurgical data set.

This paper will focus on the application of geometallurgical modelling concepts to flotation circuit design and production forecasting.

The first section of this paper describes the method of quantifying pulp

chemistry parameters. A key to practical geometallurgical modelling is having a proven and affordable test that provides the required data from small amounts of available exploration drill core. This 3 makes it possible to conduct enough tests to attain metallurgical validity for insertion in the geological model. The first section of the paper covers the various modes for carrying out the MinnovEX Flotation Test (MFT) and the information attained from these tests.

Furthermore, the authors recognise that a flotation circuit does not operate in isolation, but rather is strongly influenced both by the feed ore delivered to the mill and by the manner in which the preceding grinding circuit processed that ore. In other words, a link is required through geometallurgically enabled comminution and flotation simulation tools. Considering just the flotation characteristics of the feed ore presents a partial picture; also of significant importance is knowledge of how the rock behaved in the grinding circuit from a throughput and grind perspective. Section 2 of the paper demonstrates this concept through a step-by-step example that tracks various blocks of ore from pit through grinding to flotation feed.

The third of the paper section briefly discusses precision analysis. A benefit of a statistical approach to geometallurgical modelling is that the precision of a design or production plan can be

quantified through error analyses. With modelling tools, laboratory tests and geostatistics, there are quantifiable sources of error at each step. This section of the paper provides an overview of error sources in each step of the geometallurgical modelling process [Lozano, Bennett, 2003].

BENCH-SCALE FLOTATION TESTS

Geometallurgical modelling can be defined as an approach that measures metallurgical variability within an ore body and quantifies the effect of this variability on the comminution and metallurgical response of the ore, as it pertains to full-scale production. The geometallurgical approach surpasses the resource model as it encompasses not only the variability of alteration, lithology and metal grades, but also metallurgical process specific characteristics that influence the grinding and mineral recovery characteristics of the ore [Bennett, Lozano, 2004].

Since the success of the approach hinges on how well the *variability* of the ore is represented, it is obvious that, for geometallurgical modelling purposes, many data points from small scale tests are more valuable than a small number of detailed investigations. Therefore, a standard low-cost test that can be performed on a small mass of drill core material is essential. In order to maintain the integrity of the geometallurgical model, it is also important that the parameters from each test are *primary*¹ characteristics that can be distributed across an ore block model using standard geostatistical methods. Describing the floatability of an ore through a set of pulp kinetic parameters for each of the mineral component of the ore means that both the concentrate grade and recovery can be calculated simultaneously using fundamental flotation modelling methods.

The MinnovEX Flotation Test (MFT) has been developed to satisfy both of the above requirements. It is a standard bench-scale test that is used to measure the primary floatability characteristics of a sample of ore. The philosophy behind MFT work for samples is to have a test

program that is simple, quick and can be done at the lowest possible cost – therefore allowing as many tests as possible to be performed on individual drill core samples to characterise the flotation properties of the ore.

The objective of the MFT is to measure the pulp kinetics for each of the mineral species in the ore at a set (pre-determined) reagent suite. The MFT is designed to determine the kinetics of mineral separation in the pulp phase exclusively, while froth effects in the plant are later accounted for through FLEET (Flotation Economic Evaluation Tool) modelling. Further details on the MFT can be found elsewhere [Dobby, Kosick, Amelunxen, 2002; Dobby, Savassi, 2005].

In order to perform multiple tests in a quick and cost-effective manner, the MFT's are carried out in two forms: the full MFT and the mapping MFT. Once sufficient information about the kinetics of the mineral components in the ore body is known (through completion of sufficient full MFT's), then the method can be simplified for mapping purposes. The simplification resides mainly in reduced degree of screen analysis, which thereby reduces the number of samples submitted for chemical analyses. This can significantly lower the overall test cost, with little compromise in the quality of the results. Hence, more drill core samples can be tested at the same project cost - ultimately leading to overall better accuracy in both design and production forecasting.

