Geometallurgy – The key to improving your mine value chain

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Geometallurgy or process mineralogy is an integrated part of the scoping, pre-feasibility and feasibility phases of a project as well as an existing operation. A well planned and executed geometallurgical testwork programme during the early phases, will result in a more accurate decision making process during the selection phase. The better this integration between geological, mining and metallurgical data gathering during the early phases, the more accurate the predictions will be with regards to throughput and recovery during modelling and process and equipment selection in later phases. This will enhance and improve the mine and process scheduling and subsequent NPV.

Geometallurgical information gathered during plant operation and testwork can for example be used to statistically model the pit optimisation, plant throughput and optimise the plant for certain feed material (characterised by statistical models) as can be seen in Figure 1

Keywords: economic; geometallurgical parameters; model; optimise; value

Introduction

This paper will show the readers that using geometallurgical information in various phases of a project or existing operation is of utmost importance to improve your mine value chain. As diagnostic leaching methods are typically used to characterise the leaching behaviour of a specific mineral (Lottering and Lorenzen, 2008), geometallurgical parameters can be used to describe limitations in achieving optimum process parameters during studies and in operations (Lorenzen and Barnard, 2011). The purpose of this paper is to describe the use of geometallurgical parameters in decision making processes in the mine value chain, thus at various phases in a process and/or operation (see Figure 1).

For example most mining companies (especially juniors and mid-tier's) select an optimum grind size during the various project phases purely on the metallurgical recoveries of the valuable metal in question. Very little emphasis is placed on the financial implications of such a decision in the value chain. In this paper the authors will show that that the use of the optimum parameters such as optimum grind size and subsequent comminution circuit design can be used in geometallurgical modelling of the reserve to enable the economics of the project to be maximised.

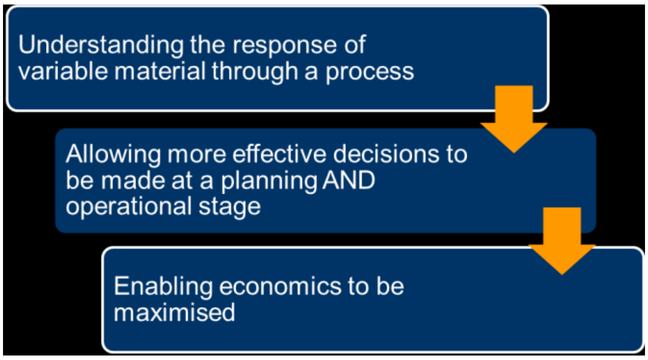

Figure 1. Geometallurgical Inputs to the Decision Process.

The costs of conducting metallurgical testwork during study phases (scoping, pre-feasibility and feasibility) are high. However, these testwork programme costs are very small in comparison to the implications of an incorrect decision in the capital and operating costs in the long term as well as mining sequence and schedule. One of the major mistakes made during the flowsheet development of a project in the study phases is to select design parameters and optimum process parameters for process design, optimisation, modelling (financial and technical) and further testwork purely on testwork results and the metallurgical recoveries achieved. This paper will focus on the selection of appropriate process parameters for a project flowsheet during feasibility stages not purely based on metallurgical results and recoveries, but on the mining and processing capabilities as well as economics of the project. This approach showed that if economic factors are used, better more informed decisions can be made during flowsheet development, mine scheduling and resource and reserve optimisation for the benefit of the company, shareholders and overall the long term viability of the project.

This paper will use two different projects as case studies to state its case. The first case study is an average grade free milling gold project with a conventional flowsheet (comminution, CIL, elution and electrowinning) and the second project is a low grade copper sulphide deposit producing a copper concentrate.

Using Geometallurgical Parameters

The author will show that using geometallurgical parameters obtained from the results of the two case studies provided in this paper enable better economic analysis of the viability of potential project/orebody. The authors will also show that without using geometallurgical parameters, incorrectly selecting for example a grind size can have major financial implications on the economic viability of the project and can cost millions of dollars to the shareholders.

