A GEOMETALLURGICAL STRATEGY FOR IMPROVING ORE QUALITY AND MINERAL PROCESSING EFFICIENCY AT KANSANSHI MINE IN ZAMBIA.

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A research report submitted to the Faculty of Engineering and the Built Environment, University of the Witwatersrand, Johannesburg, in partial fulfilment of the requirements for the degree of Master of Sciences in Engineering.

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DECLARATION

I declare that this research report is my own unaided work. Where work of other authors has been used, it has been duly acknowledged. It is being submitted for the degree of Master of Science in Engineering to the University of Witwatersrand, Johannesburg. It has not been submitted previously, for any degree or examination, to any other University. All data used within this research report was collected while I was employed by First Quantum Minerals Ltd. in the capacity of senior mine geologist, at the Kansanshi copper and gold mine, near Solwezi in north-western Zambia.

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ABSTRACT

The Kansanshi mine is located in north-western Zambia. It is a copper and gold bearing, vein hosted, structurally controlled, ore deposit, which is exploited using conventional open pit mining methods. A series of highly complex mineralogical suites have formed through the interaction of the *in-situ* geological, weathering and oxidation processes. Some of these mineralogical suites are extremely difficult for effective extraction of copper and gold. Currently ore is classified into 22 different quality categories using a system called "Mat_Type". Only the "best quality" ore is directed to the metallurgical process plant, while the remaining "poor quality" ore is directed to long term stockpiles. These stockpiles are unlikely to be processed until the end of the life of mine despite containing metal quantities of significant value.

A systematic investigation of Kansanshi's mine value chain was carried out to determine if this value could be realised sooner. It was found that, due to a lack of integration between technical silos that form the mine value chain, the Mat_Type system does not take due consideration of geological, mineralogical nor metallurgical processes. Ore quality control factors are incorrect and economic data to determine suitable cut-off grades is both outdated and applied in an inappropriate manner. As a result 12 of the 22 Mat_Type ore categories are unnecessary, while a further six categories are inaccurately defined, leaving only four categories that can be considered to be correct.

It is because of these errors, that so much ore is being directed to long term stockpiles. Through the research study presented in this report it was found that five key factors determine an effective ore classification system for Kansanshi mine. These factors can be defined as:

- Spatial distribution of mineralogical relationships between *in-situ* geological, oxidation and weathering domains;
- The impact of mineralogical groupings on copper recovery in each metallurgical process;
- The size, statistical distribution, range and accuracy of available data sets;

- The application of appropriate economic factors in the mineral resource to mineral reserve conversion process; and,
- The practicality of overcoming physical constraints at various stages of the mining process.

A more appropriate geo-metallurgical ore classification system can be developed that will consider the above listed factors. Three mineralogical groupings naturally exist in the deposit. Using total copper and acid soluble copper assay data, these mineralisation categories can be defined by a specific range of oxidation ratio. These ranges can be listed as:

- Primary copper sulphide minerals, with oxidation ratio < 0.1 ;
- Secondary copper sulphide minerals, with an oxidation ratio between 0.1 to 0.8; and ,
- Primary copper oxide minerals, with an oxidation ratio between 0.8 to 1.0.

Each mineralogical grouping can be assigned to a specific metallurgical process. Further subdivisions of these groupings can be made based upon economic grade ranges and appropriate metallurgical quality control factors which are linked to a specific metallurgical process.

By implementing the proposed geo-metallurgical ore classification system, 12 of the erroneous ore categories in the Mat_Type system can be removed. The remaining 10 categories would be accurately and consistently defined. This will lead to a significant reduction in the quantity of ore directed to long-term stockpile, thereby releasing previously lost value.

The new ore classification methodology proposed would be supported by a systematic process to manage regular, periodic updates, based on new data and developments in technical understanding across all functional areas of the mine value chain. A continued dialogue and sharing of knowledge between the main technical silos is critical when promoting a robust and integrated ore classification system. Such a system has the potential to remain relevant throughout the life of Kansanshi mine.

DEDICATION

To my darling wife Chinga, thank you for all your love and support.

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LIST OF ABBREVIATIONS

AICu	Acid insoluble copper
AOR	Acid to ore ratio
ASCu	Acid soluble copper
Brn	bornite
CACB	Central African Copper Belt
Cal	calcite
CCD	Counter current decantation
Chc	chalcocite
Chry	chrysocolla
COG	cut-off grade
Cov	covellite
Cpy	chalcopyrite
Cup	cuprite
Dig	digenite
GAC	Gangue acid consumption
GFC	Global financial crisis
g/t	grams per tonne
Gth	Goethite
HG	High grade
LG	Low grade
Mal	malachite
MF	Mixed float
Ms	Muscovite

mtpa	million tonnes per annum
OMO	Oxide marginal ore
OMW	Oxide mineralised waste
PLS	Pregnant liquor solution
Py	pyrite
Pyr	pyrrhotite
Q/Q	Quantile-Quantile comparison
Rt	Rutile
SAG	Semi autogenous grinding
SEM	Scanning electron microscope
SIBX	Sodium isobutyl xanthate
SIPOC	Supplier, Input, Process, Output, Customer
SMO	Sulphide marginal ore
SMW	Sulphide mineralised waste
SX/EW	Solvent extraction and electro-winning
TAC	Total acid consumption
TCu	Total copper
Ten	tenorite
VOC	Voice of Customer

1.0 INTRODUCTION

1.1 Business context

The Kansanshi ore deposit is located in the Northwest province of Zambia, 10km north of the provincial capital Solwezi and approximately 160km west of Nchanga mine in the Central African Copper Belt (CACB) (Broughton and Hitzman, 2002). The town of Solwezi, the provincial capital of the Northwest province is situated approximately 20km south of Kansanshi mine. A location map of Kansanshi mine with respect to other major copper deposits of the world-renowned Central African Copper Belt is presented in Figure 1.1.

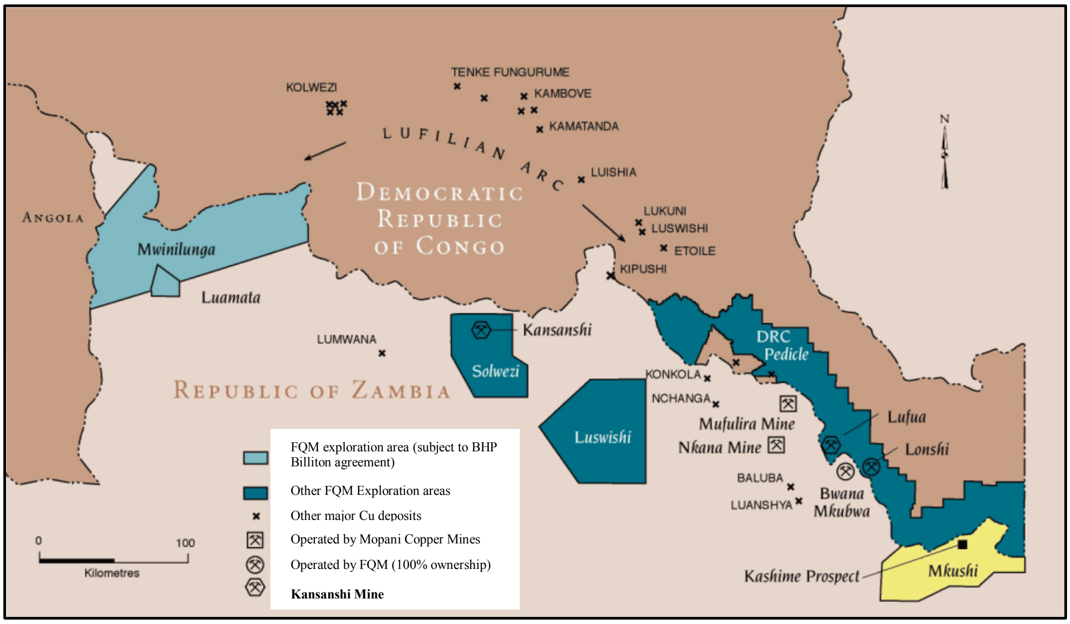


Figure 1.1 Map of major deposits in the Central African Copper Belt (First Quantum Minerals Ltd., 2004).

The deposit is currently being exploited by Kansanshi Mining Company PLC., a joint ownership company between First Quantum Minerals Ltd. (FQM), with an 80% share, while the Zambian national mining company; Zambian Consolidated Copper Mines - International Holdings (ZCCM-IH) owns the remaining 20%. The mine consists of three mineralised areas namely, Northwest, Main and Southeast Dome. The Northwest and Main areas are being mined concurrently while the Southeast Dome is being drilled to define a geological resource. The deposit is located in a broad antiform that is approximately 12km long and strikes northwest to southeast. Several regional scale faults bound the antiform to the east and west and a series of localised vertical to sub-vertical, normal and reverse faults cut across the antiform in a fracture mesh pattern. These faults are generally mineralised. Late intrusive gabbros cut off the mineralisation to the north and east of the antiform. A map of the local geology and location of mining areas is presented in Figure 1.2.

Exploration diamond drilling has found a conglomerate termed "pebble schist" at depths of 250m to 500m below surface. This is believed to be the "Kakontwe Grand Conglomerate", a sedimentary facies marker at the contact between the Nguba Group (formerly the lower Kundulungu Group) and the Upper Roan Group (Selley *et al.* 2005; Hitzman *et al.* 2012). The Kansanshi deposit is a structurally controlled, vein hosted, hydrothermal copper deposit (Torrealday *et al.* 2000). The mine's stratigraphic unit and associated mineralisation occurs much higher in the stratigraphic sequence of the Katangan Supergroup than other Central African Copper Belt deposits (Broughton and Hitzman, 2002). To illustrate the unique geological setting of the Kansanshi deposit with regards to other large scale copper deposits found in this region, reference is made to the lithostratigraphic section of the Katangan Supergroup presented in Figure 1.4.

Due to geo-physical and geo-chemical processes, mineralisation in the upper portion of the ore deposit has oxidised from a dominantly chalcopyrite, pyrite and pyrrhotite assemblage to a suite of secondary copper sulphide mineral and primary copper oxide minerals including bornite, digenite, chalcocite, malachite, tenorite and chrysocolla. The mineral assemblages have a high degree of variability over short ranges both laterally across the deposit and vertically within mineralised structures.

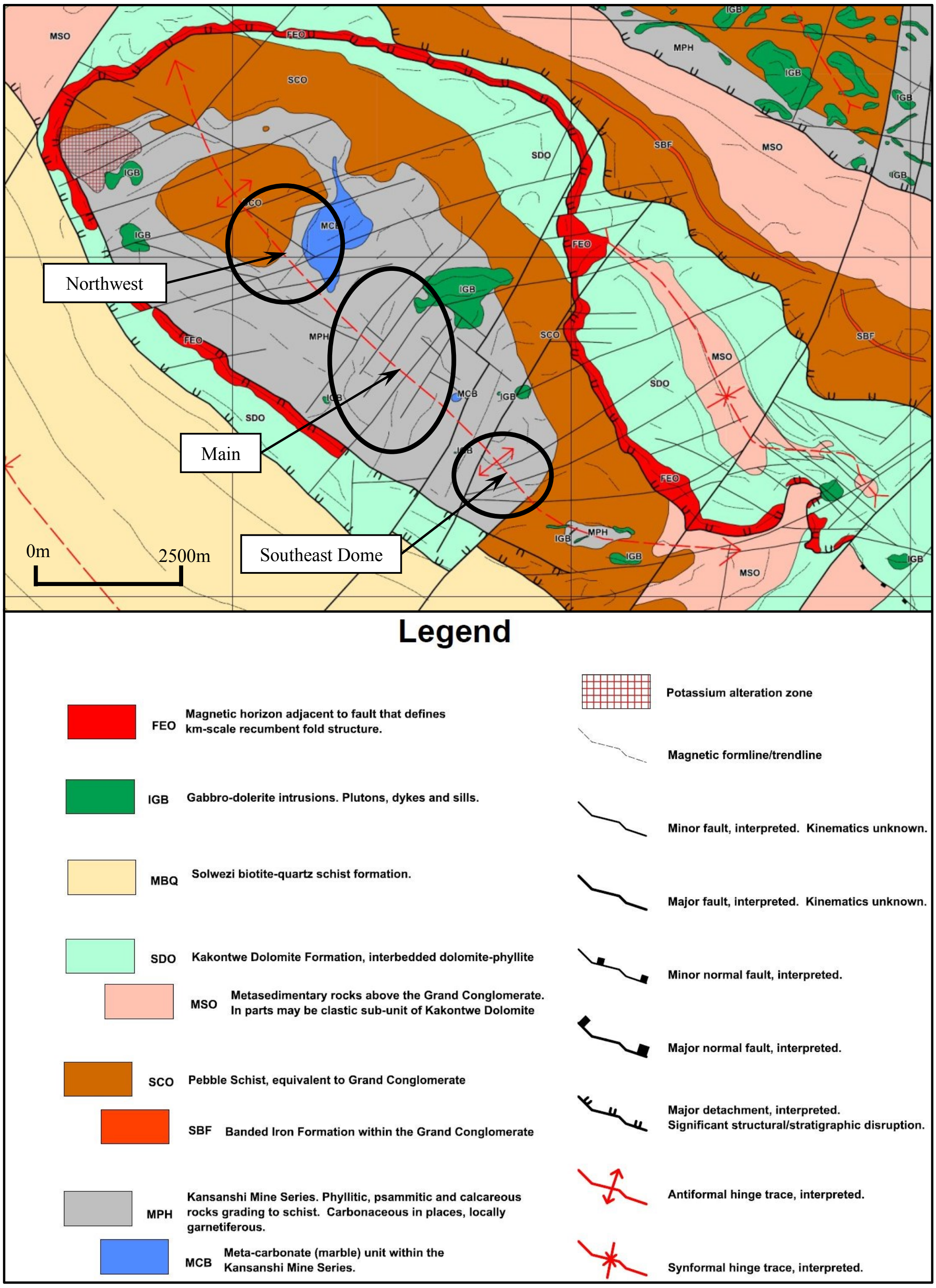


Figure 1.2 Geological map of the Kansanshi ore deposit (Adapted from Wood, 2011).

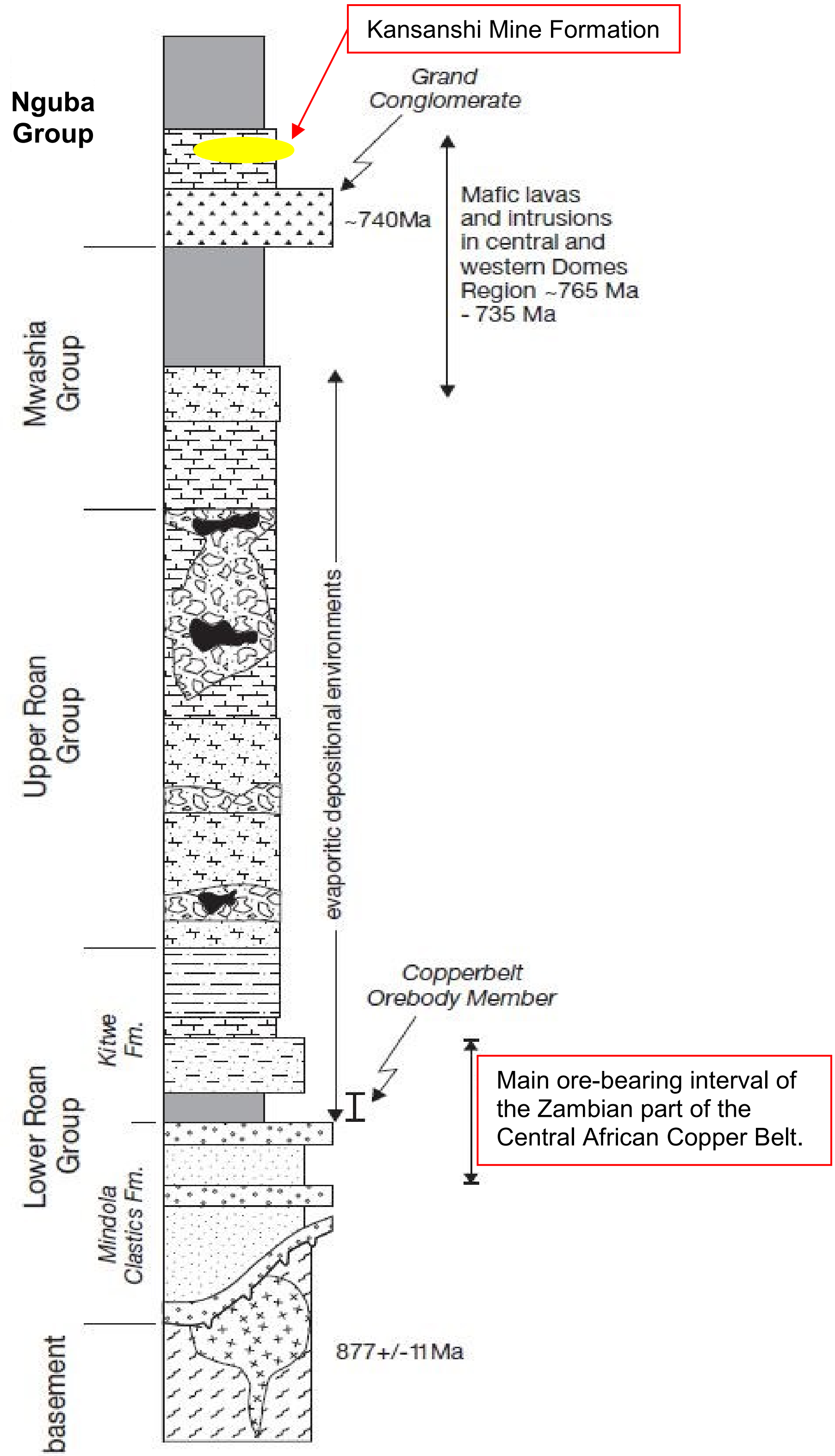


Figure 1.3 A lithostratigraphic section of the Zambian part of the Central African Copper Belt. (Adapted from Selly *et al.*, 2005 and Hitzman *et al.*, 2012)

1.2 Current mining and processing strategy

Currently copper and gold ore is only mined from the Northwest and Main deposits, utilising conventional “drill and blast, load and haul” methods. The Southeast Dome deposit is scheduled to start production during the first quarter of 2017. Ore is transported to the process plant where it is crushed and milled before undergoing a variety of specific metallurgical processes to extract the copper and gold bearing minerals from the host rocks.

Due to the effect of weathering and oxidation processes over-printing and modifying the mineralogy in the deposit, Kansanshi mine has had to adopt a highly selective mining process, which separates ore into three categories for mineral processing. These categories are:

- <u>Sulphide Ore:</u> dominantly composed of chalcopyrite and processed using a standard Xanthate floatation system to produce a copper concentrate at an average grade of 24% copper;
- <u>Mixed Ore:</u> contains a wide range of mineralogy, often termed refractory because it is difficult to recover copper effectively. Ore is processed in a modified floatation system using a suite of specialised reagents, to produce a second copper concentrate with varying grade; and
- <u>Oxide Ore:</u> dominated by a suite of oxide minerals (malachite, tenorite and chrysocolla) which is amenable to an acid leach process followed by solvent extraction and electro-winning (SX/EW) to produce copper cathode with an average purity of 99.99% Cu.

Currently the metallurgical plant is optimised to recover copper either from the sulphide floatation circuit or the oxide leach circuit which utilises the solvent extraction and electro-winning process. Only a small tonnage of the highest grade mixed ore is treated. The majority of the tonnage is stockpiled because copper recoveries are so low that it is considered uneconomic to process the ore despite being above the mineral reserve cut-off grade (COG) of 0.19% TCu. Due to the complexity of the metallurgical process, each of the three broad ore categories is subdivided into a number of different quality control bins which are used to control copper grade and ore blending as part of the overall business strategy.

A total of 226,673 tonnes of copper was produced at Kansanshi mine in 2015. A copper production profile for Kansanshi mine is presented in Figure 1.5. The proportion of copper cathode and copper concentrate from first production in 2005 to year end 2015 is presented as a stacked bar chart.