At the commencement of a drill core test program, a decision would be made on the proportion of full MFT's and mapping MFT's. This decision is based on the prerequisite that sufficient full tests are performed per mineralogy/ore type, in order to calibrate the key interpolation relationships needed for the mapping MFT parameter extraction. When the mineralogy/ore type is simple, the split is typically around 15 % full MFT's to 85 % mapping MFT's. In more complex ore-bodies, with more variable lithology, alteration and/or mineralogy, the proportion of full tests to mapping tests has to be increased.

The pulp kinetic parameters are determined from the MFT results by applying a comprehensive parameter extraction methodology that decouples true flotation and entrainment, and then models the MFT according to fundamental flotation principles [Dobby, Savassi, 2005]. The analysis of each MFT yields the following information, for each mineral (or minor element) of interest:

- The maximum recovery (R_{max}) and cumulative frequency distribution of rate constants for each mineral species at the test grind (described by K_{avg} and α , the latter being a descriptor of the spread of rate constants).
- The quantitative effect of grind on K_{avg} and R_{max} .
- A standard set of R_{max} and K_{avg} values at a single grind-size² common for the full set of drill core samples, together with R_{max} -slope parameters, all of which may be distributed across the mine block model (R_{max} -slope is the change in R_{max} per one micron change in P_{80}).

Table 1 summarises an example set of pulp kinetic parameters that would be determined for a typical copper porphyry material. A typical set of pulp kinetic parameters derived from an MFT performed on a drill core sample ground to 128 μ m is shown on the left side of the table. The kinetic parameters from each MFT need to be reconciled to a standard grind before the parameters are distributed across a mine resource model. Therefore, the terms R_{max} -slope and K_{avg} -slope are used to correct the set of test kinetic parameters to a set of standard kinetic parameters, as shown on the right side of the table.

¹ Primary data is defined as ore-specific parameters, which in the case of flotation describe the inherent floatability of the ore regardless of the plant flowsheet, equipment or operating targets and conditions.

² Described by P_{80} and the Rosin-Rammler slope, or 'm' value

TESTWORK GRIND				95.8 STANDARD GRIND FOR FLEET SIMULATION		
M = 0.93				M = 0.70		
P80 = 128				P80 = 100		
KINETICS AT TESTWORK CONDITIONS				KINETICS AT STANDARD GRIND		
MINERAL	R _{max}	K _{avg}	ALPHA	R _{max} SLOPE	K _{avg} SLOPE	
CuSulph	91.7	1.5	1.3	-0.041	-0.004	
Pyrite	96.0	2.8	3.2	0.002	0.004	
Au	71.5	1.0	1.2	-0.058	-0.002	
Ag	80.0	0.6	4.3	-0.065	-0.001	
NSG	3.9	0.3	10.0	0.023	0.001	