Case Study 1

In case study 1 a simplified financial model was developed based on the grind/recovery data for the case study. The model was developed incorporating for example various factors and assumptions such as, capital cost multiplier of equipment capital, capital cost payback period, power cost (including power station if applicable), reagent cost, operating hours, consumable cost, valuable metal grade and throughput rate. The model then calculates the break-even gold prices at varying grind sizes from analysis of the differential capital cost and operating costs. The evaluation compares gold revenue against operating and capital expenditure for the grind sizes considered. The net revenue (gold revenue less operating cost) was calculated for each grind size. The marginal change in operating cost, gold revenue and net revenue was calculated using a base case P80, i.e. the differences in the operating cost, gold revenue and net revenue for various grinds were compared to the operating cost, gold revenue and net revenue for the selected P80.

Analysis may conclude for example that 125 micron might be a more optimum grind size as the calculated break-even gold price at 125micron was of the same order as the prevailing market gold price at the time of the analysis, compared to gold recoveries showing that 75micron might be the optimum grind size if only ounces of gold produced was driving the decision.

The Process

The ore is classified as hard and tough and the gold is of an average gold grade of 2.6 g/t (Varies from 2 – 6.8 g/t).

The mineralogical composition of the ore body consists basically of plagioclase feldspar (major), carbonates (moderate), quarts and pyrite (moderate to minor) as well as calcite and chlorites (minor). The gold assays were highly variable indicating the spotty nature of the ores. There is a variance between the Average Gold Assay Grades and the Expected Grades from geological assays. No significant concentrations of elements deleterious to cyanidation were evident in the analyses other than some samples with high copper grades. The organic carbon concentration is low which suggests that preg-robbing of the gold from solution is unlikely to occur. The sulphur assays are low.

The metallurgical testwork programme developed by the company's (owner of the project) metallurgical consultants consisted of:

Grind Establishment and Grind Optimisation (by Gravity and CIL Leach);

Size-by-Size Analysis at selected optimum grind size;

Sequential Triple Contact CIP and Equilibrium Carbon Loading;

Oxygen Uptake and Viscosity Testing;

Cyanide Optimisation.

The Master Composite Samples were ground to nominal grind sizes of 212, 150, 125, 106, and 75 micron, respectively and then subjected to gravity recovery using a Knelson Concentrator (followed by Intensive Leach) with the Knelson tail subjected to air-sparged CIL leaching using bottle rolls for 24 hours at a cyanide level of 0.07% NaCN. Results are depicted in Figure 2 and Figure 3.

At that stage of the project, the company (owner) selected the optimum grind size for further testwork as 75 micron purely on the metallurgical recoveries; approximately 30% of the samples were also tested at 106micron for sensitivity.

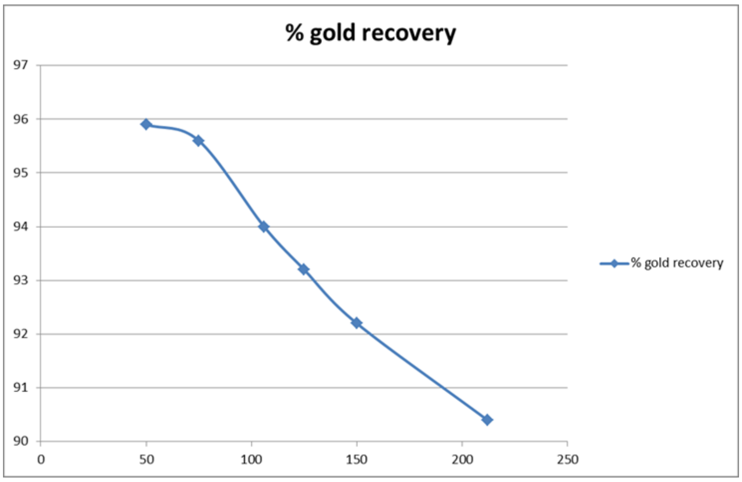

Figure 2. Gold Recoveries vs Grind Size.

From these results it was decided to continue with a grind size of 75 micron as P80 for a proposed comminution circuit. The owner requested experts in comminution circuit design to model three possible comminution circuits with the process design criteria (Table 1) from the developed from the comminution testwork programme, namely:

(1) Tertiary Crush and Ball Mill;

(2) Primary Crush SABC;

(3) Partially Secondary Crush SABC.