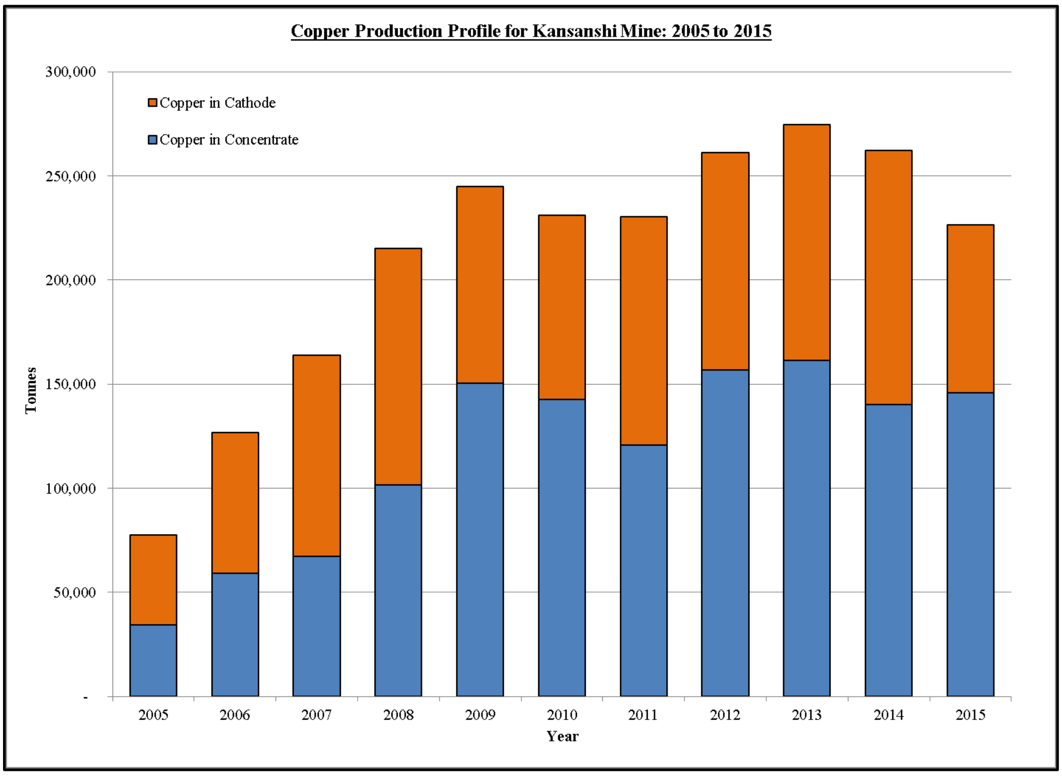


Figure 1.5 Kansanshi mine copper production profile from 2005 to 2015, (Holt 2016).

Gold is also recovered from all metallurgical circuits via gravity concentration methods. The majority of gold produced is in the form of gold Dore at a minimum purity of 92% gold. Approximately 5g/t of gold reports to the copper concentrate which is only recovered during the smelting process. A gold production profile for Kansanshi mine is presented in Figure 1.6. The proportion of gold Dore and gold in copper concentrate from first production in 2005 to year end 2015 is presented as a stacked bar chart. The mine produced a total of 136,258 troy ounces of gold in 2015.

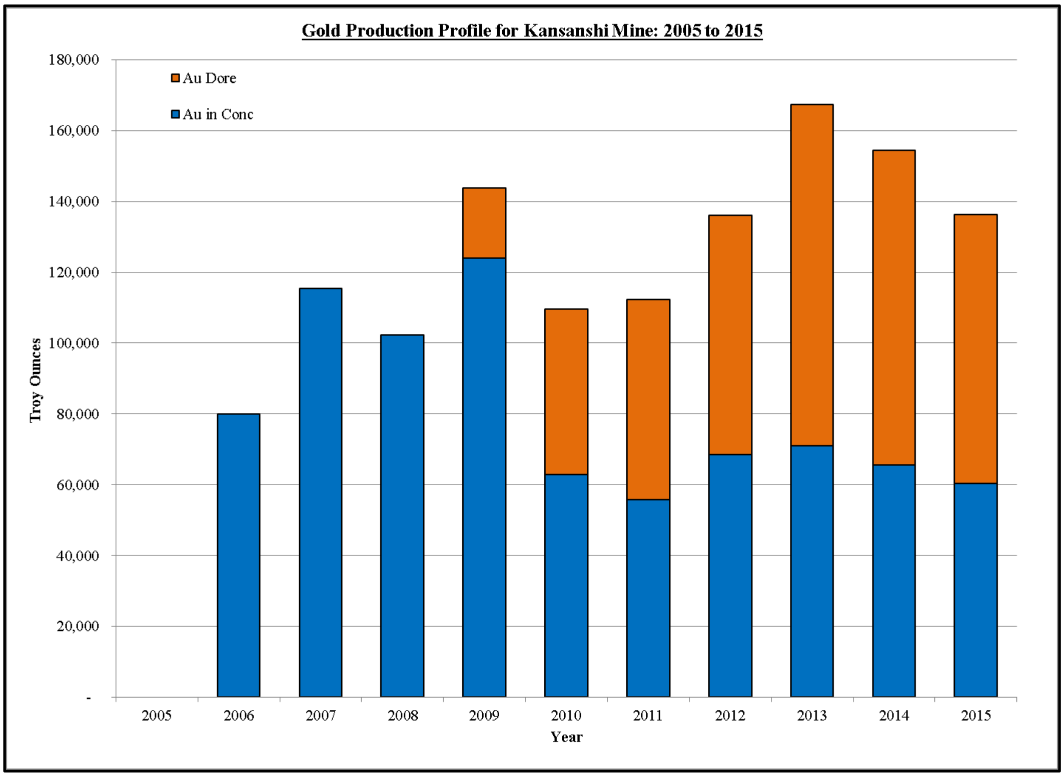


Figure 1.6 Kansanshi mine gold production profile from 2005 to 2015, (Holt 2016).

1.3 Business expansion

In January 2011 Phillip Pascall, the Chairman and Chief Executive Officer of First Quantum Minerals, announced a substantial investment in a multiphase expansion project at Kansanshi mine. In a public statement to the company's shareholders Mr Pascall outlined his strategic goal as:

> *"At the Kansanshi operation, an expansion project is underway to expand annual copper production capacity from the current 250,000 tonnes to 400,000 tonnes of copper in 2015"*
>
> (First Quantum Minerals Ltd. 2011. Pp 2.)

To achieve this, the capacity of the sulphide ore processing infrastructure was increased from 12.5 million tonnes per annum to 25 million tonnes per annum

(First Quantum Minerals Ltd. 2013). Due to a lack copper smelting capacity to process this increased concentrate output from the mine, a copper smelter was also constructed at the Kansanshi mine site. It is capable of treating 1.2 million tonnes of concentrate per year. The smelter was expected to operate at 80% of its design capacity by mid-2015 and achieved 100% in early 2016.

A by-product of the smelting process is sulphuric acid, a key chemical reagent used at the Kansanshi mine site to leach copper from oxidised ores. Coupled with the construction of the smelter is an expansion to the Oxide Leach plant from 8 million tonnes per annum currently, to 17 million tonnes per annum in 2016. This expansion will provide a use for the sulphuric acid, produced by the smelter which, if not used, can potentially lead to the shutting down of the smelting operation.

Since 2011 significant work has been carried out to expand the processing plant and optimise both tonnage throughput and metal recovery. Furthermore significant work has been carried out to increase the size of the Kansanshi ore resource and better understand complex mineralogical associations found in various domains of the deposit. Yet despite the large programme of business expansion projects and the significant increase in metallurgical and geological understanding of the deposit, very little work has been done to fully integrate all technical knowledge, in support of the business strategy going forwards.

1.4 Problem statement

Reconciliation work by Holt (2014) indicated that as of 31st December 2013 a total of, 24.3 million tonnes mixed ore, at 0.82% total copper (TCu), 0.32% acid soluble copper (ASCu) and 0.25 g/t gold (Au), had been deferred to long term stockpiles despite being above the cut-off grade of 0.19% TCu. Given the current mining and processing strategy as outlined by the Kansanshi mine five-year business plan, Hulbert (2014) noted that a further 91.9 million tonnes of ore at a grade of 0.84% TCu, 0.22% ASCu and 0.23 g/t Au, will be stockpiled over a period of five years from 2015 to 2020. This mineralised material is considered to be of poor metallurgical quality and thus difficult to process efficiently. The ore that is being directed to stockpile equates to 812,690 tonnes of copper and 711,820 ounces of gold. Based on input data to the 2014 life of mine optimisation scenario, Lawlor (2014) noted that based on the assumptions in the 2014 life of mine optimisation scenario, these stockpiles will have a combined value of $6.2 billion by the end of the life of mine which is unlikely to be realised.

Significant improvements in geological and metallurgical understanding have brought many benefits within their own technical function. However this knowledge has not been effectively integrated into the mining value chain. With cheap and abundant acid from the new smelter soon to be available and an increase in ore tonnage through all processing circuits, the current grade control strategy of stockpiling "poor quality ore" may not be appropriate going forwards. These circumstances make it imperative to answer the question:

"Can a geometallurgical strategy be designed and implemented that will integrate technical knowledge across corporate and site level departmental silos, to improve the flow of value from an in-situ location in the deposit, along the mine value chain, to a final saleable product, thereby reducing the need for stockpiling and improving the company's revenue stream?"

1.5 Overview of report structure

This report provides a detailed description of the work done to understand geological, metallurgical and economic factors that impact the revenue stream at Kansanshi mine. The purpose of this study was to ascertain the factors that can be used advantageously to increase revenue and those that need to be controlled to minimise impact.

A literature review is presented in Chapter 2 which investigates a number of geometallurgical case studies from other mines around the world. This literature review is to establish whether increased value can be realised from an ore deposit by employing a robust geometallurgical strategy.

From the literature review a methodology of research was established within the context of the problem statement being investigated at Kansanshi mine. This research methodology is presented in Chapter 3. The results of a series of systematic technical reviews of key aspects of the business are then presented in subsequent chapters. Firstly a review of geological data is presented in Chapter 4. This provides an understanding the ore deposit particularly with reference to the creation of geological domains based on the effect of oxidation and weathering processes on *in-situ* copper bearing mineralisation. Chapter 5 discusses gangue acid consumption and the method used to quantify the amount of sulphuric acid that is consumed in the oxide ore metallurgical process. An investigation into the application of a number of quality control factors that are used to control *in-situ* ore quality, as part of the grade control strategy are presented in Chapter 6. A review of the mineral processing systems employed at Kansanshi mine and the associated technical challenges faced as a result of the natural variability found in the ore deposit, are then presented in Chapter 7.

The results of these three key technical reviews (geology, grade control strategy and mineral processing) are brought together in Chapter 8. Analysis and discussion of the data gathered in earlier chapter is presented, to determine where potential opportunities exist for business improvement within a geometallurgical context. Finally a set of conclusions and recommendations to improve the integration of geological and metallurgical data into the grade control strategy and align technical silos along Kansanshi's mine value chain is presented in Chapter 9.

2.0 LITERATURE REVIEW

2.1 Introduction

This research project covers a broad range of technical disciplines within the mining business and key to its success is the ability to understand where specialist technical knowledge needs to be integrated into the business decision making process. Mining companies are now embracing geometallurgy as part of their core strategy. Typically, each technical area in a company that forms part of the mining value chain remains a defined silo. A silo is often optimised within itself without considering the impact on the creation and preservation of value either upstream or downstream of the silo walls. Geometallurgy is a strategic process that drives integration between these silos thus improving the flow of knowledge through the value chain. Geometallurgy has become an increasingly important and recognised field over the past decade; this is mainly attributed to the Amira International P843 "Geometallurgical mapping and mine modelling" project.

As First Quantum Minerals has grown, systems to support the integration of technical data into many of the complex decision making processes have not kept pace with the expansion of each business unit. In the day to day operational environment putting such integrated practices into effect tends only to be done at a higher managerial level. Critical understanding of the detail of complex issues is often not effectively communicated downwards to the lower managerial levels responsible for executing the plan. As a result strategies to improve aspects of the business are generally met with limited success.

Due to the interaction of a number of technical disciplines, it is important to understand the context of a geometallurgical program at any stage in the mining value chain (Keeney and Walters, 2011). The following sections provide a number of case studies where geometallurgical principles and processes have been applied to better understand risk of a particular ore deposit. This understanding of current industry practices will act as a foundation to assist in developing a methodology of research that will drive data collection, analysis and interpretation from the key technical silos that form Kansanshi's mine value chain.

2.2 Geometallurgical concepts

Geometallurgy is often a misunderstood and poorly defined concept (David, 2013). Definitions change from author to author based on the scale or level of detail that is being discussed. However a number of broad definitions can be found in the current literature. Geometallurgical mapping and modelling is a process documenting and quantifying ore body variability (host rock, alteration and structure) and the mineralogical impacts on metallurgical response and metal recovery (Williams and Richardson, 2004). Geometallurgy is a cross discipline approach combining geology, mine planning and metallurgy to improve the design and operation of the mining business (Dunham *et al.* 2011). By integrating geology, mining operations, mineral processing and metallurgy, geometallurgy aims to improve the fundamental understanding of resource economics (Baumgartner, 2012). It is not a mechanism guaranteed to improve resource value, but rather a means of reducing uncertainty leading to a robust, predictive business environment enabling effective mine planning and optimisation (Keeney and Walters, 2011). A number of case studies can be found in the current literature describing successes and failures of applying geometallurgical principles. These case studies can often be divided into one of two broad areas of study; early stage studies leading into a feasibility study, or later stage studies to optimise an already producing mine.

The first area of study is generally termed “geometallurgical projects”. These are often driven by a single corporate champion who realises the need for this work. This type of project is intended to develop an understanding of the geology and mineralogy at the early stages of exploration, which will lead to the development of a geological model with *in-situ* geological domains that have similar geological or mineralogical characteristics. Such models would then go on to be used in scoping studies to determine subsequent phases of geological, mineralogical and metallurgical test work. Through subsequent phases of work a more detailed understanding of each of the geological domains is developed. Each stage of test work provides greater understanding of the variability of the ore deposit and through an iterative evaluation of data determines the next phase of data gathering. Ultimately this knowledge feeds into a feasibility study to assist in the design of a strategic mine plan and suitable metallurgical process plant.

The second area of study is focussed on detailed understanding of the characteristics of ore to optimise a specific aspect of an already producing mine. This work is often driven by the onsite metallurgical team and is usually reported as a "mine to mill optimisation study". Such studies are implemented as an opportunity to realise potential value that is possibly being lost. The work is retrospective as capital investment has already been made in the process plant and mining fleet. This imposes a set of predefined constraints on the type of work that can be done, leading to limited opportunities for improvement.

2.3 Early stage geometallurgical investigations.

During the early stages of a mineral resource definition project geological data is usually gathered at defined interval spacing, in a specific orientation based on the geologist's best interpretation of the strike and continuity of the mineralised system. This spatial constraint is often based on a compromise between exploration cost and deriving meaningful numbers that can be used by the company's executive management to raise more capital for a pre-feasibility study.

Typically at the early stages of ore body development the key geometallurgical focus is on creating domains of similar geological or mineralogical characteristics. As was discussed by Armstrong *et al.* (2011) after their work on the Trilogy polymetallic deposit in southern West Australia;

> *"Geological domains are often defined in broad terms based on the style of mineralisation, assay values and lithology. These domains, while adequate for the resource estimation process, fail to translate into representative metallurgical domains."*

(Armstrong *et al.* 2011. Pp 1.)

As Johnson and Munro (2008) noted, an initial assignment of geological ore domains can be successfully based on the mineral composition of the ore (valuable and gangue minerals) and other physical and chemical processing related properties.

Due to the timeframe needed to develop an exploration project to a producing mine, most exploration geologists tend not to be exposed to the downstream processes that will be employed if the ore body was to become a producing mine. This lack of exposure to downstream processes means that an exploration geologist often does not think about the finer details that will impact daily mine production.

A three dimensional, geological block model is created by the exploration team using available data and interpretations. This is handed to a team of mining engineers and process design engineers to develop a mine plan and process plant. At the point of handover the mining engineers and process engineers are often part of an external consultancy firm and are situated many miles from the ore body. The detailed and intimate geological understanding of the true variability of the deposit which has been learned through the hard work of the exploration team does not move effectively to the next stage of the development process. Work by Begg (2013), David (2013), Dunham and Vann (2007), Richmond and Shaw (2009) and Shaw *et al.* (2013) describe in detail the impact of variability and uncertainty in developing business plans and models. This is often caused by the lack of understanding of certain details as data moves through various stages of an orebody's development from defining *in-situ* mineralisation to a producing mine. As noted by David (2007), the fundamental building block of any geometallurgical analysis is the geological domain. An individual geological domain may well have a high degree of variability. By understanding the common factors that define a domain any subsequent test work, data analysis and data interpretation can be focussed within the context of the criteria used to define that domain. As subsequent test work is completed, certain domains or classification criteria may become irrelevant, allowing for simplification that can appropriately guide further phases of test work.

As such, it is at this early stage of project development where geometallurgy can have the greatest impact in supporting decisions to continue developing an *in-situ* ore deposit into a producing mine. Improved material characterisation, combined with the spatial modelling of the variability of critical physical characteristics will form the foundation of a geometallurgical exploration strategy. This provides a much improved basis for operational and mineral processing plant design, during project evaluation studies (Dunham and Vann, 2007). A further critical point regarding the use of sound geometallurgical data in project evaluation studies is raised by David (2013) who stated that a robust negative outcome [for a project evaluation] is as valuable as a robust positive result, because it allows resources to be allocated or redirected as appropriate.

A number of examples have shown where successes can be gained by implementing geometallurgical programmes at the early stages of the deposit's history. Work by Harbort *et al.* (2011) at the Zafranal copper porphyry in Peru led to the creation of a geometallurgical model that could successfully predict the performance of ore during floatation. Work by Leichliter *et al.* (2011) on the La Colosa copper-gold porphyry in Columbia, determined that the variability within the deposit could be grouped into a series of domains based on gold mineralisation.

The Canahuire epithermal gold, copper, silver deposit in southern Peru has undergone two phases of geometallurgical test work, in an attempt to define mineralogical and metallurgical characteristics at an early stage. These characteristics are intended to be used to group ore into domains that will drive a mining and mineral processing strategy. Initial work by Baumgartner *et al.* (2011) identified several challenges such as complexity associated with multiple metalliferous by-products, a high degree of spatial variability in metal grade and the presence of deleterious elements. Through the interpretation of available geological data (lithology, mineralisation, alteration and structure), four geometallurgical domains were defined. Samples were taken from these four domains for metallurgical test work and mineralogical analysis. Further work by Baumgartner *et al.* (2013), attempted to build on the previously defined domains. However, due to the increased understanding gained over a two-year period particularly with respect to the complexity of geology, mineralogical texture and gold deportment, a much more comprehensive and tailored program of conventional metallurgical tests needed to be planned.

The case studies reported thus far have indicated the benefit of starting a geometallurgical program early in the exploration phase of a deposit's history. However they have also highlighted the difficulties posed by limited data at the early stages of mineral deposit investigation. The net result is that more money and time needs to be committed, to gather more data. However as noted by David (2007) one of the biggest barriers to conducting a geometallurgical work program is the fact that in the eyes of the owner and the market, the value of the deposit has already been established within the rules stipulated in an appropriate reporting code.

2.4 Implementing geometallurgy at the production stage

Unfortunately most mines are not set up to operate within a geometallurgical framework. Investigation programmes are often initiated after it has been realised that a mine is failing to meet the designed performance expectations. Typically such programmes are termed "mine to mill optimisation studies". They attempt to link mining processes to milling or metallurgical performance in order to reduce ore dilution, increase mill throughput or reduce recovery. Mine to mill optimisation studies often consist of a number of investigation steps that are identical to a geometallurgical program. In theory greater success could be gained from implementing geometallurgical strategies at the earliest possible stage of mine production as all the pre-requisites are at hand. A large and diverse technical team consisting of mining engineers, metallurgists and geologists are able to interpret larger and more detailed data sets. However to derive meaningful benefit from a mine to mill optimisation study or geometallurgical investigation, you have to permanently change the behaviour of the people and processes they follow which can be very challenging in a complex system such as mining (McCullough *et al.* 2013). A number of case studies in the current literature report on the successes of carrying out geometallurgical or mine to mill optimisation studies. These case studies attempt to quantify the impact of ore on a range of metallurgical characteristics. Through a series of systematic sampling programmes using the currently defined geological domains as a spatial guide, laboratory work would be conducted to ascertain data for those key metallurgical characteristics. These characteristics have been listed by Compan *et al.* (2015) as:

- Mill feed grade;
- Rock hardness;
- Mineral particle size; and,
- Mineralogy.