Table 1. Typical set of pulp kinetic parameters derived from an MFT

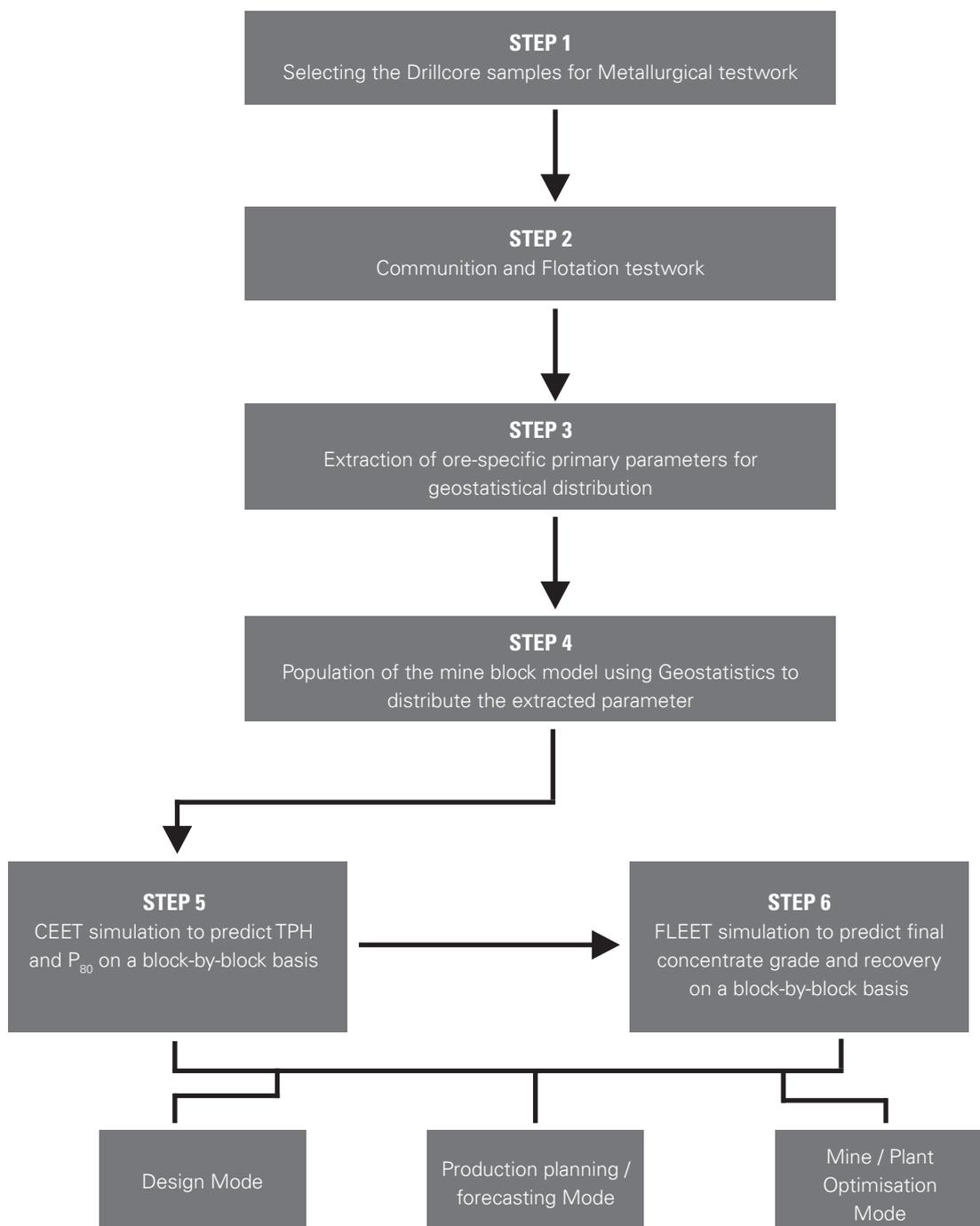


Figure 1. Creating a Geometallurgical Model

RECONCILIATION BETWEEN DRILLCORE TESTWORK AND FLOTATION CIRCUIT SIMULATION

The R_{max} -slope and K_{avg} -slope parameters presented in Table 1 provide the link between drill core flotation testwork results and FLEET flotation circuit simulation on a block-by-block basis for design and/or production forecasting studies.

A geometallurgical model is created by following the steps outlined in Figure 1 [Dobby, Bennett, Bulled, et al, 2004].

The importance of accounting for the effect of grind-size on recovery from the initial grind at which the MFT test is performed to the ultimate prediction of flotation circuit recovery and grade per ore block is illustrated in a step-by-step example.

STEP 1 The drill core samples for metallurgical testwork are selected over the region of interest within the resource. A grid pattern over the area of interest (for example, the first five years of operation) is preferred, ensuring, as a minimum, that all major geological features of the resource are represented. More complex geological conditions will dictate a greater number of samples. Clearly, the joint involvement of geological, mineralogical and metallurgical staff in sample selection is required.