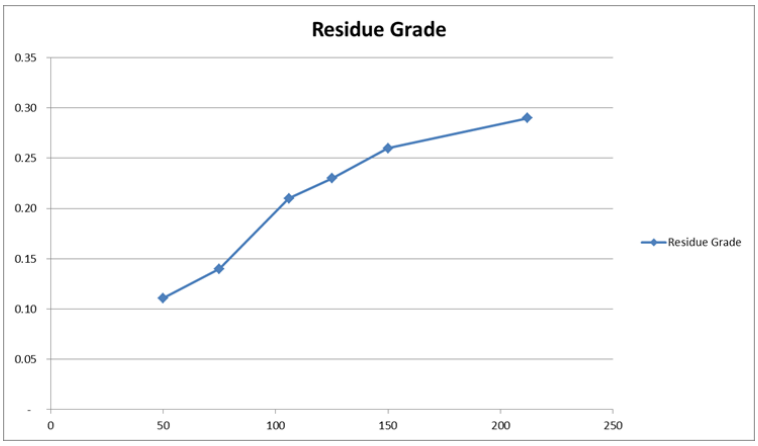

Figure 3. Residual Gold Grade vs Grind Size.

From the modelling for these circuits the following detail information was supplied, namely:

- Specific energy requirements for each circuit;
- Major equipment list for each circuit;
- Major consumable estimates for each circuit.

From this information the authors collected the following information from vendors and suppliers, namely:

- Capital cost of major comminution equipment;
- Cost of consumables;
- Energy cost for the proposed mine and project.

A simplified grind economic analysis model was developed to determine the most economic grind size for the project and to select the design grind size for process plant design. The analysis was based on the grind / extraction testwork results completed on the master composite sample selected.

Table 1. Comminution Circuit Design Criteria.

Parameters	**Units**	**Value**
	Crushing	
Throughput	Mtpa	4
	tph	615
Primary Crush P80	mm	150
Tertiary Crush P80	mm	10.5
	Grinding	
Throughput	Mtpa	4
	tph	506
Cyclone O/F P80	µm	75, 106. 125, 150, 212
	Grind Characteristics	
SG		2.74
CWi	kWh/t	24
RWi	kWh/t	25
BWi	kWh/t	20.5
Ai	g	0.448
Axb		26.8

The following inputs and sources have been used in the optimum grind size assessment:

- Plant throughput of 5,000,000 tpa;

- Milling circuit configuration based on SABC;
- 24 hours residence time (leaching);
- ROM head grade of 2.60 g Au/t;
- Three gold prices used – US$1,000, US$1,250 and US$1,500 per ounce (provided by client);
- Power requirements and comminution consumable usage rates were provided as described above, and represent the gross SAG and ball mill power and liner and media consumption to achieve the target grind. The power, media and liner consumptions are based on the average of the available comminution results ores.
- A power unit cost of US$0.33/kWh based on a Heavy Fuel Oil (HFO) power station at US$0.26/L;
- Incremental change in power station capital cost has been included at US$1,500,000 per megawatt;
- Incremental change in comminution circuit capital cost has been included at US$1,200,000 per megawatt;
- Payback for capital items 3 years;
- Costs for consumables were based on for such a project ;
- Cyanide and lime consumptions presented in the leach testwork at the grind sizes provided ambiguous reagent consumption results;
- Milling circuit maintenance costs are calculated as 4% of the mill supply capital cost, and included in the operating.

The results of the optimisation study are depicted in Figures 4 and 5. It is evident from the results that the optimum grind size selected during PFS stage of 75 micron is not the best economic grind size for the project. Results showed selected grind size is very sensitive to the gold price.

It is evident from the results that with all gold prices selected, the best economic grind size for this project is >125 micron and optimum is at about 150 micron. Thus, by selecting a grind size on recovery alone would have been disastrous as the gold price during the study phase was $1,500 per ounce and currently it is closer to the $1,250 per ounce which would have made the project marginal or uneconomic at 75 micron. A grind size of >125 micron for the feasibility study was proposed from the results, however, the company evaluated all models and still adapted 106 micron for their Feasibility Study design due to safety and risk margins used by the company.