Work by Jankovic *et al.* (2010) indicated that there is a relationship between ore hardness and the magnetite content of various magnetite ores. Samples composed of higher proportions of magnetite tended to have a “softer” gangue mineral phase, i.e. they were easier to mill. This would indicate that an economic benefit could be gained through reduced energy consumption in the initial phase of grinding prior to concentration. However the test work also indicated that magnetite itself required much greater energy to mill to the required product particle size. In conclusion it was stated that greater sampling and test-work was required to understand the grinding properties of the magnetite concentrate. This raises an important point with regards to geometallurgical programmes. At the producing stage of a mine, geometallurgical test work programmes need to be designed with a view to carrying out repeated phases of test work. Continuous periodic re-evaluation of previous test work conclusions should be made as more data is captured through ongoing daily operations.

A second case study documented by Adams (2004) discusses the impact of optimising the drill and blast design process to improve ore fragmentation. This was achieved through a systematic investigation of a range of variables in the blast pattern design. These variables are listed as:

- Burden and spacing of holes;
- Powder factors; and,
- Stemming height;

It was reported that drill and blast costs at the Mount Rawdon gold mine only accounted for 35% of the total mining cost. A potential increase of 23.9% to drill and blast costs through drilling more optimal patterns could be significantly outweighed by an overall saving of up to 30% in the entire downstream mining process, which accounted for the remaining 65% of costs.

In the case studies of mine to mill optimisation that have been considered in this review it has been noted that rarely do they make mention of *in-situ* geological domains. In the case of the Kansanshi ore deposit the level of geological complexity and variability is well known. As such any geometallurgical or mine to mill optimisation study would have to consider this variability and the associated know geological domains in order to provide any meaningful benefit.

2.5 Chapter summary

Geometallurgy is becoming increasingly more important as mineral deposits are becoming more difficult to find and grades of producing mines are decreasing. The core principle of geometallurgy is to ensure effective integration of technical knowledge along the silos of a mine value chain in order to understand the risks posed to the business. This allows sound decisions to be made that are supported by relevant data. These decisions are taken with a view to improve the overall efficiency or profitability of the business rather than improving an individual silo by negatively impacting other upstream and downstream processes. In order to implement a geometallurgical strategy it is best to start with the correct mind-set at an early stage of the project development, so that the data can be collected which is truly representative of the variability of the *in-situ* ore. This can then translate through to a suitably designed process plant and a practical mine planning and grade control protocol.

Too much data in an early stage is far better than too little as it is always possible to combine domains with similar geological/ metallurgical characteristics together for practicality or simplicity. However the cost of such a large scale data collection program and the time frame required to complete such detailed studies is often prohibitive due to other goals set by the company's larger business strategy. As a result geometallurgical test work often only represents a snap shot of a particular area or point in time regarding the knowledge of the deposit. However if geological complexity and variability is identified early on during the exploration resource definition phase this can translate to further opportunity for optimisation studies when the property becomes a producing mine. Since representivity of geological domains is key to a successful geometallurgical test work program, it may be necessary to plan a geometallurgical strategy to accommodate a regular series of test work programmes through the life of a producing mine. By doing so suitable quantities of data can be periodically gathered to represent each geological domain, leading to an iterative development of geometallurgical understanding.

In Chapter 3 a research methodology is presented that will be used to investigate the integration of data between geology, mine planning and mineral processing strategies currently employed at Kansanshi mine. By developing a detailed understanding of the current *status-quo*, opportunities for developing an integrated geometallurgical strategy can be identified.

3.0 RESEARCH METHODOLOGY

3.1 Introduction

This project covers a broad range of specialist technical areas. It is important to understand inputs, processes and outputs, of each of these areas in order to effectively devise a strategy that seamlessly integrates traditional technical silos.

A systematic method for the gathering of data is presented. This method is an industry standard that has been adapted from the Lean Six Sigma business improvement system. It effectively maps out the integrated nature of a complex system. Each of the main technical silos that form a typical mine value chain was reviewed in the context of the inputs they require for the processes they perform and the outputs they pass on to subsequent technical silos.

Additionally, a range of numerical modelling techniques are presented which are instrumental in quickly interpreting large volumes of data to understand key trends and variables. These models are interactive and show the impact of changing any particular variable, on a whole, or part of, the system being modelled. The research methodology used to gather data for this investigation, was carried out through the following steps:

1. A full mine value chain review;
2. A review of geological data and recent developments in the geological understanding of the ore deposit;
3. A review of the current grade control strategy; and,
4. A review of the metallurgical processes employed at Kansanshi mine.

3.2 Mine value chain review

At the time of undertaking this study, no evidence could be found that documented a clear systematic review of Kansanshi mine's business value chain. This seems to be a recurring theme in the mining industry and was noted by McCullough *et al.* (2013), in that business process flows that connect geometallurgy and mine planning have not been widely mapped, documented and published. It was therefore important that a large scale review of current practices in each of the three main technical areas was completed to fully understand the *status-quo*. Areas that needed to be investigated for this review were:

- The conversion of a mineral resource to a mineral reserves;
- Mine grade control strategy; and
- Mineral processing strategy.

This review brought together a wide array of data from key technical disciplines. It acts as a foundation that identifies areas for improvement which can be incorporated into a geometallurgical strategy, to drive forward value creation in the business.

Due to the complex nature of such an investigation spanning a number of highly specialised technical disciplines the "Supplier, Input, Process, Output, Customer" (SIPOC) method was employed to systematically determine a priority order for each step that contributed to this review. This is a core method utilised in business continuous improvement systems such as Lean Six Sigma. By mapping out the complex interaction of key technical processes at Kansanshi mine in this systematic context, it was possible to clearly identify movement of data between technical silos, bottlenecks along the mining value chain and gaps caused by a lack of integration between various technical disciplines.

A process flow chart in the context of the SIPOC method, showing a high level overview of the integration of data along the mining value chain is presented in Figure 3.1. This flow chart identified an appropriate starting point for the review and assisted in keeping the review within the context of the initial problem statement.

The high level process flow has established that the value chain being reviewed is composed of several defined technical silos. The metallurgical plant sits at the end of this value chain and can therefore be treated as the ultimate "customer" in reference to the SIPOC method. It is therefore important to understand the "voice of the customer" (VOC) i.e. what does the process plant require from the ore product it is receiving to operate efficiently?

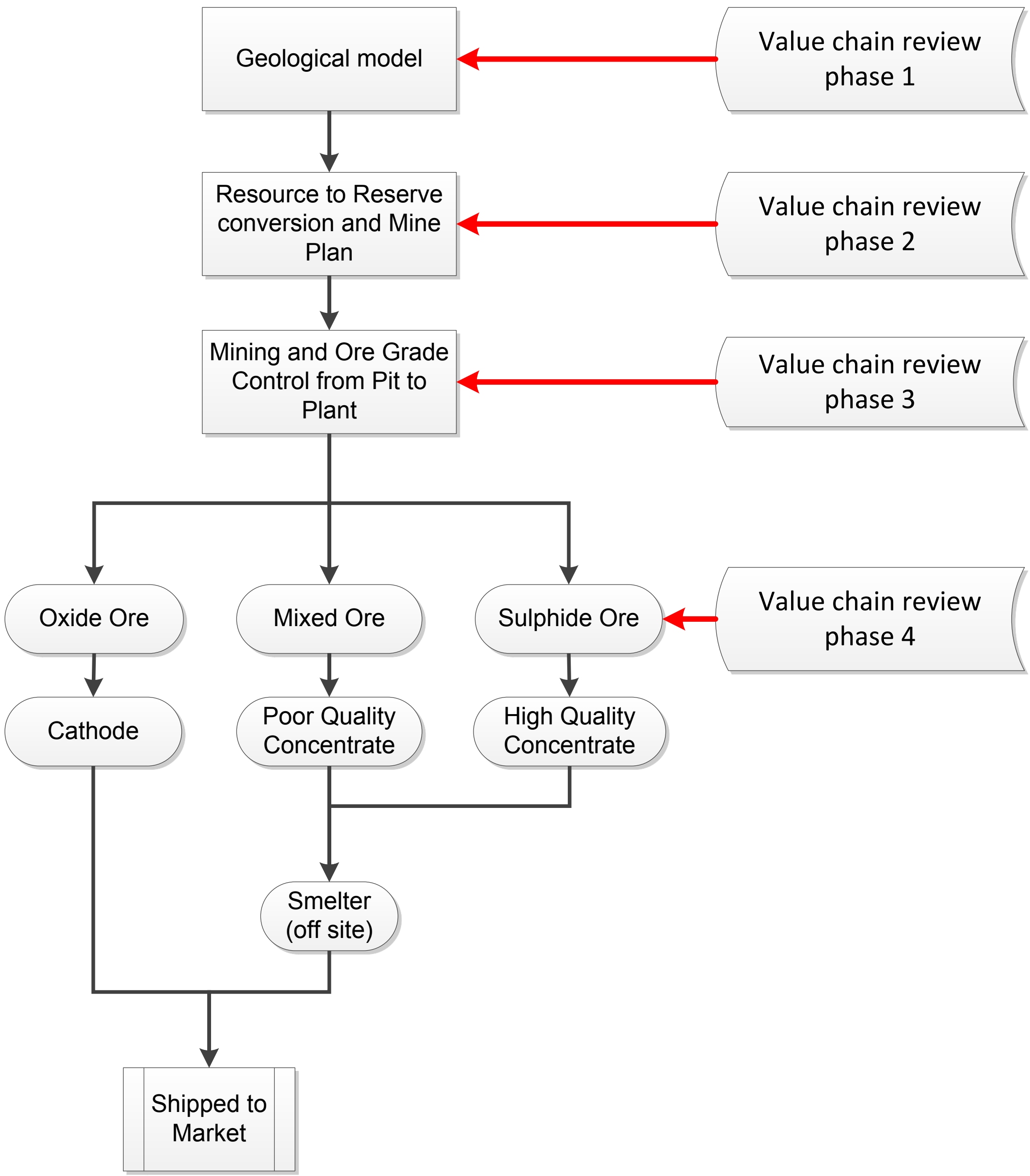


Figure 3.1 High level overview of Kansanshi mine's business value chain.

An understanding of the customer's needs, allowed focus to be placed on understanding the key variables within the deposit that have an impact on the value chain. It is important to continually be reminded that the nature of the ore deposit cannot be changed. Success or failure is determined by the ability of the company to fully understand all technical details and use them advantageously. This is the foundation that forms the basis of any integrated geometallurgical system.

By applying the SIPOC methodology both at a broad scale in terms of the overall process and at a smaller scale within each of the technical areas being investigated, it was possible to develop a set of key questions that, when answered, created a detailed picture of what actually happens in the business and where opportunities for optimisation and improvement could be made.

An indisputable fact in any mining business is that the ore deposit *"is what it is"*. Therefore processes utilised to economically extract a commodity from an ore deposit must be aligned with the nature and characteristics of the *in-situ* ore. At any point in the value chain if a silo is not able to operate efficiently due to the characteristics of the ore, that silo must adjust itself, rather than expect the ore deposit to adjust to suit the silo.

To put this into the context of the work carried out in subsequent chapters of this report, understanding of the mineral resource and the modifying factors applied to create a mineral reserve are first reviewed. From that each subsequent technical silo "downstream" was reviewed. This enabled attention to be focussed on what was needed to improve the output and flow from one silo to the next, along an integrated business value chain.

3.3 Overview of geological understanding of the Kansanshi ore deposit

Significant quantities of geological data have been (and continue to be) collected through mapping, diamond drilling and reverse circulation drilling. This data has been used to define geological structures, mineralised zones and the interaction of physical and chemical processes such as weathering and oxidation with these zones. A three dimensional geological model has been created using assay and logging data to support geo-statistical estimations of copper grade continuity. A cross section showing a portion of this model is presented in Figure 3.2. The model has been colour coded using a set of total copper (TCu) grade ranges. This model is used by the mine planning team to create mineral reserves and mine plans. Grade control assay data is also estimated into this model to feed into short term mine planning and ore production.

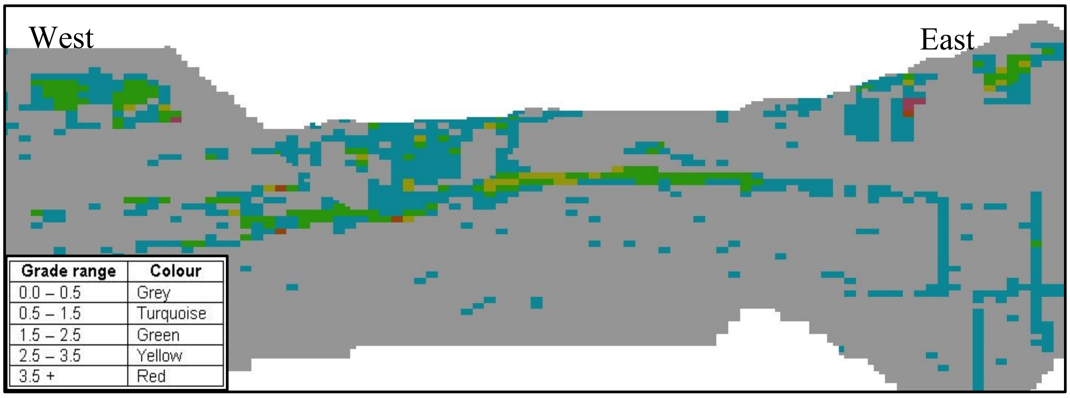


Figure 3.2 Cross section through the Kansanshi mine geological model.

A range of statistical analysis techniques can be employed to further analyse assay data and determine geological or mineralogical domains defined by ranges of oxidation intensity and associated mineralogy. The domains would then need to be further scrutinised in a spatial context to apply them to the geological model. This will allow a more comprehensive prediction of ore characteristics with respect to the available metallurgical processing methods. This analysis was completed using the Snowden Supervisor analysis software.

3.4 Overview of the “Mat_Type” ore classification system

A full review of the current grade control system was completed to determine if ore is being grouped together appropriately, as per the requirements of the metallurgical plant. This review needed to both ask and answer the following questions:

- What are the main copper bearing minerals in each of the ore feeds;
- Are there distinctive mineralogical groupings that can be separated to produce more appropriate “high quality” ore types; and,
- Can factors that negatively impact recovery be modelled into spatial domains in the geological model.

A review of the current grade control process started with an interrogation of the data used to construct the geological resource model that feeds into the mine planning and grade control process. In turn a review was carried out to understand what data flows from the geological resource models into the mine economic reserve model to drive the mine planning and grade control strategies. Further statistical analysis of copper assay data was completed to determine spatial groupings of mineralogy in the deposit. These spatial groupings of mineralogy were compared to the quality control requirements that had been outlined by the process plant.

The Mat_Type system was developed for Kansanshi mine by an external consultancy company. The source code that controls the macro has been locked using the “tool command language byte code” format (.tbc format) and considered to be the intellectual property of the consultant that created it. As a result it was not possible to view the functions performed by the macro directly. However a document that detailed the mathematical formulas used by the macro to calculate the cut-off grade was provided by the consultant to aid in this review. By combining this document with data that is written to the geological model via the macro, it was possible to replicate the entire process using the Microsoft® Excel™ software package. A discussion of key risks and opportunities identified through the review of the current grade control method are presented in Chapter 5.

3.5 Review of the current metallurgical process

There was a need to understand key aspects of each of the three ore processing systems. This part of the data review needed to answer the following questions:

- How is the current metallurgical plant optimised to recover copper?
- What are the key impacts to copper recovery in each circuit? and
- What range of tolerance can the plant effectively operate in?

To achieve this, fact finding meetings were held with senior members of the metallurgical team. These meetings coupled with a continued literature review of mineral processing systems used at Kansanshi mine developed an in-depth knowledge of constraints and limitations on the current ore processing system.

Once key variables that impact plant recovery were determined, they could be placed within the context of the mineralogy and geology of the ore deposit. This would then support the development of an appropriate system to classify a set of ore types that are appropriate for the mineral processing system.

3.6 Chapter summary

Due to the broad nature of this project which attempts to integrate knowledge across a number of technical silos, it was critical to complete a systematic review of all system processes that are completed within each of the technical silos. This review needed to answer key questions about the main value drivers for this company and their integration to the overall business. By using the SIPOC method, it was possible to control the data gathering and data review processes in a systematic manner, within the context of the problem being investigated.

Data in .csv format was gathered as part of the review. This data was then viewed and manipulated in a range of data analysis software packages. Trends and relationships were identified that could be used to better define aspects of the ore deposit. These relationships were then integrated into a strategy that would drive value along the mine value chain.

All the geological data gathered at Kansanshi mine has been incorporated into a three dimensional geological model. Chapter 4 analyses the available geological data to understand *in-situ* ore mineralogy and relationships with weathering and oxidation processes.

4.0 GEOLOGY OF THE KANSANSHI DEPOSIT

4.1 Introduction

After publication of the 2012 Kansanshi mine mineral resource update, a series of statistical analyses were completed to understand the composition of the "transitional ore zone". This zone had been previously defined as the area where mineralisation transitions from "oxide ore" to "sulphide ore" as both types of copper mineralogy are present. It is composed of primary sulphide minerals, secondary sulphide minerals and oxide minerals (Gregory *et al.* 2012). This portion of the deposit contained a significant quantity of copper tonnes but was poorly understood with respect to the geological processes that formed it and the metallurgical processes that could be used to recover the copper. Confusion over the definition of the transitional ore zone was further compounded by the use of a number of inappropriate ore type definitions that had been created as part of the Mat_Type mineral resource to mineral reserve conversion and ore classification system. A clear and robust geological definition of this zone was required as a foundation that could be translated into mine planning, grade control and mineral processing.

4.2 Analysis of assay data to understand oxidation

Copper assay data at Kansanshi mine has three components. The first component is total copper (TCu), which is expressed as a percentage indicating the total amount of copper contained within a single tonne of *in-situ* material. The second component is acid soluble copper (ASCu), which is the proportion of the total copper that is soluble in concentrated sulphuric acid. The third component is acid insoluble copper (AICu), which is the copper that remains after the acid soluble portion is completely recovered. Typically acid insoluble copper represents sulphide minerals and its content within a sample is calculated using Equation 4.1.

$$AICu = TCu - ASCu \qquad \textbf{Equation 4.1}$$

Analysis of diamond drilling assay data that had been collected across the entire Kansanshi deposit by Gray and Hipwell (2013), showed a relationship between two distinct populations of ASCu and Tcu grade. This analysis method was repeated by Beaumont and Ogilvie (2014) on a sample of close spaced reverse circulation (RC) drilling assay data from the active mining area. Total copper grade was plotted against the corresponding acid soluble copper grade to understand the relationship between the two. All data was filtered above the 0.19% TCu life of mine cut-off grade and below the 12.00% TCu estimation top cut grade using the Snowden© Supervisor™ statistical analysis software. Scatter plots created from the analysis are presented in Figure 4.1 and Figure 4.2 respectively. The scatter plots show two clear populations of data, indicating two dominant suites of mineralogy with a set of defined characteristics. These characteristics can be described as:

i. **ASCu consistently below 0.75%:** Increasing TCu grade with consistently low corresponding ASCu grade. TCu tightly constrained with a gentle positive correlation i.e. ASCu increases marginally (from ~0.15 to ~0.5) with increasing TCu. This indicates a suite of minerals dominated by copper bearing sulphide minerals.

ii. **ASCu increases almost proportionally with increasing TCu:** High ASCu grades with correspondingly high TCu grades. ASCu is always slightly less than TCu. This indicates a suite of minerals dominated by copper bearing oxide minerals.

From this analysis it can be determined that economic mineralisation has been modified through a single chemical process, oxidation, to create two mineralised zones; a copper sulphide zone and a copper oxide zone. A third transitional domain that has formed as a result of different geological processes is not supported by the assay data.