STEP 2 Bench-scale laboratory testwork is carried out on each of the drill core samples. For SAG hardness tests (MinnovEX SPI) typically 2 kg of drill core is required. For ball mill hardness measurements, 1.2 kg for MinnovEX Modified Bond tests or 10 kg for full Bond tests is required [Kosick, Bennett, 1999]. Since full Bond tests are only carried out on 5 to 10 % of the samples, with Modified bond tests performed on the remaining samples, the core requirement is 12 kg for up to 10 % of the samples and only about 4 kg for the other 90 %.

The MFT requires no additional drill core as this test is performed on the products from the SPI and Bond dry grinding tests. At the outset of the testwork program, a standard grind-size for the feed for the MFT is selected. Based on initial hardness information for each drill core sample (SPI and BWi), an estimate is made of the required grinding time to attain the standard grind-size for the MFT, and each sample is wet ground in a laboratory mill for the 8 specified length of time. MFT's are then carried out on all samples, with product streams analysed either according to the full MFT or mapping MFT protocols.

STEP 3 The primary ore parameters, shown in Table 1, which describe fundamental grindability and floatability of the ore samples, are extracted from the lab data according to model fitting techniques [Dobby and Savassi, 2005]. Table 2 provides a Case Study selection of typical parameters measured for three random drill core samples (we will follow the three samples through the next few steps of the process). Note that this table does not show the complete set of parameters, but only a selection for illustrative purposes - the full set of floatability parameters per sample would be as per Table 1.

DRILL CORE LABEL	SPI [MIN]	BWI [KWh/t]	MFT GRIND	$R_{MAX-CuSulf}$ (AT TEST GRIND) [%]	$R_{MAX-CuSulf}$ (AT STD GRIND) # [%]	$R_{MAX-CuSulf}$ SLOPE
DC-1	28	12.4	65	81.0	77.6	-0.090
DC-2	35	11.7	108	94.3	94.6	-0.023
DC-3	91	15.0	128	91.7	92.9	-0.041

Table 2: A selection of typical grindability and floatability parameters for three random drill core samples. Note: A standard target grind of 100 µm had been selected.

Although a standard grind of 100µm had been selected, it can be seen that the actual grindsizes (P_{80}) at which the three tests were performed were 65, 108 and 128µm, respectively. DC-1 is the softest of the three samples and as a result, the laboratory mill over-ground this sample. Likewise, DC-3 is a much harder sample, and the desired grind was not attained in the specified grinding time. Even when attempts are made to account for the Bond value in selecting the lab grind time, these differences often occur. However, a unique aspect of the MFT is that the kinetic parameters measured at each test grind can be reconciled to the standard grind through the R_{max} -slope function.

STEP 4 The grindability parameters and floatability parameters at standard grind are distributed throughout the blocks in the region of interest in the mine resource model using geostatistical techniques (involving the consideration of sample location, ore type and grade). An outcome is illustrated in Figure 2.9 Early in a project development, often there will be insufficient samples tested to develop a geostatistical distribution. In this situation, the distribution of parameters is made along either a regional basis or ore type basis.

Figure 2 is a sample region of a mine resource model, demonstrating areas of higher and lower theoretical recovery of copper sulphide minerals, after geostatistical distribution of the $R_{max-cusulph}$ parameter across the resource model.