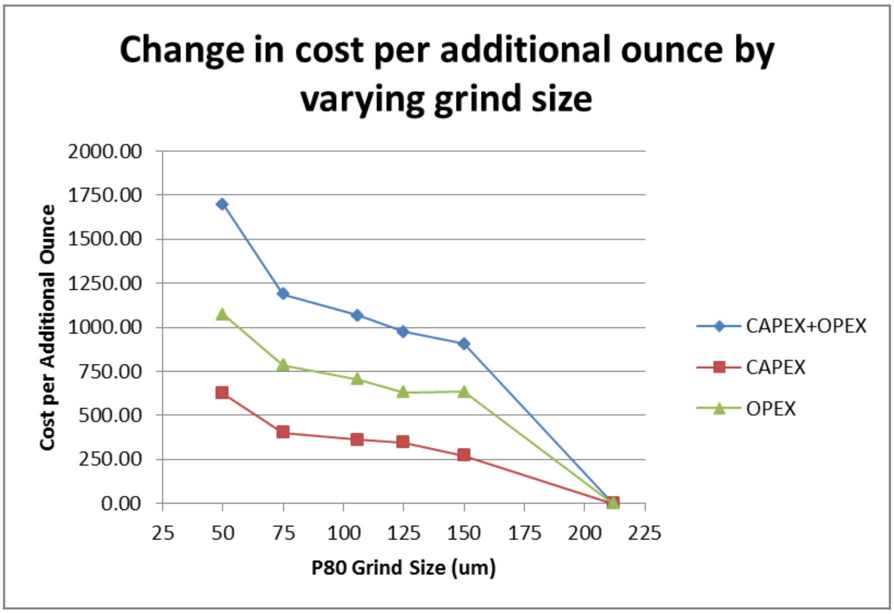

Figure 4. Cost per ounce vs Grind Size.

This process was repeated for 4 different ore domains using only one circuit namely primary crush SABC. The throughput models and cost models was then incorporated as geometallurgical inputs into a pit optimisation, thus for each ore domain there will be an optimum grind size and optimum throughput resulting a cost per ounce. The outputs from the pit optimisation process was included within an economic model to establish the best mining and processing scenario's for each ore domain as well as various blends.

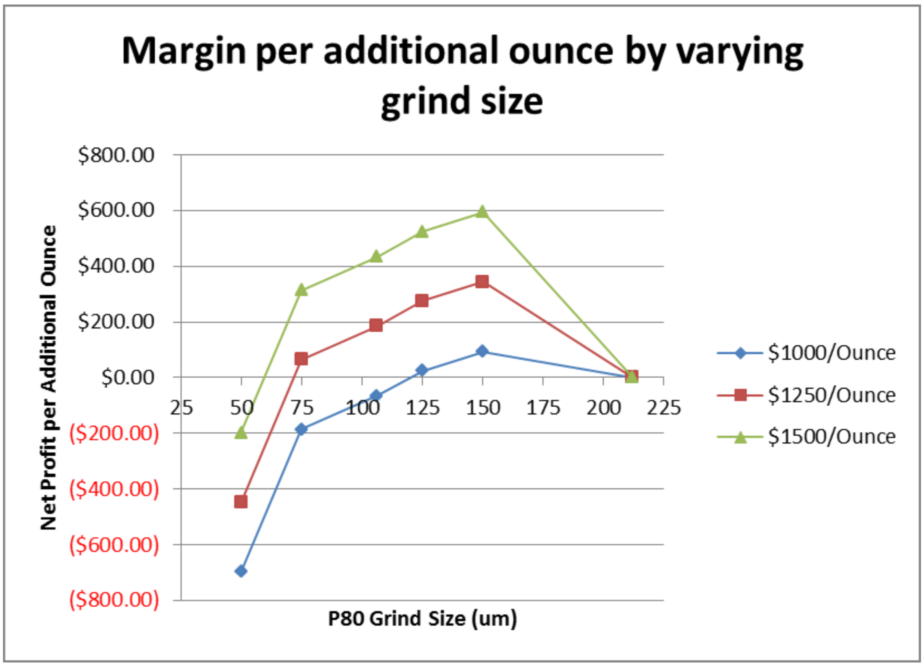

Figure 5. Net Profit per additional ounce vs Grind Size.

These geometallurgical inputs into the pit optimisation process allowed the owner to make calls on what will be the best mining strategy, i.e.:

- Processing various ore domains separately;
- Processing blends of various ore domains;

to obtain the best possible economic outcome for the project.