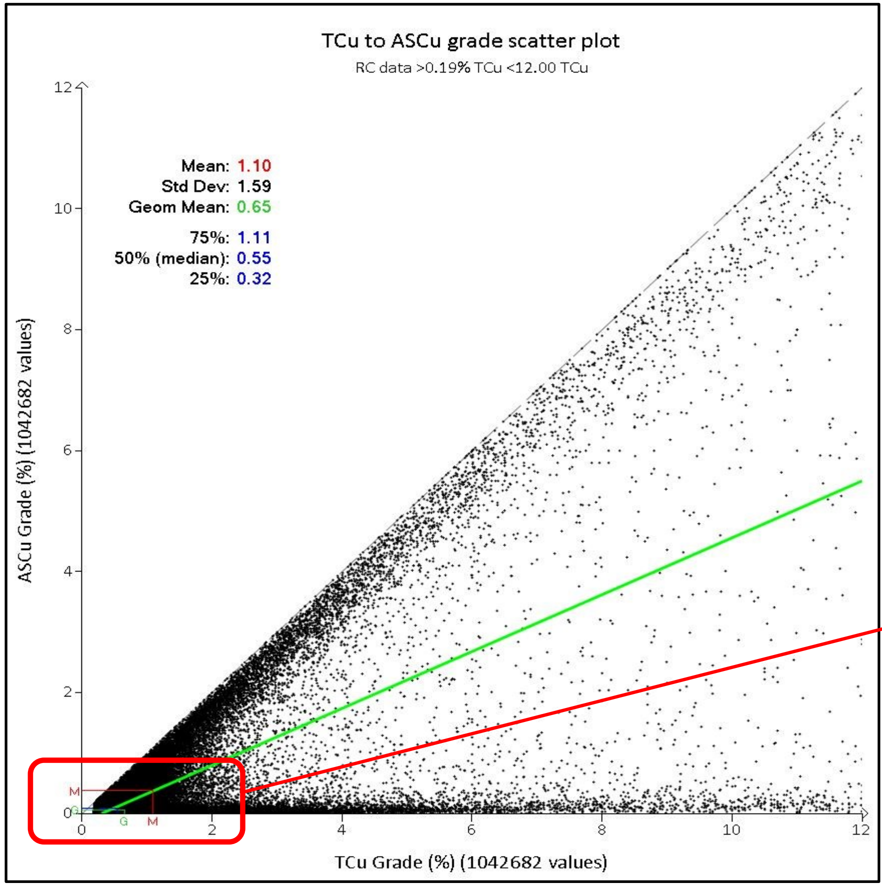


Figure 4.1 Scatter plot of assay data with ASCu grade plotted as a function of TCu grade. (Beaumont and Ogilvie 2014)

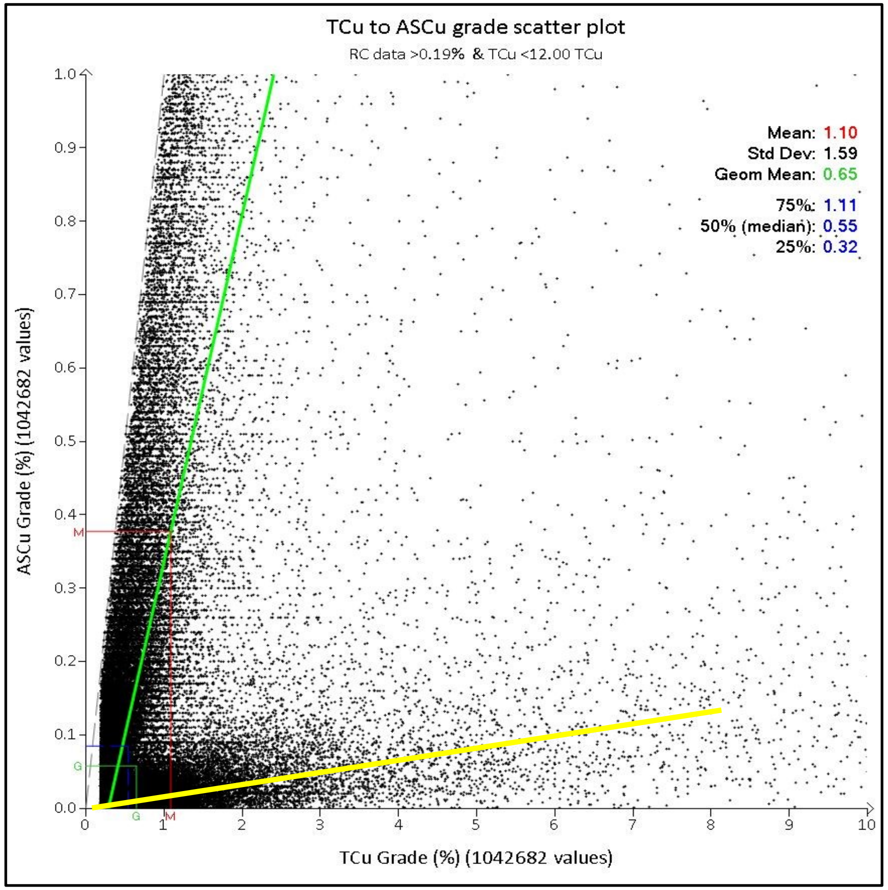


Figure 4.2 Data filtered to <1.00% ASCu to de-cluster the mass of data near the origin. (Beaumont and Ogilvie 2014)

It is important to note that the two zones are very broad and highly variable. The scatter of data points between the two trends identified in Figure 4.1 and Figure 4.2 indicates that there is no clear boundary between the sulphide and oxide domains. The Kansanshi mine ore deposit has a high degree of grade variability over very short ranges (+/- 20m). This variability is both lateral and vertical as mineralisation is structurally controlled. Structures also play a key role in defining weathering and oxidation profiles. The overprinting and modification of ore mineralogy by ranges of oxidation intensity adds a further variable dimension to the data being analysed. Therefore the domains that have been identified can only be treated as ranges which capture a majority of the mineralisation being defined. Further analysis of data presented in subsequent sections of this report will refer back to this point, as attempts are made to define ranges that contain populations of data with similar trends or characteristic, rather than defined hard boundaries.

4.3 Oxidation domains

Oxidation is a penetrative process that starts at the mineral surface and works inward along lines of weakness such as a mineral cleavage. The relationship between weathering and oxidation is not direct. Weathering can be considered as a physical process that breaks down *in-situ* rock to form a range of clay minerals. On the other hand, oxidation is a chemical process; acting on a mineral surface to change its chemical composition to a more stable form given the modification of the local environment by weathering processes. With specific reference to copper in the context of the Kansanshi mine ore deposit, a range of secondary copper sulphide minerals form under weak to moderate oxidation levels. As the intensity of oxidation increases, the secondary copper sulphide mineral will complete the oxidation process to a primary oxide mineral. The presence of acid soluble copper indicates the presence of primary oxide minerals. If the amount of acid soluble copper increases with respect to the amount of total copper then the quantity of primary oxide mineralogy is increasing while the quantity of primary and secondary sulphide mineralogy is decreasing. This observation leads to the determination of an oxidation ratio, presented in Equation 4.2.

$$\boldsymbol{Oxidation\ Ratio = \frac{ASCu}{TCu}} \qquad \textbf{Equation 4.2}$$

By plotting a log histogram of assay data as a function of oxidation ratio, it is possible to determine ranges that signify the degree of oxidation intensity that has affected a mineralised domain within the deposit. Analysis of assay data presented in Figure 4.3 and Figure 4.4 respectively, shows three distinct populations of data controlled by ranges of oxidation ratio. These populations can be described as:

1. A narrow range of oxidation ratio (< 0.1). This can be inferred as primary sulphide mineralisation and can be identified by the yellow circle;
2. A second population with a narrower range of oxidation ratio is present at the upper end of the scale (0.8 to 1.0), which defines oxide mineralisation and can be identified by a green circle; and,
3. A third population of data ranging from 0.1 to 0.8 identifying the start of the oxidation process where primary copper sulphide minerals have started to oxidise to secondary copper sulphide minerals and can be identified by a red circle.

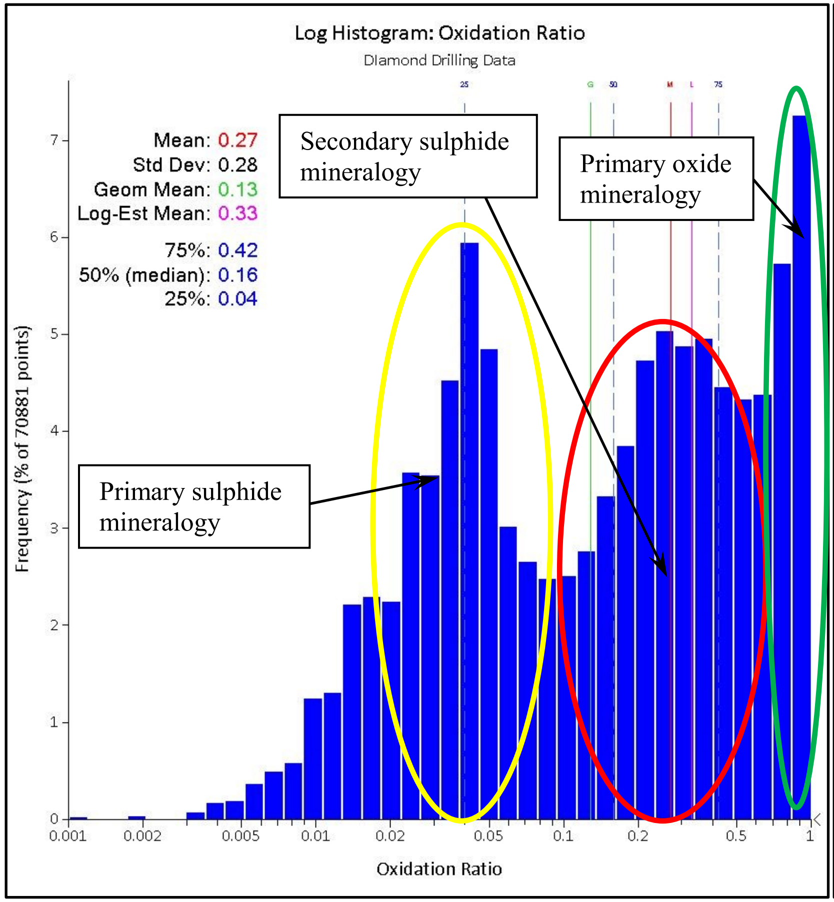


Figure 4.3 Log histogram of diamond drilling assay data, with corresponding oxidation ratio. (Adapted from Gray and Hipwell 2013)

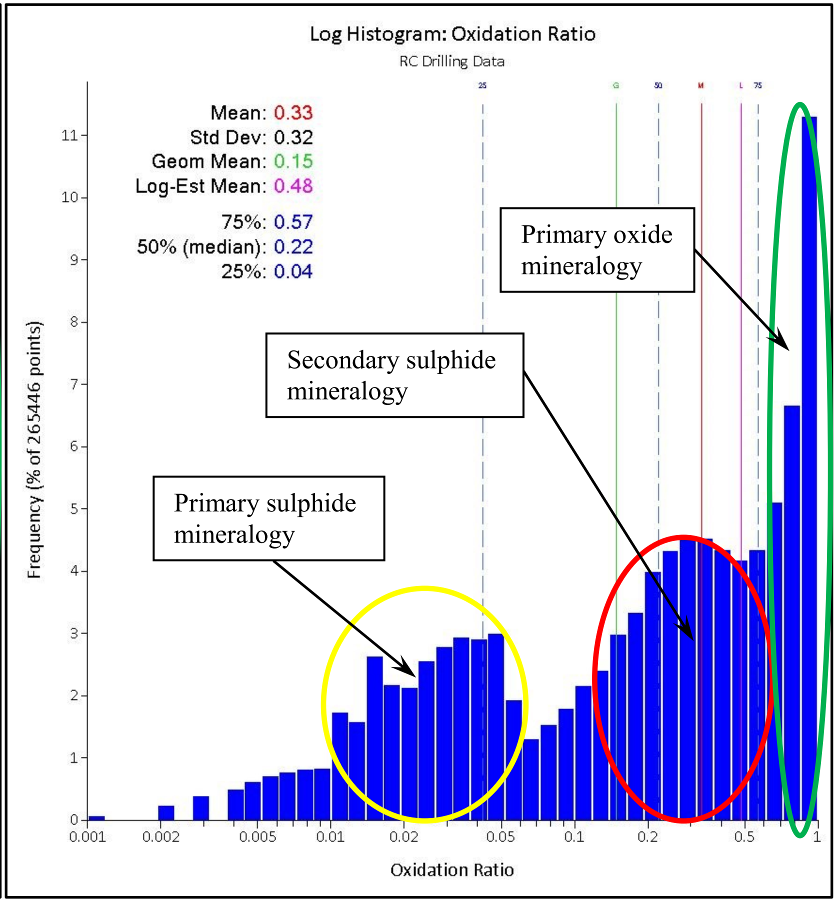


Figure 4.4 Log histogram of R.C. drilling assay data, with corresponding oxidation ratio. (Adapted from Beaumont and Ogilvie 2014)

By plotting assay data as a function of oxidation ratio, it is possible to determine ranges that hold the dominant mineral suite i.e. primary sulphide mineralogy, secondary sulphide mineralogy or primary oxide mineralogy. By incorporating the analysis and understanding of oxidation ranges into the grade control strategy it is possible to determine the dominant mineralogy of a particular spatial domain. In Chapter 8 reference is made to the utilisation of this understanding of oxidation and mineralogy, when determining a suitable mineral processing strategy for a particular suite of economic minerals.

Two cross sections through the geological block for the main pit deposit are presented in Figure 4.5 and Figure 4.6. The sections have both been taken in an east-west direction along the 12,000m northing line, looking northwards. Figure 4.5 shows the geological block model colour coded by weathering profile based on geological logging of diamond drill core and R.C. drilling chips. The oxidation ratio calculation presented in Equation 4.2, has been applied to the estimated TCu and ASCu grades in the geological block model and is presented as a cross section in Figure 4.6.

A comparison between Figure 4.5 and Figure 4.6 shows a poor spatial correlation between weathering and oxidation domains. Larger weathering zones do have more intense oxidation. However saprolite and sap-rock domains are spatially connected in both lateral and vertical extent. Broad weathering domains at deeper elevations are caused by the interaction of surficial weathering processes on large fault structures. Oxidation is tightly controlled in narrow flat lying structures on the edges of the deposit while broader lenses are present in the central areas around the main fault zones. Oxidation has also occurred at deeper elevations of the deposit, in areas that have been unaffected by weathering. Furthermore throughout the deposit, a significant proportion of weak to moderate oxidation occurs in discrete patches often with no spatial connection to other areas of oxidation or weathering. Weathering and oxidation processes are not directly related to each other and have affected different areas of the deposit in different ways.

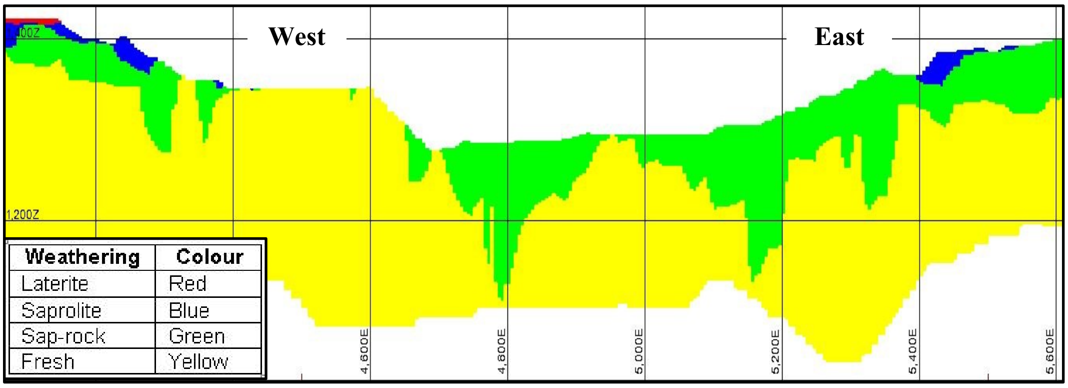


Figure 4.5 Cross section through the geological model, colour coded by logged weathering characteristics.

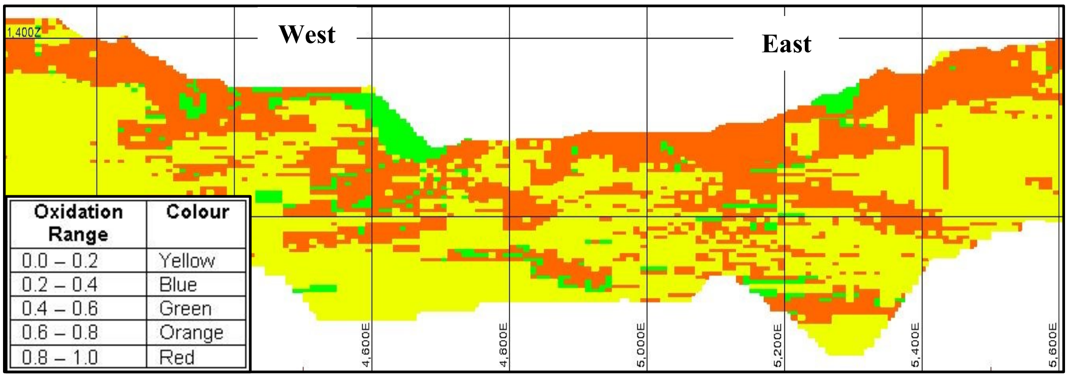


Figure 4.6 Cross section through the geological model, colour coded by oxidation ratio.

4.4 Geological mapping

A spatial analysis of raw assay data was conducted using Supervisor™ to understand where the *in-situ* boundaries of oxidation were located. Due to the quantity of data available the analysis was conducted on diamond drilling data and R.C. drilling data separately. The analysis of diamond drilling data is presented in Figure 4.7. Ratios have been grouped as per the ranges identified through the statistical analysis presented in Section 4.3.

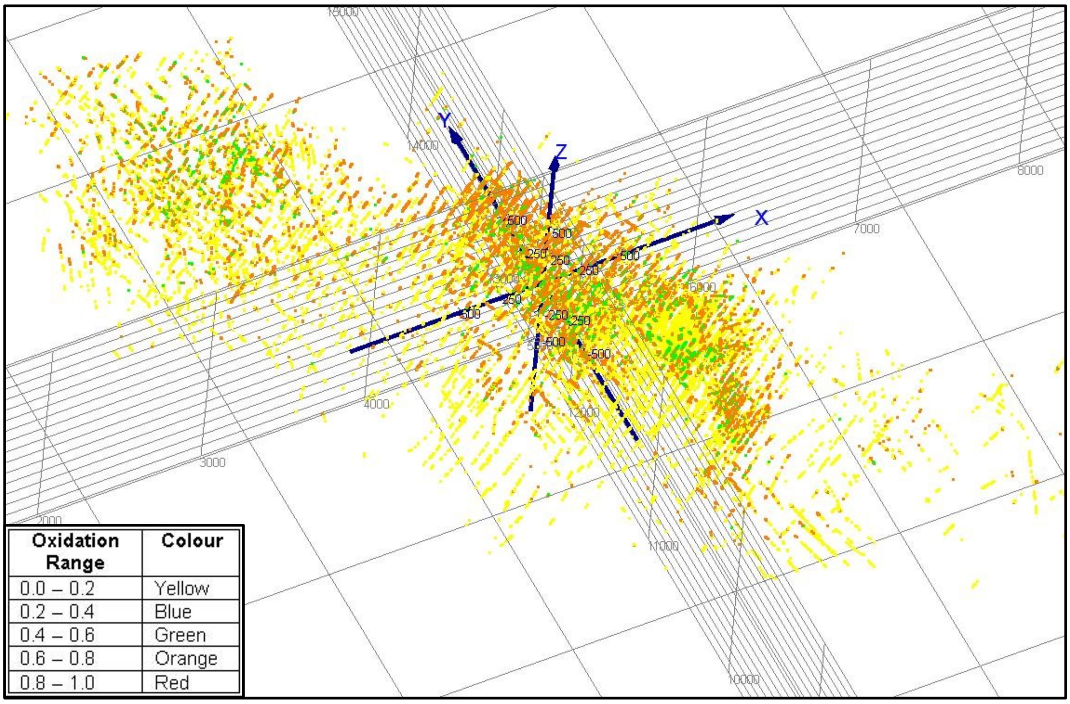


Figure 4.7 Orthographic view of diamond drilling data across the Kansanshi ore deposit.

Through this method of spatial data analysis, locations in the current mining area that had oxidation ratios within the three oxidation ranges were identified. Weathering profiles and dominant structures such as veins and faults were overlaid onto the spatial analysis using Surpac™. An idealised cross section through the deposit showing key mineralised structures and their interaction with the weathering profile has been presented in Figure 4.8. This section shows clearly that weathering plunges down the vertical fault structures exploiting weaknesses that cross cut through the stratigraphy. By bringing together multiple sources of data in a spatial context it was possible to identify areas that could be targeted for an in-pit mapping campaign. The purpose of this mapping was to understand the interaction of weathering and oxidation in mineralised structures.