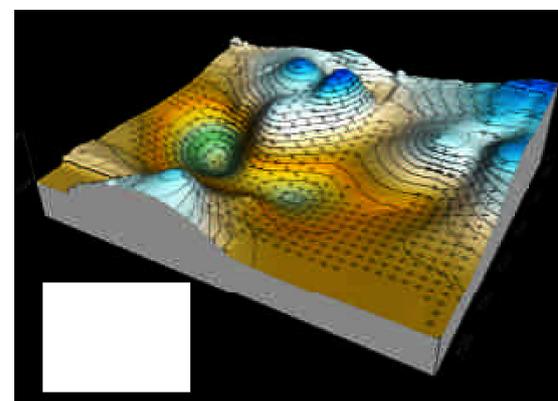


Figure 2: Sample region of a mine resource model

Step 5 At this stage, each block in the geometallurgical model has been attributed the interpolated ore-specific parameters, necessary for grinding simulation. Within a web-based simulation platform, the grinding simulation software ‘mines’ the resource model on a block-by-block basis, calculating the grinding circuit throughput (TPH) and product grind (P_{80}) for each block. Example grinding simulation results for the three blocks of ore (samples) listed in Table 2 are given in Table 3. (In this table it is assumed that DC-1 was located in Block 1, DC-2 in Block 2 and DC-3 in Block 3, and a typical SAG – ball mill circuit was used for the simulation.)

	SPATIAL LOCATION			SPT [MIN]	BWi [kWh/t]	TPH	P80 [MICRONS]	CASE	NOTE
	x_1	y_1	z_1						
BLOCK 1	x_1	y_1	z_1	28	12.4	5185	120	Ball-mill limited	(i)
BLOCK 2	x_2	y_2	z_2	35	11.7	5250	109	Down stream Limited	(ii)
BLOCK 3	x_3	y_3	z_3	91	15.0	4436	89	SAG Limited	(iii)

Table 3: Typical CEET simulation results for three blocks of ore

Note the following:

- i. In this scenario, Block 1 comprises soft ore, therefore affording the opportunity to maximise tons by producing a coarser grind. A limit of 120 μm (based on coarse fraction results of flotation tests) has been set on how coarse the grind is allowed to become, thereby limiting the throughput for Block 1 to 5185 tph.
- ii. For Block 2, neither the SAG mill nor the Ball mill has become the limiting element due to this blocks’ lower BWi. Throughput therefore can be maximised up to a limit typically set by some constraint external to the grinding circuit (typically tailings pumps or concentrate filter capacity dictates this limit on maximum throughput). At this maximum throughput of 5250 tph the grinding circuit can produce a grind of 109 μm .
- iii. Block 3 comprises the hardest ore and the grinding circuit is SAG mill limited. The maximum tonnage that can be treated through the SAG mill is only 4436 tph before the mill would overload. At this lower tonnage, the ball mill circuit produces the finer grind of 89 μm .

STEP 6 The tonnage and P_{80} results from the grinding simulation become input values to the flotation circuit simulation. Similar to Step 5, the FLEET software tool calculates final concentrate grade and recovery on a block-by-block basis based on the tonnage and grind feeding the flotation circuit for the block, and the floatability characteristics attributed to that block. Table 4 provides the set of feed parameters for the same three blocks of ore discussed in Step 5.

	SPATIAL LOCATION			TPH	P_{80} [MICRONS]	$R_{\text{MAX-CuSulf}}$ (AT STD GRIND) [%]	$R_{\text{MAX-CuSulf}}$ SLOPE	ACTUAL $R_{\text{MAX-CuSulf}}$ USED IN FLEET SIMULATION
	x_1	y_1	z_1					
BLOCK 1	x_1	y_1	z_1	5185	120	77.6	-0.090	75.8
BLOCK 2	x_2	y_2	z_2	5250	109	94.6	-0.023	94.3
BLOCK 3	x_3	y_3	z_3	4436	89	92.9	-0.041	93.4

Table 4: Typical FLEET feed values for three blocks of ore

The kinetic parameters at standard grind (100 μm) are distributed through the mine block model. However these values are adjusted on a block-by-block basis according to their R_{MAX} slope values and the P_{80} value calculated for that block. The slope values are also distributed across the block model and are therefore available on a block-by-block basis. The actual R_{MAX} values used in the FLEET simulation are adjusted according to the difference in P_{80} between the standard grind and the block grind and the slope for that block.