Case Study 2

Case study 2 is a low grade copper sulphide and copper oxide deposit in Chile, South America. The project which is located in Northern-Chile is a porphyry-copper style deposit (Cu-Au-Mo) that can be exploited using open pit mining and preparation of saleable copper concentrate from fresh ore as well as heap leaching, solvent extraction and electrowinning (SX-EW) of copper from oxide ores. The deposit is a Cu-Au-Mo deposit and various different geometallurgical models were developed for this deposit by various researchers to enhance the project value, to understand the various variability issues within the ore body and to predict the process response of both the oxide and sulphide ore. This case study will focus on the latter, thus to predict the process response of both the oxide and sulphide ores.

Escolme, et al. (2016) developed some predictive geometallurgical models to develop the Cu-Au-Mo deposit based on geochemistry to reflect the variability in Cu sequential leach data (i.e. oxide, transitional-oxide, transitional sulphide, sulphide and non-recoverable Cu). A simple Cu species classification scheme based on sequential leach data and S per cent was devised to account for the non-recoverable CU (i.e. non-sulphide Cu) which is insoluble in weak acids. Through machine learning a proxy for the Cu species classification scheme was developed. The validity of the proxy Cu species classification models was tested against results from the flotation analysis and initial results shows that geochemical proxies can be used to successfully predict Cu species class and provide high density of classification data.

King and Macdonald (2016) developed a predictive geometallurgical model with the authors by using both discovery and integration aspects. In their paper they discussed the concepts of geometallurgical modelling in terms of the underlying relationships that are used in geology, metallurgy and economic value, and how the early-stage preparation of spatial geometallurgical models enhances project value and provides for a sound basis for further studies. This economic geometallurgical model by Kinga and Macdonald (2016) used tests results from mineralogical, metallurgical and comminution testing, as well as other studies completed on a suite of samples selected from t6he many geological domains of the deposit.

Case study 2 will focus on showing how the authors developed the various index proxies and relationships to be used in the King and Macdonald (2016) geometallurgical economic model. Using Bond Work index and Abrasion Index proxies in the geometallurgical model, the model could predict the variability of sulphide ore throughput and comminution costs for example. Heap leach acid consumption in oxide ore was estimated from results and drill hole calcium concentrations. This model was then used in the mine scheduling to identify high and low throughput in sulphide plant and high and low acid ore zones for processing in oxide heap leach.

The Process

General Information of Deposit

- Copper Oxide and Copper Sulphide Deposit (with recoverable molybdenum);
- Determine the effect of grind size (P80) on copper sulphide flotation response;
- Best grind size for optimal copper recovery (taking into account molybdenum recovery);
- Deposit in the Americas;
- Client's initial decision: select grind size of 150 to 180 micron (previous experience and initial rougher testwork in previous phases).

Metallurgical Testwork:

- Rougher and Cleaner Copper Flotation testwork at various grind sizes:
    - 180, 150, 125, 106 and 75 micron;
- Standard Comminution Testwork – SAG, Ball, HPGR, Crusher, etc.

Assumptions:

- Recoveries from the grind series testwork - only from the main pit. Data equalized for grade.
- Actual comminution testwork results used.
- Ball mill capital costs – from quotation.
- Power cost US$0.10/kWh – from client.
- The marginal operating cost includes ball mill power, grinding media and liners.
- Initially only ball mill capital and operating costs varied with grind size.
- Copper price used (US$ 6,000 per tonne) is nett of TC-RCs.
- Net revenue calculation: (Copper Price -TC-RCs) -marginal operating cost - marginal capital cost.
- Marginal operating cost includes ball mill power, grinding media and liners.
- Marginal capital cost is the installed ball mill cost divided by a nominal payback period of 5 years.
- The emphasis of this analysis is to define a design point for the Ball Mill:
- In operation, actual grind size and throughput can be varied.

A BWi proxy correlating the elemental concentrations of aluminium and potassium to the BWi for the main body of the deposit was developed (King and McDonald, 2016):

$$BWi = 0.9796\ Al + 1.5071\ K + 3.3686 \qquad (1)$$

The authors of this paper added a relationship between throughput and BWi (see Figure 6) to the main body of the deposit as well to ensure throughput is also included as a function of hardness.

The authors of this paper were also able to derive the key variable processing costs as functions of the BWi and Ai in order to utilise the geometallurgical model to become an economic analysis tool. The key variable processing costs identified as having relationships to the BWi were:

- Power cost;
- SAG mill grinding media cost;
- Ball mill grinding media cost; and,
- Process plant throughput.