Figure 4.9 shows the geological model for the Main Pit at Kansanshi mine. The model has been depleted up to the end of February 2015 using a survey digital terrain model (.dtm) file in Surpac™. The weathering domains have been colour coded to give a spatial reference of data to in pit locations.

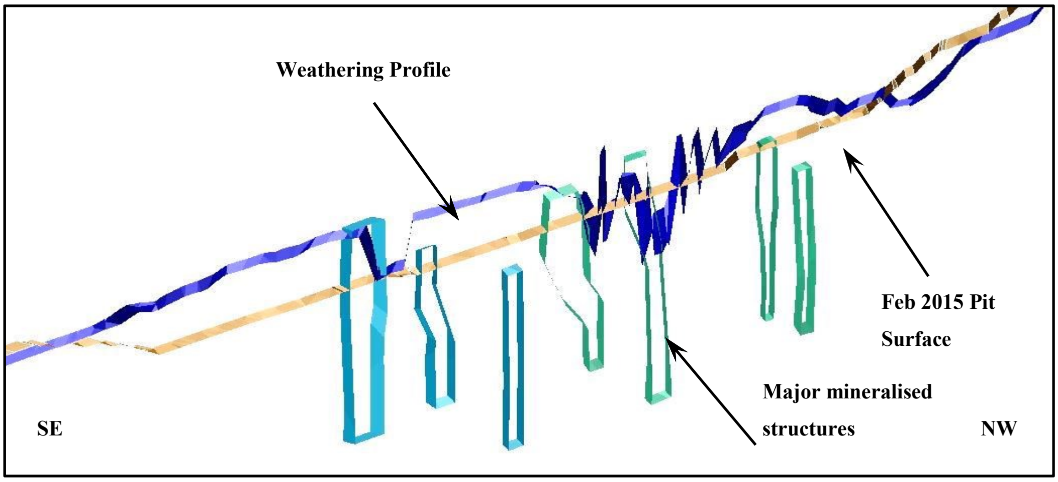


Figure 4.8 An oblique cross section through the deposit showing the interaction of weathering domains with major geological structures.

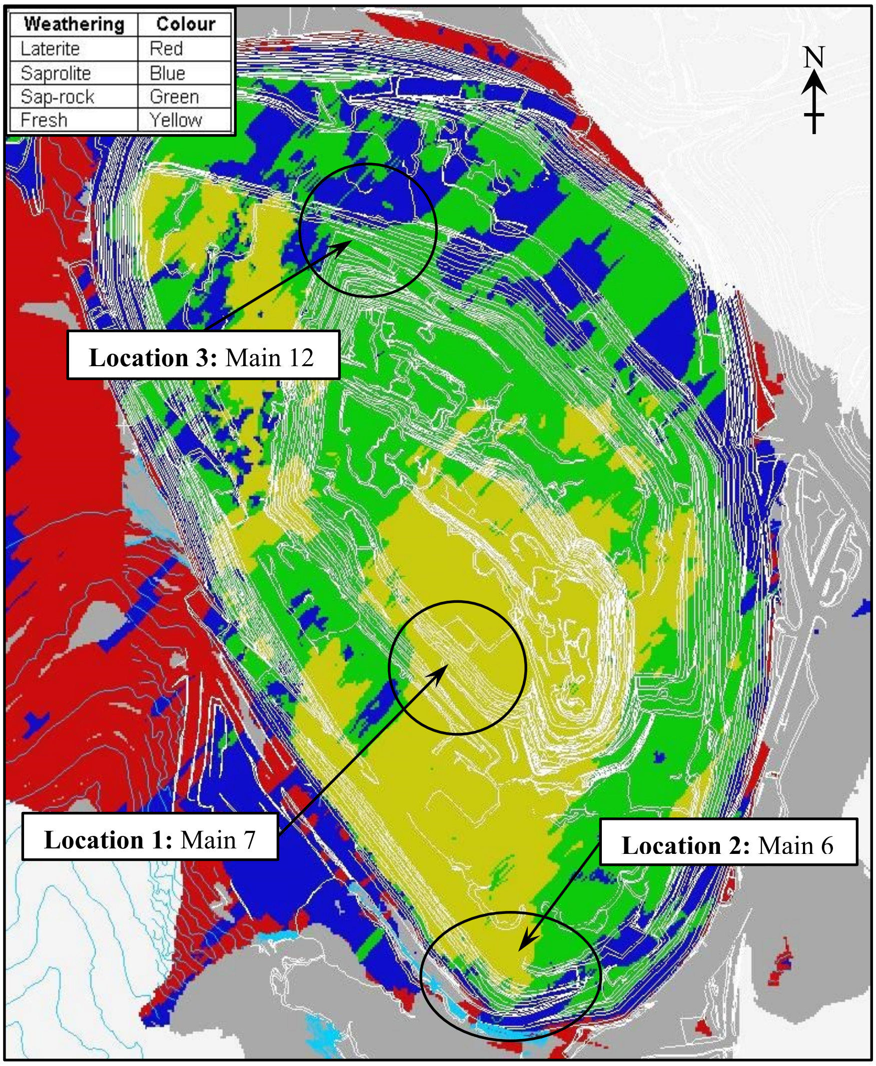


Figure 4.9 Plan view of Main Pit, indicating three key mapping locations.

In-pit mapping is ongoing as a matter of routine as new faces are exposed through mining. Data gathered from three key mapping locations in the Main Pit is presented in Sections 4.4.1, 4.4.2 and 4.4.3. These locations give a detailed explanation of the typical phenomena that continue to be observed throughout the pit as a result of the ongoing mapping programme.

4.4.1 *Key geological features observed in Main 7*

Main 7 is an active mining cut-back currently at an elevation of 1295m which is 120m below the original surface topographic survey. This area is currently the main source of sulphide ore feed to the Kansanshi metallurgical plant. A typical mineralised structure that can be found in this area is presented in Figure 4.10. Note that the host lithology is un-weathered and mineralised structures contain fresh chalcopyrite with quartz and calcite gangue minerals. There is no evidence of oxidation on the copper bearing mineral surfaces.

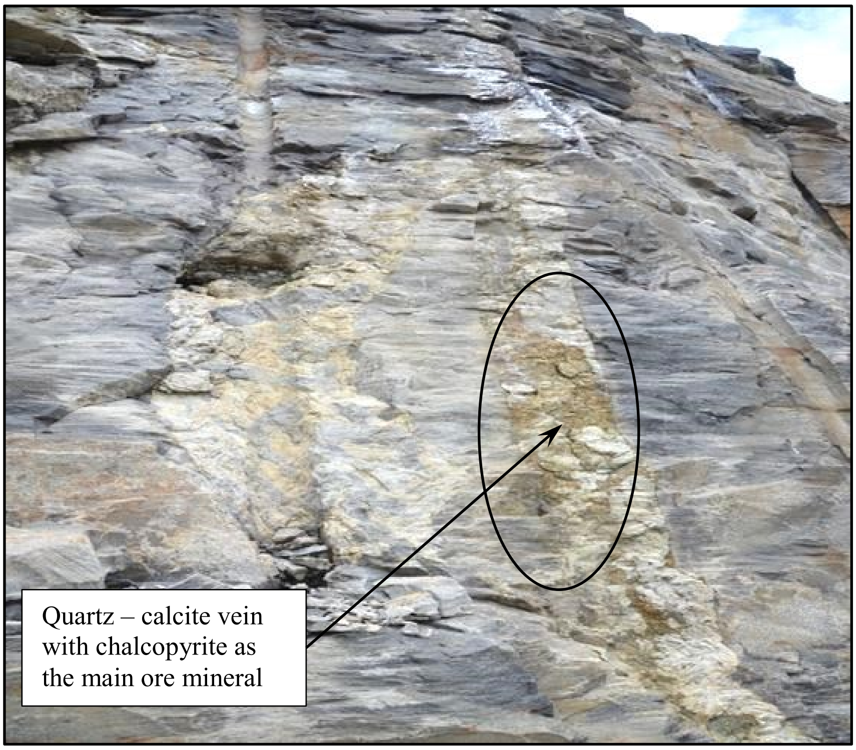


Figure 4.10 A typical mineralised vein structure in Main 7.

4.4.2 *Key geological features observed in Main 6*

Main 6 is an active mining cut-back, currently at an elevation of 1375m. The main mineralised structures that strike through this cutback are presented in Figure 4.11. At the 1375m elevation, below the weathering surface, the structure is moderately oxidised, yet the host lithology, a phyllite, is un-weathered. Some characteristic brown/red coloured iron oxide minerals can also be seen.

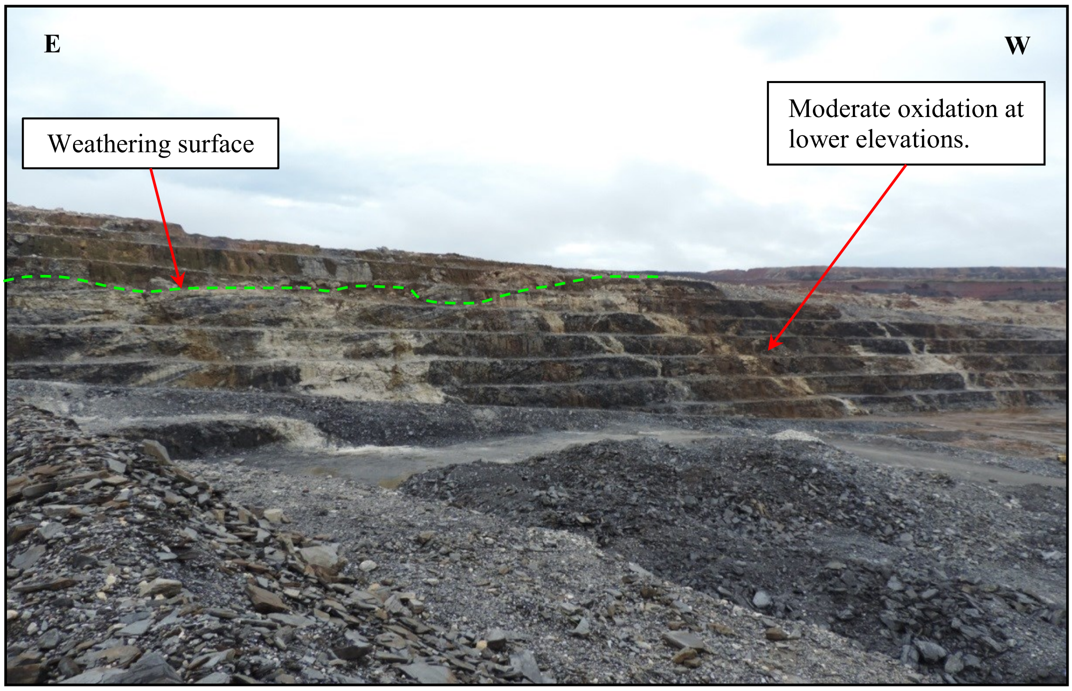


Figure 4.11 Mineralised structures and weathering along the southern wall of Main 6.

A closer examination of a typical structure is presented in Figure 4.12. Weak oxidation in the form of tarnish can be observed on the surface of the primary sulphide mineral chalcopyrite. In localised areas, oxidation is more intense leading to the formation of chalcocite a secondary sulphide mineral. However despite this localised more intense oxidation the lithology is un-weathered.

Since these structures are vertically extensive they are cut across by the weathering zones. This allows ground water to percolate downwards into the structure and react with the sulphide mineralisation, creating localised oxidation of the ore mineralogy well below the weathering surface. It can also be noted that in structures with calcite as the dominant gangue mineral the depth of oxidation is much greater than in structures with quartz as the dominant gangue mineral. As water interacts with the ore and gangue sulphide mineralogy the oxidation process occurs.

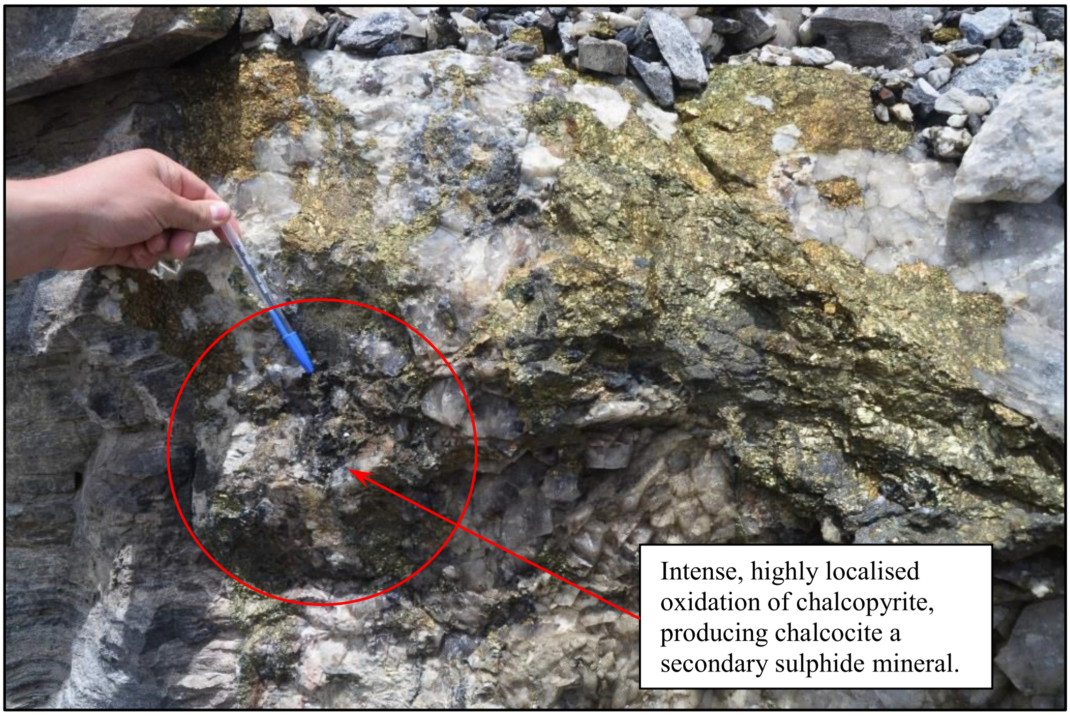


Figure 4.12 Moderately oxidised portion of a large vein structure containing chalcopyrite and chalcocite.

The breakdown of chalcopyrite to form secondary copper sulphides and primary copper oxides creates a range of other products including sulphuric acid and iron oxides. Work by Chavez (2000) and Sillitoe (2005) has documented a series of oxidation reactions for copper bearing minerals. The reaction series demonstrate that a range of copper minerals are created concurrently as the oxidation process intensifies. The reactions that pertain to the Kansanshi deposit have been presented in Equations 4.3, 4.4, 4.5 and 4.6 respectively.

$$CuFeS_2 + 3Cu^{2+} \rightarrow 2Cu_2S + Fe^{2+}$$ **Equation 4.3**

chalcopyrite + copper cations ➔ *chalcocite + iron cations*

$$2Cu^{2+} + CO_3^{2-} + 2OH^- \rightarrow Cu_2(OH)_2CO_3$$ **Equation 4.4**

copper cations + carbonate anions + hydroxide anions ➔ malachite

$$2CuFeS_2 + 4H_2O + 10\ O_2 \rightarrow Cu_2O + 4\ H_2SO_4 + Fe_2O_3$$ **Equation 4.5**

chalcopyrite + water + oxygen ➔ cuprite + sulphuric acid + iron(III) oxide

$$4CuFeS_2 + 17O_2 + 10H_2O \rightarrow 4Fe(OH)_3 + 4Cu^{2+} + 8SO_4^{2-} + 8H^+$$ **Equation 4.6**

chalcopyrite + oxygen + water ➔ Iron(III) hydroxide + copper cations + sulphate anions + hydrogen cations

4.4.3 *Key geological features observed in Main 12*

Main 12 is an active mining cut-back currently at an elevation of 1385m. This area is dominated by a large fault structure termed the "4800 Zone" that crosses the central portion of the mining area. To the east of the cut-back along the pit wall several gabbro intrusions can be found. These gabbros have some copper mineralisation around the edges. As a result of the interaction of the "4800 zone" with the surface intense weathering to deep elevations has occurred along this fault. Figure 4.13 shows a large area of primary oxide mineralisation dominated by chrysocolla. Note that the area is intensely weathered. An albite alteration halo (visible in the foreground of Figure 4.13) is present along the contact of the gabbro, phyllite and mineralised structure. The quartz content of the gabbro is a significant factor in the type of primary oxide minerals formed. Due to the abundance of silica and lack of carbonate in this area, chrysocolla, a copper silicate mineral, forms as the dominant primary oxide mineral.

Away from the gabbro intrusion on the same elevation are areas where the upper marble is in contact with mineralised structures. An example is presented in Figure 4.14. These structures have also been weathered and oxidised to a similar intensity. Malachite has formed as a result of the lack of silica and abundance of carbonate from weathering processes acting on the upper marble lithology.

Figure 4.13 Chrysocolla mineralisation in an intensely weathered fault structure that cross cuts the upper mixed clastics.

Figure 4.14 Malachite mineralisation in an intensely weathered fault structure that cross cuts the upper marble.

4.4.4 *Summary of key geological mapping data*

Much improvement in the understanding of weathering and oxidation across the deposit has been gained from this geological mapping campaign. The depth and intensity of weathering across the deposit is highly variable. Where structures such as veins and faults are intersected by the weathering profile, weathering exploits the zone of weakness and penetrates down to deeper elevations while remaining constrained to the boundaries of the fault.

Oxidation shows a similar trend and where areas are heavily weathered the ore mineralogy is also intensely oxidised. However in many areas the host stratigraphy has not been weathered at all yet there is still evidence of oxidation in the form of secondary sulphide minerals and in some cases primary oxide minerals also.

Although there is a general correlation between intensity of weathering and intensity of oxidation, other factors play a key role. The size of structures and their interaction with certain host stratigraphies also play a part in the intensity of weathering and oxidation. This in turn determines the type of mineralisation that forms.

4.5 Ore mineralogy

Geological grab samples were taken from each of the mapped domains presented in Section 4.4. These samples were then cut into polished blocks for detailed mineralogical analysis at the Kansanshi mine metallurgical laboratory. Samples were analysed using reflected light microscopy and a Zeiss "Mineralogic" scanning electron microscope (SEM). A series of images taken using the Zeiss SEM are presented in order of increasing weathering and oxidation intensity. These images show the key economic minerals that are developed during the oxidation process.

4.5.1 *Analysis of samples gathered in Main 7*

A typical example of Kansanshi mine's primary sulphide mineralogy in the form of chalcopyrite was gathered from Main 7. An SEM image of a polished block created from this sample is presented in Figure 4.15. Note the intergrowth of euhedral pyrite (Py) crystals, a gangue sulphide mineral. The chalcopyrite (Cpy) mineral is heavily fractured and contains a great deal of internal void space. These fractures could potentially be exploited by oxidation processes.

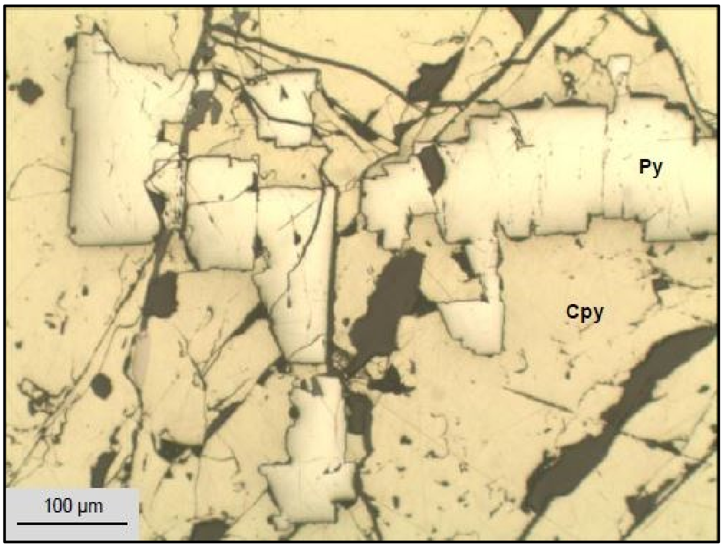


Figure 4.15 Un-oxidised chalcopyrite (Cpy) containing euhedral crystals of pyrite (Py).