It is important to make the correction for the difference in grind-size between that produced in the MFT to that attributed to each block of ore, to achieve accurate design and production forecasting. Table 5 highlights this by showing the original test grind for a drill core sample compared to its associated block grind, and the impact that this has on the prediction of ultimate recovery (R_{MAX}) for that block. Modelling is conducted by utilising the kinetic parameters of all minerals.

	GRIND AT WHICH MFT TEST WAS PERFORMED [MICRONS]	R _{MAX-CuSulf} VALUE EXTRACTED FROM MFT TEST RESULTS [%]	GRIND-SIZE ATTRIBUTED TO THE BLOCK OF ORE AFTER CEET SIMULATION [MICRONS]	ACTUAL R _{MAX-CuSulf} VALUE FOR THAT BLOCK, USED IN FLEET SIMULATIONS [%]
BLOCK 1 (DC-1)	65	81.0	120	75.8
BLOCK 2 (DC-2)	108	94.3	109	94.3
BLOCK 3 (DC-3)	125	91.7	89	93.4

Table 5: The actual R_{max-cusulph} used in FLEET simulations, compared to the values established from the MFT test

PRECISION ESTIMATION FOR RIGOROUS GEOMETALLURGICAL MODELLING

A statistical approach to geometallurgical modelling presents the user with an opportunity estimate precision on plant circuit design or production forecasts. For the model to be used for production planning and/or design purposes, it can only be considered rigorous and definitive if accompanied by an associated precision model.

A complete geometallurgical modelling study comprises the following main components [Dobby et al, 2004]:

- A bench-scale laboratory testwork program,
- Geostatistical distribution over the resource model of the parameters extracted from the bench-scale test,
- Plant benchmarking for model calibration (for existing operations), and
- Circuit simulation using the distributed metallurgical parameters and a system of process models, calibrated according to plant measurements.

Each of these steps presents a source of error that can be accounted for and can be summarized as:

1. Measured (test) precision and kinetics estimation precision
2. Distribution (geostatistical) precision
3. Model calibration precision

The authors have considered the precision associated with each of these steps. Selected examples follow:

- A comparison of the extracted parameters from testing a single

sample of copper ore using 3 trained operators is shown in Table 6. Standard error on the CuSulph R_{max} is less than 1% and on the NSG R_{max} about 0.5%. This could obviously improve with operator experience.

OPERATOR	1	2	3
CuSulf			
Rmax	90.5	91.2	89.5
Kavg	1.6	1.6	1.6
ALPHA	2.4	2.2	1.9
NSG			
Rmax	6.5	6.9	5.7
Kavg	0.4	0.3	0.3
ALPHA	10	10	10

Table 6: Reproducibility of MFT parameters

- In the study of a copper porphyry ore body where full MFTs were conducted on 15% of the samples, with mapping MFTs on the remainder, the estimation of CuSulph kinetics for the mapping tests was found to introduce a standard error of 1.5% for R_{max} and 0.1% for K_{avg}. This could be improved by the development of the estimation techniques used in the mapping MFTs.
- 160 MFTs were conducted on drill core samples to represent the approximately 1.5 million tons of ore to be treated in 1 year in a large copper plant. The kinetics data was distributed across the mining blocks with respect to feed grade and ore type of the blocks. The standard error in estimating the individual block values of the distributed data for CuSulph is shown in Table 7.