Having defined unit costs for energy and grinding media, the variable cost functions of BWi and Ai for the variable processing costs were:

$$\text{Power Cost} = 0.2067\ \text{BWi} - 1.6051 \tag{2}$$

$$\text{SAG Mill Media Cost} = 0.01733 \times (700/\text{BWi}) - 0.07542 \tag{3}$$

$$\text{Ball Mill Media Cost} = (0.0794\ \text{Ai}^{0.498})\ (1.667\ \text{BWi} - 19.367) \tag{4}$$

An output of the comminution modelling was the relationship between the BWi and the specific comminution circuit energy required to achieve the design grind size. The underlying assumption is that the SAG mill would be the limiting factor in the circuit and therefore the energy consumption is constant. The variable energy consumption occurs within the ball mill. Modelling the specific energy of each component in the comminution circuit across a range of BWi's allowed the corresponding power costs to be evaluated for each BWi and the empirical relationship established (Figure 8).

Relationships between the BWI, Ai and the grinding media consumptions – in turn grinding media costs – were established from the comminution modelling results. Across a range of BWI's, the grinding media costs were calculated to determine the resultant relationship (Figure 7).

At a fixed grind size, the plant throughput is constrained by the limiting energy of the comminution circuit. The direct relationship between the BWi and the plant throughput was developed via equating the BWi/specific energy relationship with the comminution circuit power limit (Figure 8).

$$\text{Throughput} = 113829\text{BWi}^{-1.461} \qquad (5)$$

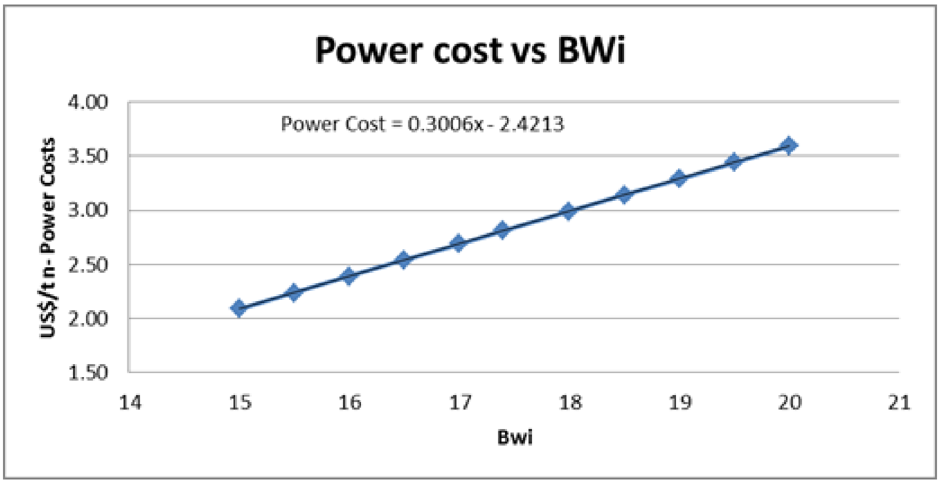

Figure 6. Power Cost as a Function of BWi.