4.5.2 *Analysis of samples gathered in Main 6*

Samples gathered from Main 6 displayed weak to moderate oxidation across the surface of the hand specimen. Analysis of polished blocks using the Zeiss Mineralogic SEM showed several examples of secondary sulphide minerals forming due to more intense localised oxidation. Images of this sample taken using the SEM are presented in Figure 4.16 and Figure 4.17, respectively. Figure 4.16 shows an original chalcopyrite crystal being replaced by a halo of digenite (Dig) and covellite (Cov). Figure 4.17 shows chalcopyrite (Cpy) being surrounded and replaced by a halo of chalcocite (Chc). This halo of secondary sulphide mineralogy forming around the original primary sulphide mineral is often termed as “rimming”. Rimming is defined as the growth of a new mineral around the edge of an existing mineral creating a halo. Rimming represents the first stage of the formation of a suite of secondary sulphide minerals.

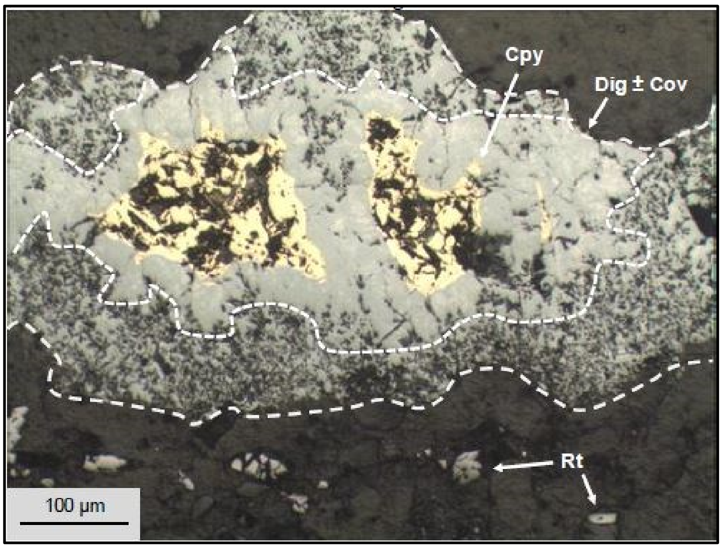


Figure 4.16 Highly fractured chalcopyrite (Cpy) bleb surrounded by a halo of digenite (Dig) and covellite (Cov).

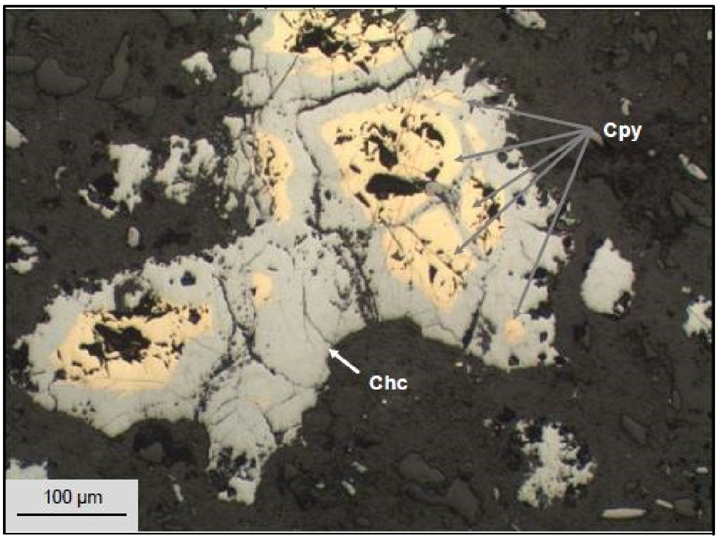


Figure 4.17 Heavily fractured chalcopyrite (Cpy) bleb surrounded by a halo of chalcocite (Chc).

An example of massive chalcocite (Chc) representing the end of the secondary sulphide oxidation process is presented in Figure 4.18. Note that cuprite is also starting to form along cleavage planes and fractures, indicating that localised variation in physical characteristics of the mineral leads to variations in the intensity of the oxidation process.

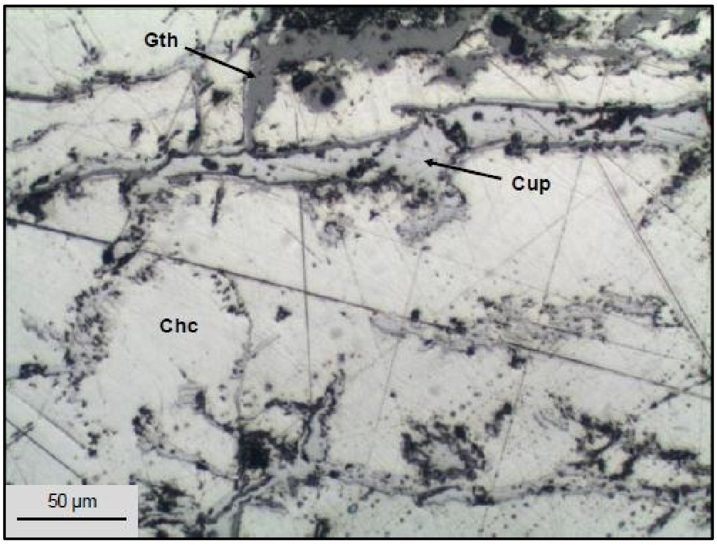


Figure 4.18 Massive chalcocite (Chc) crystal, with small intergrowths of cuprite (Cup) and goethite (Gth) along an internal fracture.

4.5.3 *Analysis of samples gathered in Main 12*

Hand specimens gathered from Main 12 represent the most intensely oxidised end of the range and as such are dominantly composed of primary oxide minerals. An example of malachite laths forming as a replacement of cuprite is presented in Figure 4.19. This sample was taken from an exposure that showed the interaction of a large mineralised fault with the upper marble.

An example of chrysocolla mineralisation is presented in Figure 4.20. This sample was taken from a location 22m higher up the pit wall from the sample presented in Figure 4.19. Above the upper marble sits the upper mixed clastics a stratigraphic sequence composed of siliceous metamorphosed mudstones and siltstones.

Typically the mixed clastic sequences are dominated by quartz bearing minerals rather than carbonate bearing minerals. During the oxidation process the breakdown of sulphide minerals creates unstable copper ions that need to bond to a cation. Through the physical process of weathering quartz is produced and copper readily bonds to it producing chrysocolla rather than malachite.

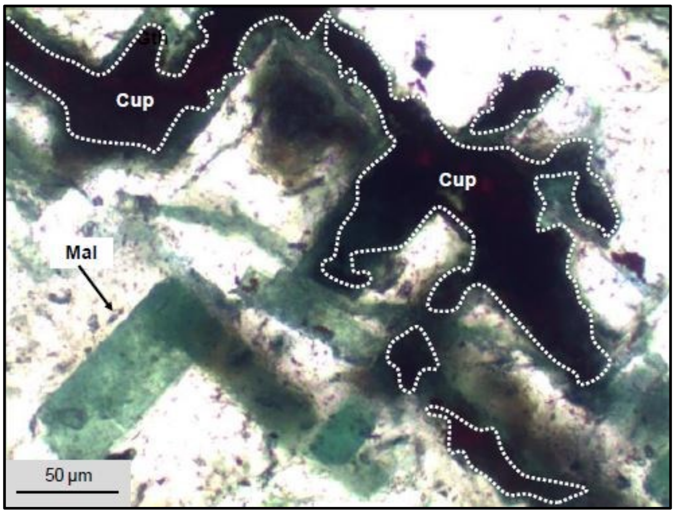


Figure 4.19 Euhedral malachite (Mal) laths forming as a result of the replacement of cuprite (Cup).

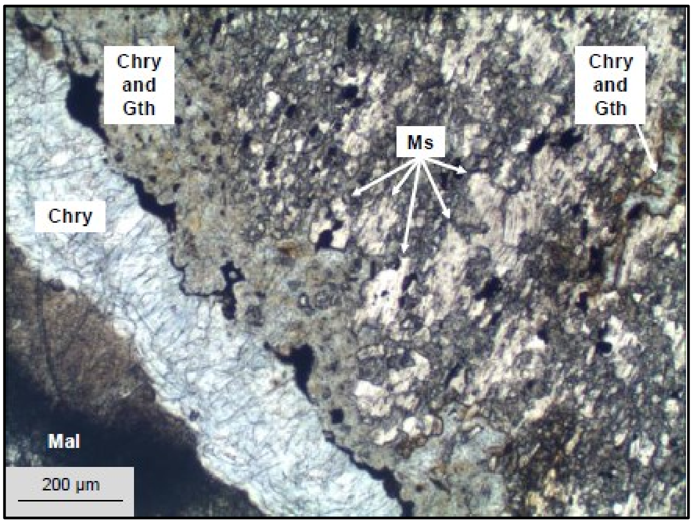


Figure 4.20 Chrysocolla (Chry) forming as a skin over a mat of chrysocolla, goethite (Gth) and muscovite (Ms).

4.5.4 ***Summary of the mineralogical analysis***

The results of the mineralogical analysis were threefold. Firstly, an understanding of the paragenesis from the primary sulphide mineral chalcopyrite into a suite secondary copper sulphide minerals and primary copper oxide minerals was determined. Secondly, this paragenesis can be linked spatially to defined oxidation domains within the deposit. Thirdly, features such as “rimming”, have been identified at the microscopic scale particularly with respect to secondary copper sulphide minerals. In almost all cases of primary to secondary copper sulphide mineral conversion, a relict amount of the original chalcopyrite mineral can be found in the centre of a newly formed secondary copper sulphide mineral.

Rimming may have a significant impact on the prediction of metallurgical performance. The floatation plants are optimised to recover copper sulphide particles at a certain size range with defined surface characteristics that are amenable to the collector reagents. These mineral surfaces become modified during grinding. The resulting particle surface area is composed of a number of different copper sulphide minerals. Control of this surface modification could be an important factor in defining a suitable metallurgical process to recover the copper from this ore type. This hypothesis will be covered in greater detail in Chapter 8.

Coupled with the development of a mineral paragenesis, the mineralogical work also highlighted the degree of variability in the deposit thus supporting the statistical analysis presented in Sections 4.2 and 4.3. Features such as the intergrowth of cuprite along a fracture plane within chalcocite indicate that the transition from a secondary sulphide mineral suite to a primary oxide mineral suite does have an element of gradation. This evidence further supports the large ranges observed in the statistical analysis of oxidation data.

By combining geological mapping with detailed mineralogical analysis, and correlating it spatially to the oxidation domains presented in Section 4.3, a range of data has been built up to support the principle that economic mineralogy in the deposit is directly linked to the intensity of oxidation. Oxidation domains should therefore play a key role in ore type definition. Additionally it is possible to subdivide the two broad zones of mineralisation (sulphide mineralisation and oxide mineralisation) originally identified in the statistical analysis. These additional sub-domains are presented in Table 4.1 and represent natural groupings of similar mineralogy. These groupings are formed by the complex interaction of structure, weathering and oxidation processes on the deposited mineralisation.

Table 4.1 Sub-domains identified by mineralisation style within the two main oxidation zones at Kansanshi mine.

Zone	Sub-Domain 1	Sub-Domain 2
Sulphide (ratio 0.0 to 0.8)	Primary sulphide mineralisation with an oxidation ratio <0.1.	Secondary sulphide mineralisation with an oxidation ratio >0.1 and <0.8
Oxide (ratio 0.8 to 1.0)	Malachite dominant mineralisation linked to marble or carbonate bearing lithologies.	Chrysocolla dominant mineralisation linked to silica dominated clastic lithologies.

4.6 Application of mineralogical understanding to ore classification.

Optimal recovery of copper in each processing circuit could be achieved by ensuring mineralogy is consistently grouped together in the ore product. Analysis of copper assay grades with respect to oxidation ratio has identified natural domains that can be used to infer certain suites of minerals. Further mineralogical analysis of geological samples from within these domains presented in Section 4.5 has shown that typical suites of mineralogy are associated with each oxidation ratio grouping. Each of these domains has been defined spatially within the deposit through a combination of geological mapping (presented in Section 4.4), geological logging and sample assay data. All this data is stored in a single three dimensional geological block model which is continually updated as more data is captured.

The transition zone defined Gregory *et al.* (2012) was applied to the geological model using a combination of a grade cut-off of >0.10% ASCu and weathering domains (sap-rock and fresh). However through the work presented in this chapter it has been shown that the transition zone has not formed as a result of separate geological processes. Furthermore it can be more accurately defined by using a range of oxidation ratio from 0.1 to 0.8 which represents a dominant suite of secondary sulphide minerals.

A discussion on the integration of geological and mineralogical data into the mineral processing methodology is presented in Chapter 8. Developing a grade control system that groups similar styles of ore mineralisation together appropriately during the mining process is crucial. A more robust geological definition of the oxidation zones has been presented in this chapter. These definitions will act as a foundation to the mineral resource to mineral reserve conversion process which will determine a suitable mining strategy and mineral processing system for the ore contained in each zone.

4.7 Chapter summary

Through a statistical review of assay data, greater understanding of oxidation processes affecting *in-situ* ore was gained. This understanding was then developed into a spatial context using the Snowden "Supervisor" software, to demarcate oxidation domains within the deposit that can be coded into a geological model. Mapping and sampling of these domains coupled with analysis using reflected light microscopy and scanning electron microscopy techniques revealed a detailed mineral paragenesis caused by oxidation processes. This supports the statistical analysis carried out. Two broad mineralisation zones have been defined, each with two sub-domains based on specific geological and mineralogical characteristics. These sub-domains have been defined in Table 4.1 and can be used to group mineralisation into broad categories for mineral processing.

It is clear from this work that a robust understanding of the ore deposit mineralogy must be built into grade control, mine planning and mineral processing strategies. Continuing with the development and integration of geological knowledge into a grade control and mineral processing strategy, an evaluation of gangue acid consumption (GAC) and its relationship to *in-situ* geology and the oxide metallurgical process is presented in Chapter 5.

5.0 GANGUE ACID CONSUMPTION

5.1 Introduction

Gangue Acid Consumption (GAC) is a measure derived by laboratory test work. The test work is routinely carried out on a selection of geological samples that are sent to the Kansanshi mine laboratory for copper and gold assay analysis. The test was introduced in the 2007 to 2008 period at the request of the metallurgical department due to the escalating consumption and cost of sulphuric acid. The purpose of the test is to determine the quantity of sulphuric acid that will be consumed by a tonne of oxide ore during the leaching phase. Sulphuric acid is a key reagent in the metallurgical oxide leach process. Sulphuric acid was produced on site by sulphur burning acid plants using the Contact process. Increases in global sulphur prices prior to the 2008 Global Financial Crisis (GFC) meant that onsite acid production was becoming extremely expensive. To illustrate the rapid rise in acid production costs over this period, price data for sulphur from 2005 to 2014 is presented in Figure 5.1.

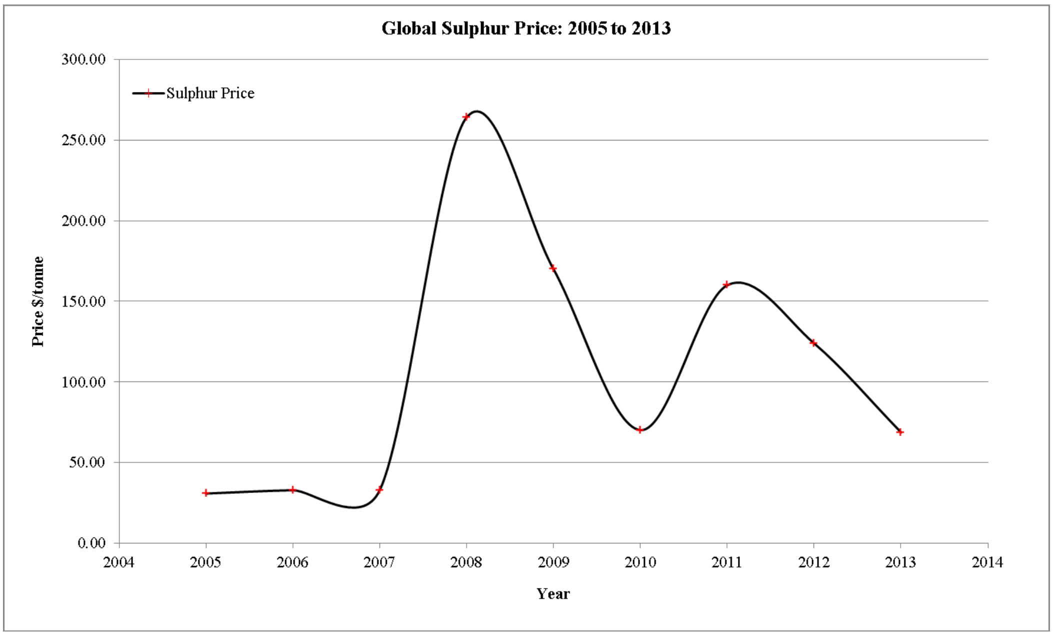


Figure 5.1 Global sulphur prices from 2005 to 2013. Adapted from Kelly and Matos (2015).

Contracts with two Zambian copper smelters allowed Kansanshi mine to purchase acid from their "off-gas" acid plants. However acid is a difficult and dangerous commodity to transport. The distance from the smelter to the mine is ~240km on crowded and poorly maintained roads. Thus the cost of smelter off-gas acid was also extremely high.

The principle that drove the introduction of a gangue acid consumption test was; acid consumption (and therefore acid costs) in the oxide leach circuit was rising. By determining a quantity of gangue acid consuming material in each tonne of ore, the geology team could identify that ore in the pit and defer it to stockpiles.

5.2 Gangue acid consumption test

Gangue acid consumption (GAC) values are determined by titration analysis. A quantity of pulverised sample is placed in a beaker and dissolved into a measured amount of sulphuric acid with a pH indicator solution. Sodium carbonate is added to the beaker using a volumetric burette, until the pH indicator shows neutrality. A series of calculations are carried out using the titration measurements to determine the kilograms of gangue acid consuming material per tonne of ore. The value is then reported as kilograms of gangue acid consuming material per tonne of ore (kg/t).

The GAC test is only carried out on selected geological samples. Once the ASCu assay for a range of geological samples has been completed the laboratory will select all samples with an ASCu >0.30% for further GAC analysis. The 0.30% ASCu limit is set to allow the laboratory to focus the GAC analysis on samples of ore that are likely to report to the oxide process plant.

5.3 Errors associated with the gangue acid consumption data

Test work conducted by Phiri and Chongwe (2014) has shown that a range of physical characteristics can affect the GAC analysis result. These physical characteristics were listed as:

- The sample particle size;
- The mass of the sample;
- The strength of the acid used in the digest; and,
- The quantity of acid used in the digest.

This work went on to compare the laboratory conditions for GAC analysis to the physical conditions that leachable oxide ore is subjected to in the oxide metallurgical circuit. The differences between laboratory analysis conditions and leach plant conditions were listed as:

- Laboratory samples are pulverised to a much finer particle size when compared to the particle size from mill grinding. A finer grind size increases the surface area of the particles and increase opportunity for a reaction to occur;
- The laboratory analysis uses a weaker concentration of sulphuric acid (10g/mol) compared to the oxide leach plant (~95g/mol). Weaker acid concentration means that the reaction is less vigorous under laboratory conditions and therefore may not fully react with all of the potential GAC minerals in the sample; and,
- The laboratory uses a finite quantity of acid in the titration as a basis for the calculation. In very high GAC content samples this leads to "caking" where all of the acid is immediately consumed and the sample forms a thick paste. As a result there is no acid left to perform the titration process. In this situation, the sample is given a nominal 300kg/t GAC value which marks an upper limit of detection. In the process plant there is abundant acid available. Reported GAC values from the process plant are often higher than 300kg/t.