ORE TYPE	STD ERROR % IN CuSulf ESTIMATE	
	R max	k avg
1	2.3	0.9
2	1.3	0.9
3	1.7	0.8

Table 7: Standard errors in block parameter estimates

- These figures can be reduced by applying geostatistical methods, such as Kriging, for distribution of data across the mine blocks. Such methods would involve the determination of variograms to establish the range of influence of each sample to neighbouring mine blocks. Any significant reduction in the number of samples or increase in the variability of the ore will naturally result in an increase in the estimated standard errors. Distribution of data across the mine blocks is usually the greatest source of error in forecasting due to insufficient samples and testing.
- The precision of the process model is obtained from plant calibration work. The model error is the difference between the measured values of plant concentrate grade and recovery and the forecast values determined by application of the model to the MFT parameters of the feed. The standard error has been shown to be within 1% on large copper operations. The higher the number of benchmark calibration surveys carried out, the more accurate the process calibration becomes.

The total standard error on the kinetics parameter estimation can be determined by normal statistical methods for error

accumulation and the FLEET model is then run as a Monte Carlo simulation to estimate precision on the projected plant results. Model precision must then be accounted for to give an overall picture of the precision of the technique.

Finally, however precise, the process model must be geometallurgically enabled in order to generate meaningful results, i.e. it must be capable of automatically processing all the blocks in the resource model and therefore able to optimise designs or forecast results based on the complete resource or geometallurgical model. The process model must also be capable of accommodating variations in grind and tonnage for each block in the geometallurgical model. If the process model is not geometallurgically enabled, then it becomes relatively ineffective for industrial design or production forecasting.

PROGRESS OF GEOMETALLURGICAL MODELLING FOR FLOTATION

The first holistic use of geometallurgical modelling began in 1999 when the process model CEET was developed. Prior to CEET there was no mechanism for exploiting the block comminution data in a resource model for optimisation of plant design or production forecasting and planning. CEET was developed using a geometallurgically enabled structure in order to utilize the SPI and Bond data block by block throughout the resource model to design and optimise comminution circuits.

The development of a geometallurgically enabled comminution process model was one of the critical steps required to facilitate the development of an accurate geometallurgically enabled process model for flotation. Knowledge of the throughput and grind parameters per block (which translate to residence time and liberation in the flotation circuit) is very important if the geometallurgically enabled flotation process model is to be accurate enough to use for design or production forecasting (which is the ultimate test of a geometallurgical approach).

The success of CEET led to the development of FLEET and the MFT for the application of geometallurgy to industrial flotation. The final step to link SPI, Bond, CEET, FLEET, the MFT and future process models, was an Internet portal that could host a common geometallurgical dataset for any particular project or resource. This was important from two perspectives. First, inputs from a common geometallurgical dataset for a particular project would be used by all geometallurgically enabled models. Also, the output from any particular run of one process model would form part of the input for the process model for the subsequent downstream process, and the data transfer and integrity had to be handled through this common dataset. Second, by configuring this interface and dataset handling ability through the Internet, on-line support worldwide could be made available continually and inexpensively. This Internet portal and interface was developed in 2002 for the mining industry and is called Process Access [Kosick, Bennett and Dobby, 2002].

It is anticipated that as more mining companies become aware of Process Access, it will become a standard interface method for facilitating geometallurgical work worldwide.

Since the completion of FLEET in 2002 and until the time of writing this paper FLEET geometallurgical modelling studies have found wide application in a range of mineral processing plants and projects world-wide for copper, lead, zinc, nickel, gold, coal, molybdenum and iron ore flotation. FLEET has been used for circuit design, flotation plant troubleshooting, optimisation, and production forecasting. Over 100 calibration-benchmarking surveys have been carried out in 17 concentrators.

Due to the limitation on paper length, case studies using FLEET will be presented in a subsequent paper.

CONCLUSIONS

It is the authors' belief that the geometallurgical modelling approach for flotation circuits, as described in this paper, is the leading scientific method to quantify the impact of ore variability on the metallurgical performance of industrial flotation circuits. The quantification of ore variability linked with geometallurgically enabled process models opens the opportunity for more accurate circuit design and the most reliable production forecasting method that is currently available. Finally, there is considerable opportunity to apply these geometallurgical technologies to optimisation studies and advanced process control to maximize the value from mine to mill.

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