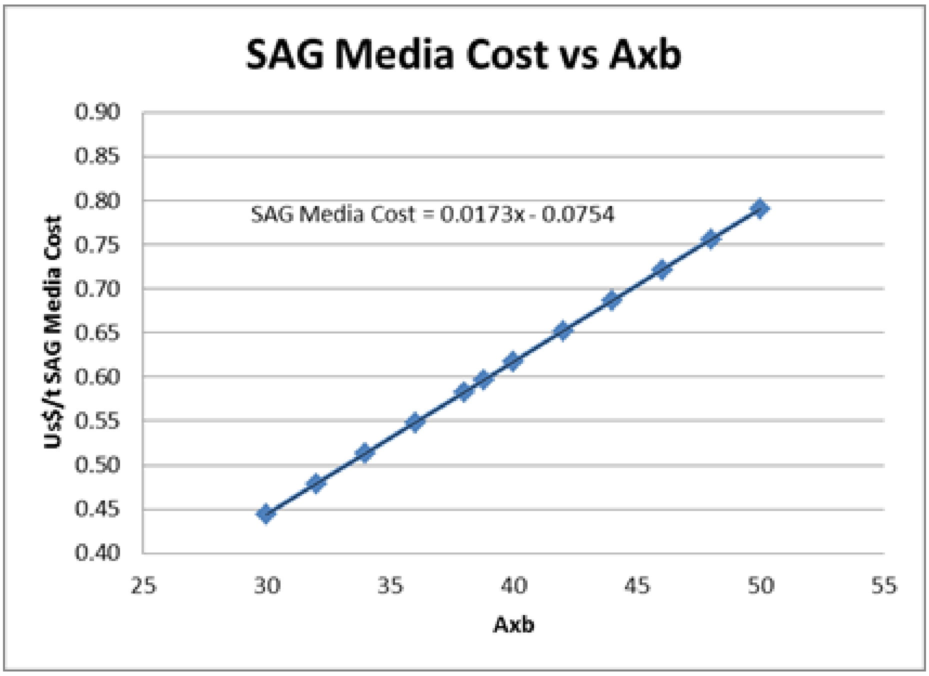

Figure 7. SAG Mill Grinding Media Cost as a Function of A*b Parameter.

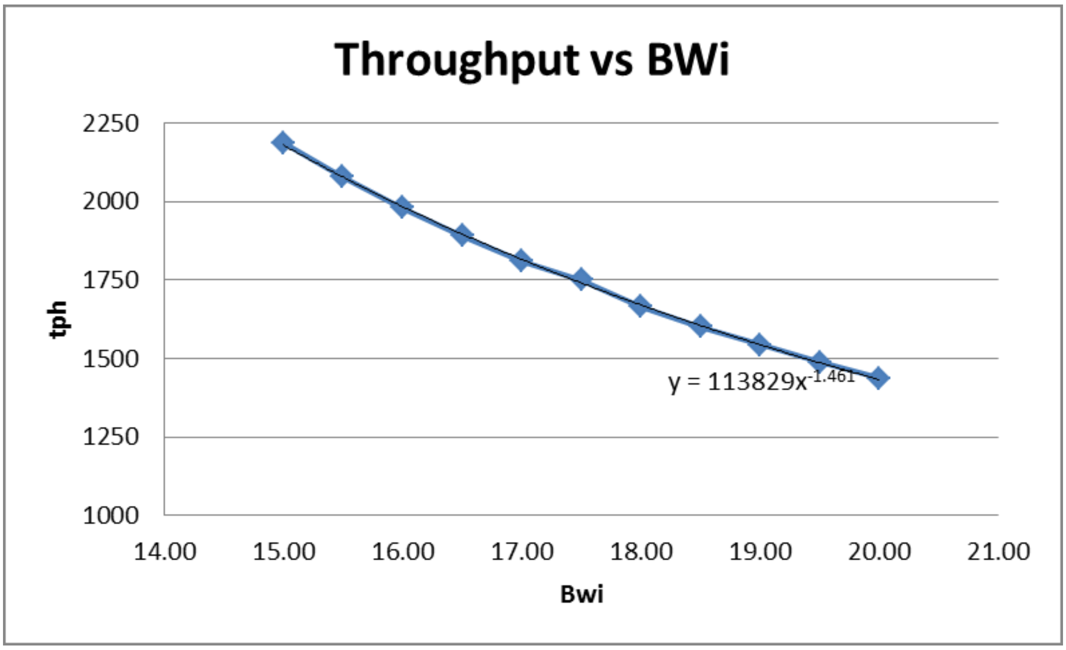

Figure 8. Throughput vs BWi.

The development of these four relationships allows the geometallurgical model to become a tool for economic analysis. The application of the proxy to the deposit to estimate the BWi in turn allows the estimation of the variable processing costs. Certain high or low processing cost ores can be brought forward or deferred in the mining schedule to improve the overall NPV of the project.

Conclusions

This paper describes the use of various geometallurgical parameters using two case studies from different ore bodies to model the performance of a processing plant. The paper shows that it is of utmost importance to understand the response of variables throughout the process. Using relationships between these variables (geometallurgical variables), geological models as well as economic models allow the authors to make effective decisions at both a planning and operational stage. That enables the authors to maximise economics and thus viability of the project.

The results from a well-designed geometallurgical programme can thus be used for:

- Better flowsheet design (more flexible);
- Better use of algorithms for throughput and recovery in resource and reserve models;
- Better use of the mining schedule to optimise plant performance;
- Better plant and equipment design and sizing;
- Optimise plant performance and forecasting;
- Reduce risk in subsequent phases; and,
- Enable economics to be maximised.

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