Oxide leach plant samples are collected every hour from the cyclone overflow immediately after the oxide ball mill. These samples are sent to the laboratory for copper, gold and GAC analysis, which feeds into the process plant control system. The hourly cyclone overflow samples are composited on a daily basis to provide an average GAC value for all milled oxide ore during the previous 24 hours. This value is reported along with the average copper and gold mill head grades for the same time frame. Calculated acid consumption figures are also derived as part of the control of the counter current decantation (CCD) circuit. The CCD circuit is the immediate downstream phase of the oxide metallurgical process after leaching. Acid consumption in the CCD circuit is measured by flow meters located at the outflow point of the acid storage tanks. This measurement is reported as "total acid consumption" (TAC). An average GAC for a 24 hour period is calculated using a series of equations that account for acid consumption by the leached copper minerals presented in Equation 5.1, acid lost to tailings presented in Equation 5.2 and acid consumed by gangue minerals presented in Equation 5.3.

$$Tonnes\ H2SO4\ consumed\ by\ Cu\ in\ leach\ =\ Cu\ metal\ tonnes * 1.54 \qquad \textbf{Equation 5.1}$$

$$Acid\ lost\ to\ tails\ =\ \textbf{Equation 5.1} * 1.04 \qquad \textbf{Equation 5.2}$$

$$GAC\ =\ TAC\ -\ \textbf{Equation 5.1}\ -\ \textbf{Equation 5.2} \qquad \textbf{Equation 5.3}$$

To illustrate the difference between the laboratory derived GAC value from the cyclone overflow sample and the actual plant acid consumption data calculated at the CCD circuit, a comparison between the two data sets has been presented in Figure 5.2. The two data sets cover the time frame from 1st January 2008 to 31st December 2014. The data sets were plotted against each other on a "Log Quantile/Quantile scatter plot", (Log Q/Q plot) using Supervisor™. The comparison shows three areas where there is a clear bias in the data. These areas have been indicated on Figure 5.2 and can be listed as:

1. Deviation away from the “line of equality” towards the calculated CCD circuit data at the upper range of data values (green circle). This indicates that more acid is actually consumed in the process plant than the laboratory assay test shows;
2. At the lower end of the data range deviation away from the line of equality towards the CCD circuit data indicates that the laboratory process is unable to accurately analyse GAC at such low levels (red circle); and,
3. There is no laboratory data for GAC values >300kg/t (above blue dashed line). However calculated acid consumption in the CCD circuit data indicates that, GAC values greater than 300kg/t are being recorded in the process plant.

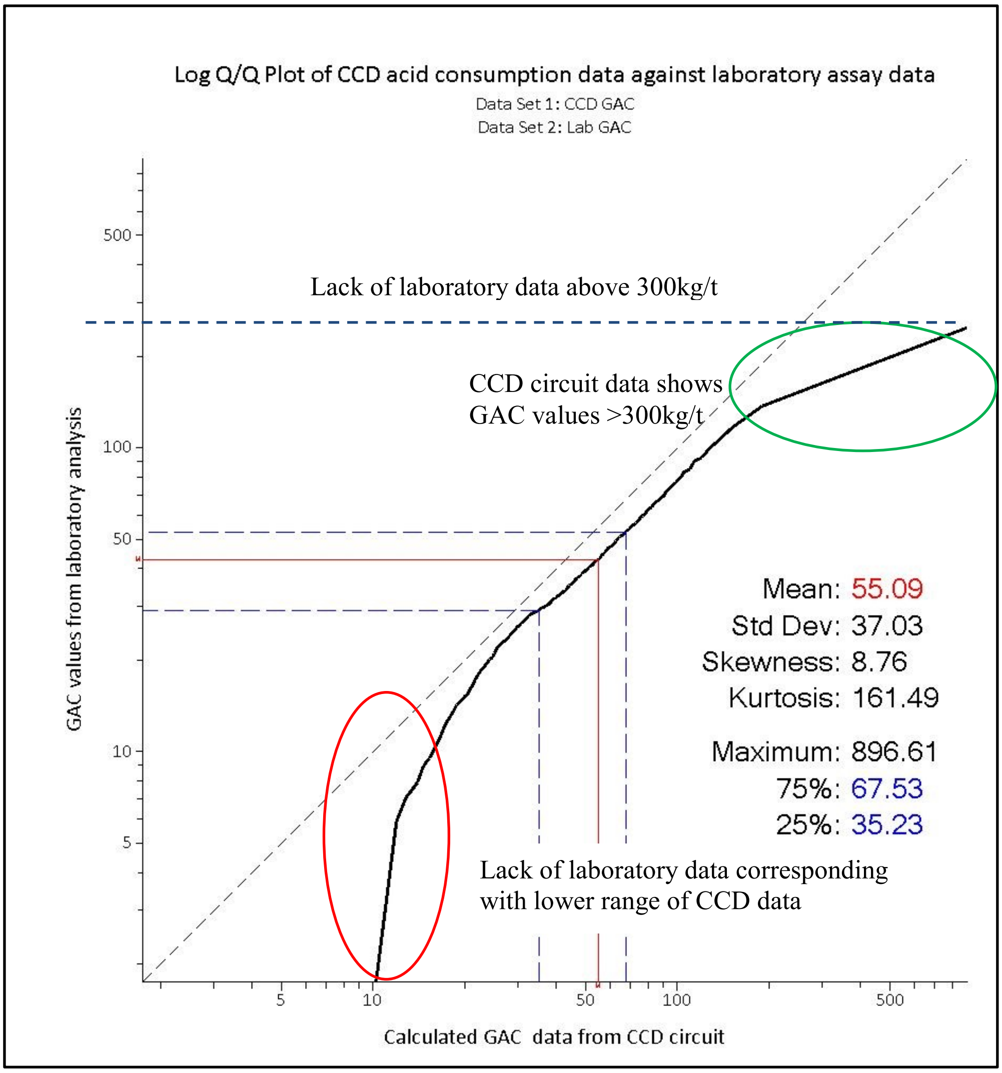


Figure 5.2 Log Quantile-Quantile analysis of laboratory derived GAC values against corresponding CCD circuit acid consumption data.

5.4 Gangue acid consumption in a geological context

As part of the update to the Kansanshi mineral resource model a review of GAC assay data was carried out by Ogilvie (2014). This review was independent of any work on GAC presented thus far. During the preparation of data for the 2015 Kansanshi mine mineral resource model update, it was realised that GAC numbers being reported contained significant risk. The review of geological data was intended to understand the geological distribution of GAC within the deposit.

5.4.1 *The spatial distribution of GAC at a deposit scale*

Using Supervisor™, GAC assay data was plotted on a standard histogram to understand the distribution of the data population. The results of this analysis are presented in Figure 5.3 Histogram of gangue acid consumption A clear bi-modal distribution of data indicates that there are two populations of data present. The first range is from 0kg/t to 150kg/t while a second population greater than 150kg/t is dominated by a spike at 300kg/t with a few additional peaks up to 350kg/t. Each population has a different geological control on their respective GAC content. Further analysis was needed to understand the spatial distribution of these two populations with respect to their geological controls.

Using the spatial analysis tool pack in Supervisor™, it was possible to correlate the two observed populations to their respective host stratigraphies. Figure 5.4 shows a cross section through the geological model that has been colour coded by laboratory derived analysis of GAC content.

A lower range of GAC data, from 0kg/t to 150kg/t, is coloured blue and represents clastic host rocks such as schists and phyllites which are dominantly composed of silicate minerals. The repeating layers of blue in Figure 5.4 indicate the top most clastics, upper mixed clastic, middle mixed clastics and lower mixed clastics sequences respectively. The second population of data around the spike at 300kg/t observed in Figure 5.3 has been coloured red in Figure 5.4. This population of data represents units dominantly composed of carbonate bearing minerals i.e. marble. Again multiple repeating layers can be observed in Figure 5.4, an upper red layer corresponds to the top most marble followed by the upper marble and lower marble, respectively.

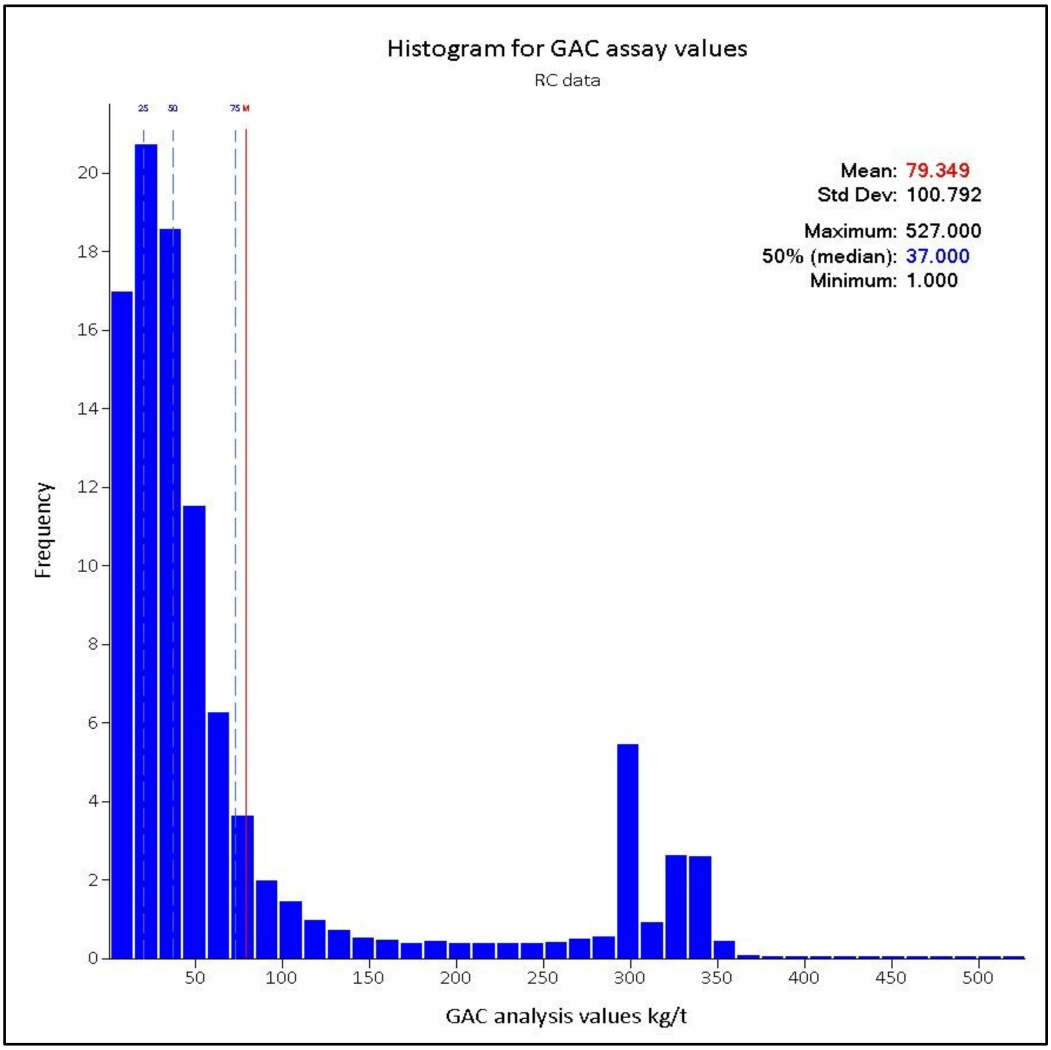


Figure 5.3 Histogram of gangue acid consumption data from R.C. drilling assays.

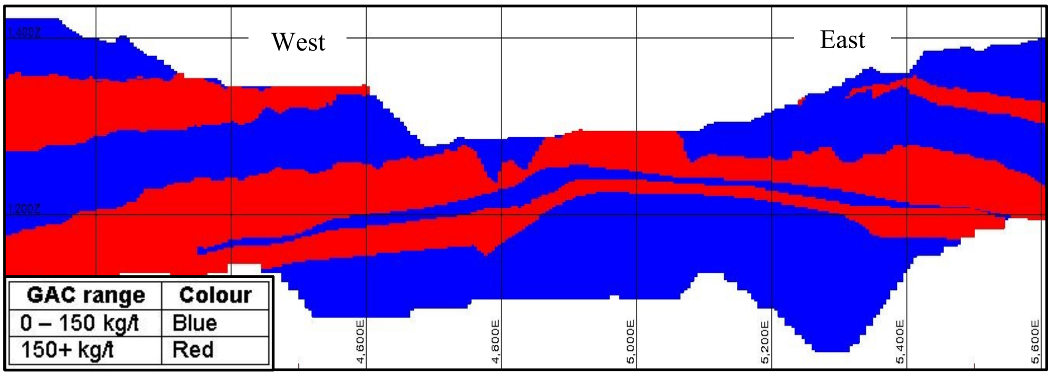


Figure 5.4 A cross section through the geological model colour coded by the identified ranges of GAC values.

5.4.2 *The relationship between GAC and ASCu grade*

As previously mentioned in this chapter, the need for GAC analysis was determined by a sample having a grade >0.30% ASCu. It was therefore important to understand the relationship between GAC and ASCu. Using a similar method to that presented in Section 5.3.1, an analysis of GAC at various ASCu grade was completed. Data was filtered from 0.30% ASCu up to 3.30% ASCu in 0.60% ASCu increments. At each 0.60% ASCu increment a histogram of GAC data was created. A composite of the six histograms is presented in Figure 5.5. The histograms were compared to each other to identify any changes in the distribution of data populations due to increasing ASCu grade. This analysis clearly demonstrated that there is no relationship between ASCu grade and GAC values. Despite a range of data filters based on ASCu grade being applied, the bi-modal population distribution presented in Figure 5.3 continues to hold true at all ASCu grades analysed. A noticeable drop in the number of GAC assay values can be seen from 2.10% ASCu upwards. This is due to there being relatively fewer data points available at higher ASCu grades.

The laboratory method of selecting samples for GAC analysis based on a grade >0.30% ASCu is not appropriate. There is no link between ASCu grade and GAC. The sample selection method does not account for the spatial distribution of GAC within the deposit and it does not correspond to the composition of major lithological units that produce higher or lower GAC values.

5.4.3 *Using GAC in the geological model*

Through this investigation it was found that there is a significant disconnect between the intended use of GAC data in the geological model and the actual use of that data. Mine planners and metallurgists believe that due to the laboratory analysis, the data is estimated into the geological block model and therefore each mining block has a single precise GAC value. However the mine geology team use the GAC analysis to determine ranges of GAC for each stratigraphic package. These ranges have a statistical mean, which is assigned to the geological model through a set of constraints based on the wireframe surfaces created through geological logging. These assigned values are periodically reviewed during the mineral resource model update process. Sample selection for GAC analysis should be independent of ASCu grade and represent the spatial distribution of the lithological units.

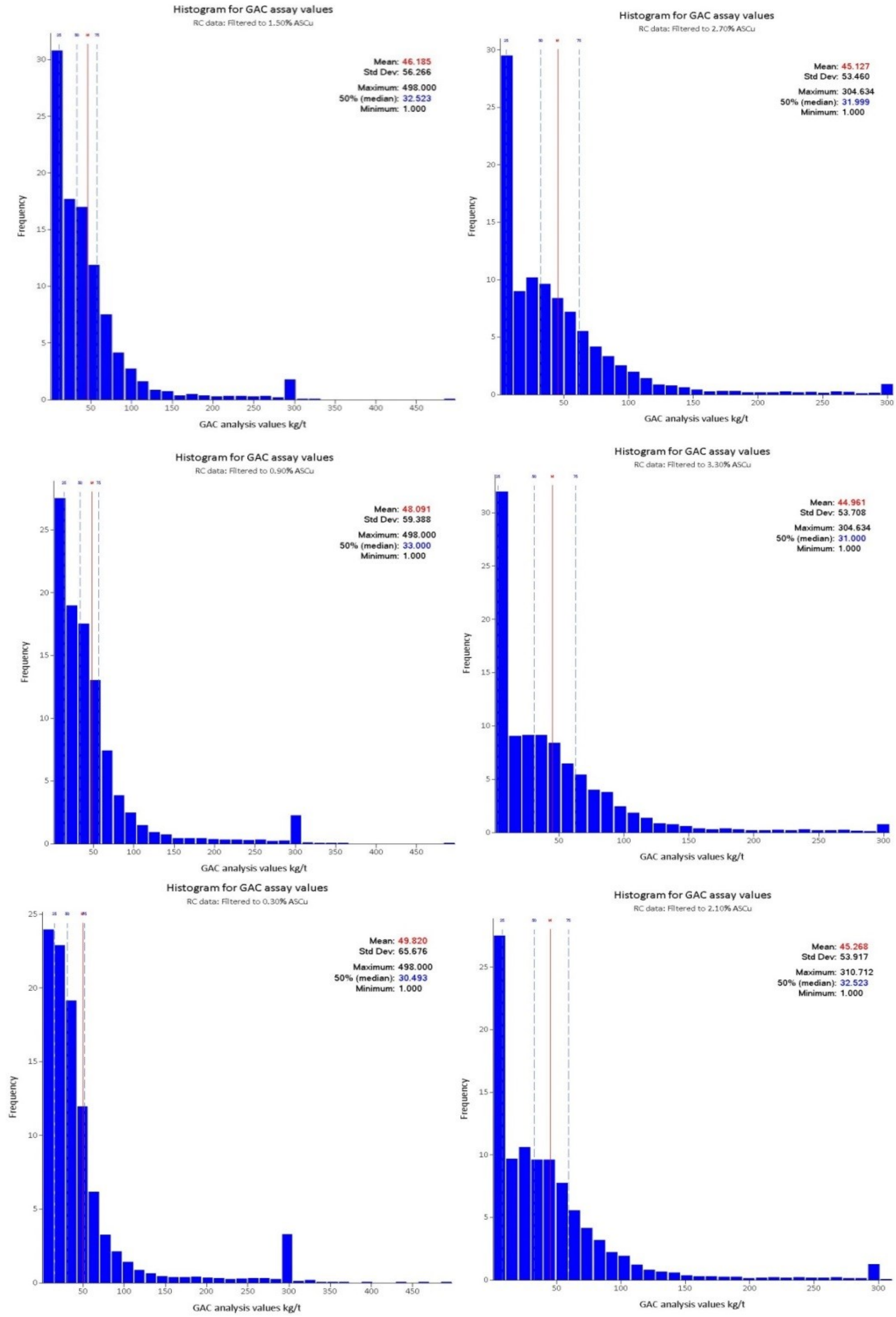


Figure 5.5 Composite of six histograms showing the similarity in distribution of GAC grade for varying ASCu grades.

5.5 Summary of gangue acid consumption analysis

The laboratory analysis of gangue acid consumption is a metallurgically driven requirement to determine a costing for acid consumption. This method has been incorporated into the grade control strategy without suitable understanding of the data. Through an investigation of the GAC analysis method and corresponding data it has been found that there are three significant concerns with the use of this data.

Firstly; the sample selection method is inappropriate as it assumes a correlation between ASCu grade and GAC value. The selection of samples with an ASCu >0.30%, creates a bias in the representation of data across the deposit. Analysis presented in this chapter shows that there is no correlation between GAC and ASCu grade. As a result GAC values do not account for the spatial geological relationship between carbonate bearing and non-carbonate bearing lithologies.

Secondly; the laboratory analysis method is only relevant under a certain set of conditions. Inappropriate assumptions are used to determine the upper and lower limits of detection. Any GAC value is affected by three key variables these variables have been reported as:

- Particle size;
- Acid concentration; and,
- Acid quantity.

Thirdly; the physical conditions present in the oxide ore leach plant differ from the laboratory method used. As a result the process plant is able to record actual GAC values that are both greater than the upper limit of laboratory detection and less than the lower limit of laboratory detection.

Further discussion on more appropriate, analysis and use of GAC data is presented in Chapter 8, as part of the integration of data into a geo-metallurgical strategy. A detailed review of the current grade control strategy employed at Kansanshi mine is presented in Chapter 6. This review is intended to provide a context to the current integration of data across Kansanshi’s mine value chain.

6.0 A REVIEW OF THE MAT_TYPE ORE CLASSIFICATION SYSTEM.

6.1 Introduction

As a result of the findings presented in Chapter 4 and Chapter 5, it was necessary to review the current system of ore classification employed at Kansanshi mine. The purpose of this review was to provide a context for the current use of all available data in making sound business decisions. By fully understanding the current practices employed at the mine it may be possible to identify opportunities for improved integration in a geometallurgical context.

The grade control process at Kansanshi mine is driven by an ore classification and mineral resource to mineral reserve conversion system called "Mat_Type". It was introduced during the third quarter of 2009 as part of the strategy to survive the global financial crisis (GFC) of mid-2008. From April 2008 to February 2009 copper prices plummeted from highs of US$3.82 per pound (/lb) to only US$1.39 /lb, (Roth, 2015). During this period, the company's operating costs could not be reduced in proportion to the declining copper price. As a result maintaining the mine's profitability was becoming increasingly more difficult.

To keep the company operating a number of cost saving initiatives were introduced but ultimately, a step change in the mine plan was needed. Larger volumes of higher grade ore were needed to increase copper production, while significantly lowering fixed and variable operating costs. This approach would result in an overall decrease in the production cost per pound of copper (US$/lb), to fall in line with the prevailing market price for copper.

At that time the metallurgical process plant was very sensitive to a range of factors that could negatively impact copper recovery. In order to manage the impact of these factors on the metallurgical process, a set of quality control rules was developed by the metallurgical team to ensure that only the "best quality" ore available was fed to the process plant at any point in time.

For the sulphide ore floatation process, good ore quality was defined through achieving the highest possible copper recovery. Through the oxide leach process good ore quality was defined as having the highest possible ASCu grade and the lowest possible GAC value.

The quality control rules determined by the metallurgical department were incorporated into a mineral resource to mineral reserve conversion process which was called the “Mat_Type” system. This system was designed to use cost data, plant recoveries and quality control rules in a series of calculations, to create cut-off grades that would define a series of ore categories which could be coded into the geological model. It was believed that this system would allow control of the economic factors that defined good and poor quality ore, thus allowing the mine to keep pace with a highly dynamic economic environment. The system was designed and implemented in 2009 to achieve the following objectives:

- Add modifying factors to the geological models and thus ensure that only the geological data relating to the mineral reserve is used in the grade control process;
- Improve the “quality” of the sulphide ore product by reducing the ASCu content to <0.05%;
- Improve the “quality” of the oxide ore product by increasing ASCu grades to >0.6% and reducing GAC content;
- Manage the consumption of sulphuric acid (H_2SO_4) which was an important yet costly reagent in the oxide ore processing circuit; and,
- Direct high grade (>1.00% TCu) mixed float ore to the process plant, while stockpiling all low grade mixed float ore.

6.2 Overview of the Mat_Type system functions

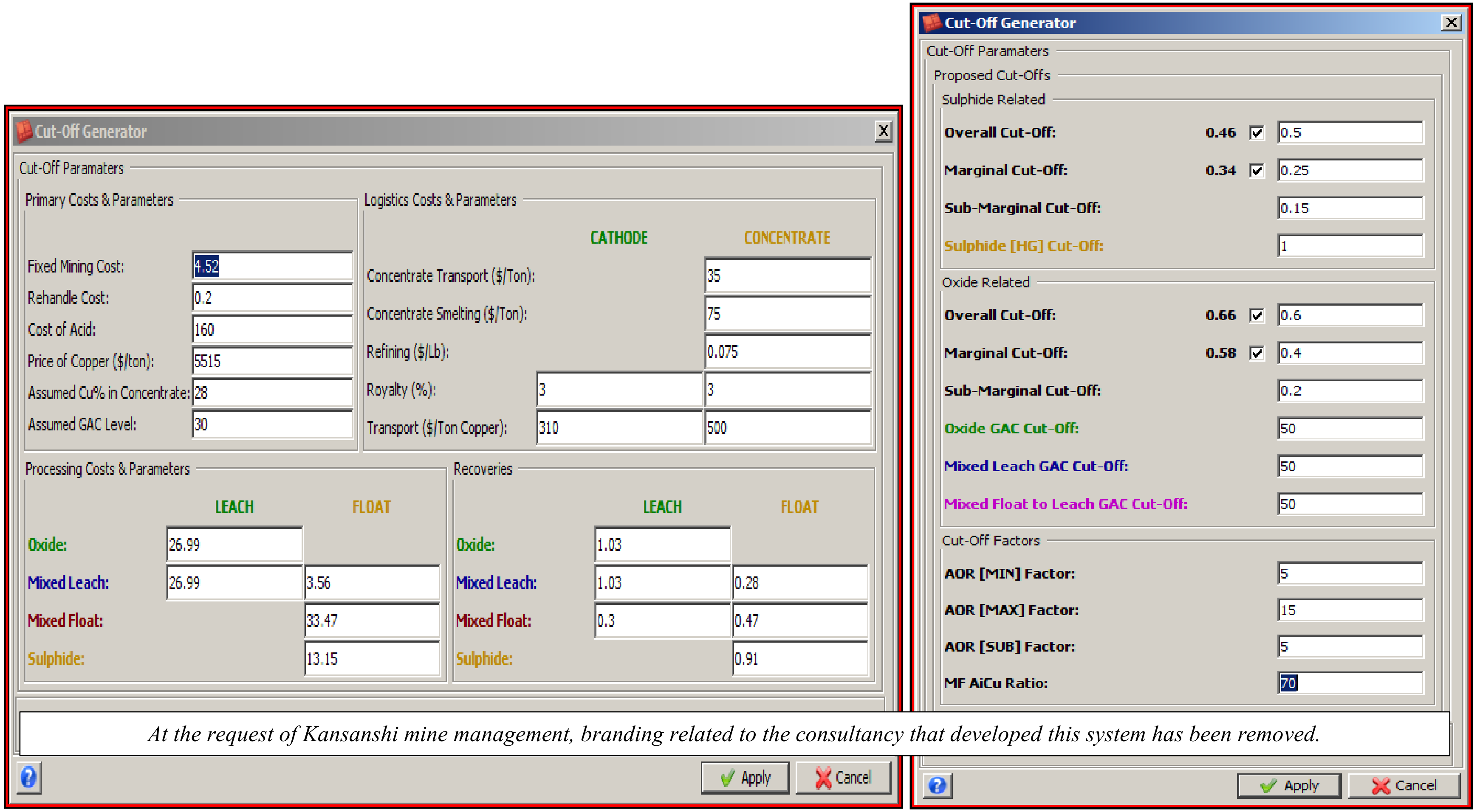
The “Mat_Type” system is controlled by an automated macro, written in “Tool Command Language” (.tcl) and operated in Surpac™. The macro has been further modified into “.tcl Byte Code” (.tbc) format for security purposes.

Metallurgical recovery factors and financial data are applied to a data entry form as presented in Figure 6.1. The macro then carries out a series of calculations to determine cut-off grades for each of the three mineral processing systems; sulphide flotation, mixed floatation and oxide leach.

Calculated cut-off grades are published in a second form also presented as part of Figure 6.1. This provides an opportunity to review the values before applying them to a geological model. This second form also provides functionality to overwrite any calculated value with a manual input. Once the values in the second form are accepted, they are applied to a series of logical “IF” statements. These “IF” statements create a numerical code to place ore into one of 22 “ore quality” bins which define numerous ore, waste and mineralised waste categories based on the different mineral processing methods. Through a systematic review of the Mat_Type system a number of key risks were identified. These risks can be summarised as:

- A lack of ownership and management of the macro, supported by a formalised review process to ensure that it is fit for purpose;
- Inappropriate application of inaccurate and out-dated input data;
- Application of inappropriate or contradictory “quality control” factors at various stages of the ore classification process; and,
- Lack of application of geological data to support the definition of certain ore categories.

Current calculated cut-off grades and corresponding ore categories defined by the Mat_Type classification system are summarised in Table 6.1. A table of factors and calculations used to classify ore with a Mat_Type code is provided in Appendix 1. A flow chart showing the complex array of “IF” statements used to make decisions that will group ore into Mat_Type bins, has been included in Appendix 2. To assist in the explanation and understanding of how the system works and where these risks lie, reference is made to Appendix 1 and Appendix 2 respectively, in conjunction with subsequent sections of this report.



At the request of Kansanshi mine management, branding related to the consultancy that developed this system has been removed.

Figure 6.1 Primary data input tables for the Mat_Type system macro.

Table 6.1 Mat_Type ore classification matrix aide memoire.

Mat_Type Number	Lower Grade	Upper Grade	AOR Range	GAC limit	Ore Type	Characteristics
11	0.20% ASCu	0.40% ASCu	n/a	n/a	Oxide mineralised waste	Very low Cu and ASCu grade, intensely weathered.
12	0.40% ASCu	0.60% ASCu	< 5	n/a	Low grade oxide	Intensely weathered ore with low acid consuming characteristics.
13	0.60% ASCu	>0.60% ASCu	< 5	>50 kg/t	High grade oxide ore, high GAC	Intensely weathered, good quality, leachable ore with high acid consuming characteristics, direct to HGAC stockpile
14	0.60% ASCu	>0.60% ASCu	< 5	<50 kg/t	High grade oxide ore, low GAC	Intensely weathered, good quality, leachable ore with low acid consuming characteristics.
22	0.60% ASCu 0.68% AICu	>0.60% ASCu >0.68% AICu	< 5	>50 kg/t	High grade mixed leach, low GAC	Leachable ore with less ASCu compared to "13" but similar GAC. Low AOR indicates it can be leached.
24	0.60% ASCu 0.68% AICu	>0.60% ASCu >0.68% AICu	< 5	<50 kg/t	High grade mixed leach, high GAC	Leachable ore with less ASCu compared to "14" but similar GAC. Low AOR indicates it can be leached.
32	0.60% ASCu 0.68% AICu	>0.60% ASCu >0.68% AICu	>5 & <15	>50 kg/t	High grade mixed leach to stockpile, low GAC	Leachable ore similar to "22" but a very high AOR indicates very high acid consumption potential, not worth leaching, direct to stockpile
34	0.60% ASCu 0.68% AICu	>0.60% ASCu >0.68% AICu	>5 & <15	<50 kg/t	High grade mixed leach to stockpile, high GAC	Leachable ore similar to "24" but a very high AOR indicates very high acid consumption potential, not worth leaching, direct to stockpile
41	0.15% AICu	0.25% AICu	n/a	n/a	Sulphide mineralised waste	Very low TCu grade, un-weathered.
42	0.25% AICu	0.50% AICu	n/a	n/a	Low grade sulphide	ASCu <0.05%, easy to float and maintain high recovery
43	0.50% AICu	1.00% AICu	n/a	n/a	Medium grade sulphide	ASCu <0.05%, easy to float and maintain high recovery
44	1.00% AICu	>1.00% AICu	n/a	n/a	High grade sulphide	ASCu <0.05%, easy to float and maintain high recovery
52	0.60% ASCu 0.68% AICu	>0.60% ASCu >0.68% AICu	>15	n/a	High grade mixed float	High ASCu grade weathered ore, high AOR therefore not worth leaching, MF_AICu_Ratio >70 indicating suitable for mixed ore floatation
54	0.25% AICu	0.60% ASCu	>5	n/a	Low grade mixed float	Low ASCu grade weathered ore, high AOR therefore not worth leaching, MF_AICu_Ratio >70 indicating suitable for mixed ore floatation.
62	0.25% AICu	0.60% ASCu	>5	n/a	Low grade RED	Low TCu grade weathered ore, high AOR indicates high acid consumption potential and an MF_AICu_Ratio <70 indicates it is not suitable for mixed ore floatation, direct to stockpile
64	0.60% ASCu 0.68% AICu	>0.60% ASCu >0.68% AICu	>15	n/a	High grade RED	High TCu grade weathered ore, very high AOR indicates very high acid consumption potential and an MF_AICu_Ratio <70 indicates it is not suitable for mixed ore floatation, direct to stockpile
70	0.00% ASCu 0.00% AICu	0.20% ASCu 0.15% AICu	n/a	n/a	Waste	Blocks estimated with sufficient assay data to demonstrate a lack of copper and therefore be classed as waste.

6.3 Lack of ownership and management of the system

There is no formal documented ownership of the Mat_Type macro or the onsite mineral resource to mineral reserve conversion process. This system has been in operation at Kansanshi mine for five years without any form of a review to understand whether it is still fit for purpose.

The Mat_Type macro currently sits with the mine geology team. This is a legacy issue as the geology team are the custodians of the three dimensional block model. Each time new assay data is updated in the model, the script is re-run by a junior geologist to re-assign ore categories into the model. The Mat_Type macro is a "black box" and no one using it fully understands how inputs were derived, what functions the macro performs and what the data outputs mean.

Cut-off grades are created as part of the process of converting of a mineral resource to a mineral reserve. As such, they are considered to be the realm of the mining engineer. No documentation can be found to explain why calculated cut-off grade values have been overwritten by a set of manual data inputs at some point in the last five years. The difference between calculated cut-off grades created by the Mat_Type macro and the manually created cut-off grades is presented in Table 6.2.

An additional aspect to the risk posed through lack of onsite ownership, is that the source code for the Mat_Type macro is the intellectual property of the consultant who created it. Any error correction, development or improvement would have to be made by the original creator. Thus it cannot be easily validated by a member of the Kansanshi mine technical services team to ensure any change is being correctly applied. In-depth understanding and ownership of any ore classification system must reside on site, with the Kansanshi mine technical services department to allow for regular systematic review, thereby ensuring that any such system remains fit for purpose in the face of changing business dynamics.

Table 6.2 Variance between calculated cut-off grades and manually applied cut-off grades used in the Mat_Type system.

Ore Category	Calculated Value	Manual Input Value	Variance
Medium grade sulphide (43) to low grade sulphide (42)	0.46	0.50	+8%
Low grade sulphide (42) to sulphide mineralised waste (41)	0.34	0.25	-26%
Sulphide mineralised waste (41) to waste (70)	n/a	0.15	+100%
High grade oxide (14) to low grade oxide (12)	0.66	0.60	-9%
Low grade oxide (12) to oxide mineralised waste (11)	0.34	0.40	+17%
Oxide mineralised waste (11) to waste (70)	n/a	0.20	+100%

6.4 Inappropriate application of out-dated data inputs

Over the past five years key events have created a significant disconnect between the input data being used and the current economic situation. Key components of the economic data applied to the Mat_Type macro were compared to financial data used in the 2014 annual budget. A summary of this comparison is presented in Table 6.3. Since the introduction of the Mat_Type system a number of key data inputs have changed as a result of internal business decisions and external geo-political or geo-economic factors. Examples of some of these key changes are summarised below:

- The copper price rose from US$1.39/lb at the start of January 2009 to US$3.00 /lb by January 2014 (Roth 2015);
- In late 2009 the Government of the Republic of Zambia increased the mineral royalty tax from 3% to 6%. A further revision of mineral royalties from 6% to 20% occurred in January 2015;
- Transportation, smelting and refining charges are revised annually as part of the sales contract for copper concentrate;

- Milling and processing capacity for oxide ore has increased from 6 million tonnes per annum in 2009 to 14.5 million tonnes per annum in 2014;
- From 2006 to 2013 acid prices fluctuated between US$150/t and US$350/t;
- Fixed and variable costs structures within the company have changed; and,
- Gold production has become a major revenue stream for the business.

Table 6.3 A comparison between key costs currently applied to the Mat_Type cut-off grade generator and the values applied to the 2015 annual budget.

Cost element	Mat_Type	2015 Annual budget
Copper Price (US$/t)	US$5,515/t	US$6,944/t
Mineral Royalty	3%	20%
Smelting Charge (US$/t concentrate)	US$75/t	US$93/t
Acid price (US$/t)	US$160/t	US$280/t

All of these changes to input parameters have a significant impact on cut-off grades. Any change in economic circumstances should trigger a series of reviews of cut-off grades and other modifying factors by an appropriately qualified member of the mine technical services team. A simple annual review of input data, supported by a sign off process, controlled by the technical services manager, with peer review from metallurgical and financial managers would ensure that input data to an ore classification system is regularly checked and validated. Coupled with this review process a set of periodic reviews of the methodology utilised in the macro should be completed in order to understand if the process is still fit for purpose, to account for business developments that may have occurred since the previous review.

6.4.1 ***Incorrect application of cut-off grades to ore categories***

The most simplistic form of cut-off grade, often termed the "break-even cut-off grade" is defined by Bohnet (1990 Pp. 478) and is calculated using the formula presented in Equation 6.1. Fixed and variable operating costs are defined as; the mining and processing costs per tonne of ore processed and net return cost is defined as; smelting, refining transportation and royalty costs.

Equation 6.1

$$Cut - off\ grade = \frac{((fixed + variable\ costs) * mine\ dilution\ factor)}{((commodity\ price - net\ return\ costs) * plant\ recovery)}$$

Calculated cut-off grades should be applied in a manner that best represents the overall value of the contained metal in that estimated ore block. In the case of Kansanshi mine, the calculated cut-off grade must be applied to the estimated total copper grade, demonstrating that the block is economic to process under that set of economic assumptions.

The Mat_Type system utilises a break-even cut-off grade calculation as part of the ore classification methodology. However, the "IF" statements used by the Mat_Type system to categorise ore, applies cut-off grades to acid soluble copper grade and acid insoluble copper grade independently. As noted in Chapter 4 ASCu and AICu are functions of TCu and thus cannot be considered as independent entities. The result of the incorrect application of cut-off grades to ore is that each ore bin has an elevated average TCu grade with respect to the cut-off grade used to define it. In addition a quantity of economic ore is artificially downgraded to a lower grade category because the TCu value sits just below the cut-off grade which is being applied to either ASCu or AICu .

To investigate the impact of elevated copper grades in each ore bin on the total volume of processable ore, two block model reports were created. Firstly, the 2013 Kansanshi mine mineral resource model was constrained using survey wireframe surfaces to select the volume of ore mined between 1st January 2014 and 31st December 2014. Tonnes of ore above each Mat_Type cut-off grade were reported. Cut-off grades were then changed to apply to total copper rather than AICu or ASCu and the report was regenerated.

A variance between the two cut-off grade methodologies is presented in Table 6.4. To simplify the interpretation of the resulting data, the 17 Mat_Type ore bins have been grouped together to represent the three main ore categories with respect to mineral processing. This analysis shows that over 3.4 million tonnes more ore could have been realised, across the three main metallurgical process routes, during the period under review. Since each ore type has been categorised by a cut-off grade value being applied incorrectly to AICu this ore has been directed to inappropriate ore processing circuits or inappropriate stockpiles. It is also important to note that the average grade of each ore grouping has been artificially raised by using AICu and ASCu cut-off grades independently of the TCu grade.

Table 6.4 Variance of available ore tonnes between correct and incorrect application of cut-off grades for all material mined in 2014.

	Cut-off grade incorrectly applied to AICu			**Cut-off grade correctly applied to TCu**		
Ore Type	**Tonnes**	**TCu**	**ASCu**	**Tonnes**	**TCu**	**ASCu**
Oxide	10,035,519	2.53	1.69	9,977,726	2.54	1.69
Mixed Float	20,061,618	0.81	0.21	22,870,832	0.76	0.21
Sulphide	10,380,903	0.84	0.02	11,126,853	0.80	0.02
Mineralised Waste	7,773,211	0.22	0.04	8,697,981	0.21	0.04
Waste	66,671,487	0.06	0.01	62,249,342	0.07	0.01
Grand Total	**114,922,734**	**0.53**	**0.21**	**114,922,734**	**0.50**	**0.20**

Variance	**Tonnes**	**TCu**	**ASCu**	**Metallurgical grouping of Mat_Type codes**
Oxide	-1%	0%	0%	Mat_Type ore bins: 12, 13, 14, 22, 24,
Mixed Float	12%	-6%	-2%	Mat_Type ore bins: 32, 34, 52, 54, 62, 64
Sulphide	7%	-5%	0%	Mat_Type ore bins: 42, 43, 44
Mineralised Waste	11%	-6%	-1%	Mat_Type ore bins: 11, 41
Waste	-6%	-13%	0%	Mat_Type ore bin: 70

6.5 Application of inappropriate or contradictory “ore quality” factors.

After cut-off grades are calculated additional “ore quality” factors are applied using the Mat_Type “IF” statements to further sub-divide ore categories. These ore quality factors are:

- Limiting ASCu in sulphide ore to <0.05%;
- Acid to ore ratio (AOR);
- Gangue acid consumption (GAC); and
- Mixed float acid insoluble copper ratio (MF_AICu ratio).

These factors were developed by the metallurgical department based on their need to maintain high copper recoveries to meet production targets. Each factor was developed at a separate time for a particular metallurgical process in isolation of the larger business value chain. They were then all brought together into the Mat_Type system in 2009 in order to create centralised control in their application.

6.5.1 *Limiting acid soluble copper in sulphide ore*

The first categorisation clause utilised by the Mat_Type system separates sulphide ore from any other possible ore type. To do this a limit of <0.05% ASCu is used. This limit was derived from the analysis of recovery data as a function of ASCu head grade and is presented in Figure 6.2. It was shown that recovery could drop below 90% if ASCu grade rose above the 0.05% limit. This analysis supported the development of a quality control rule, to be built into the Mat_Type system, which would ensure sulphide recovery remained above 90%. Any ore that could negatively impact copper recovery in this circuit is directed elsewhere i.e. stockpile or mixed ore processing. Using an arbitrary numerical limit of 0.05% ASCu on sulphide ore feed creates two significant disconnects in the business. These disconnects are summarised in the following points:

i. Chalcopyrite is the abundant sulphide mineral in the deposit. A >0.05% ASCu limit does not represent any natural boundary that can be attributed to a geological process such as; mineral emplacement, weathering or oxidation. Furthermore it does not consider the chemical properties of the mineral species that impact the metallurgical process being employed to recover copper.

ii. The limit does not consider precision errors in the ASCu laboratory assay methodology. An error of +/- 0.2% ASCu can be applied to any ASCu assay above a limit of detection of 0.01% ASCu (Ogilvie 2015).

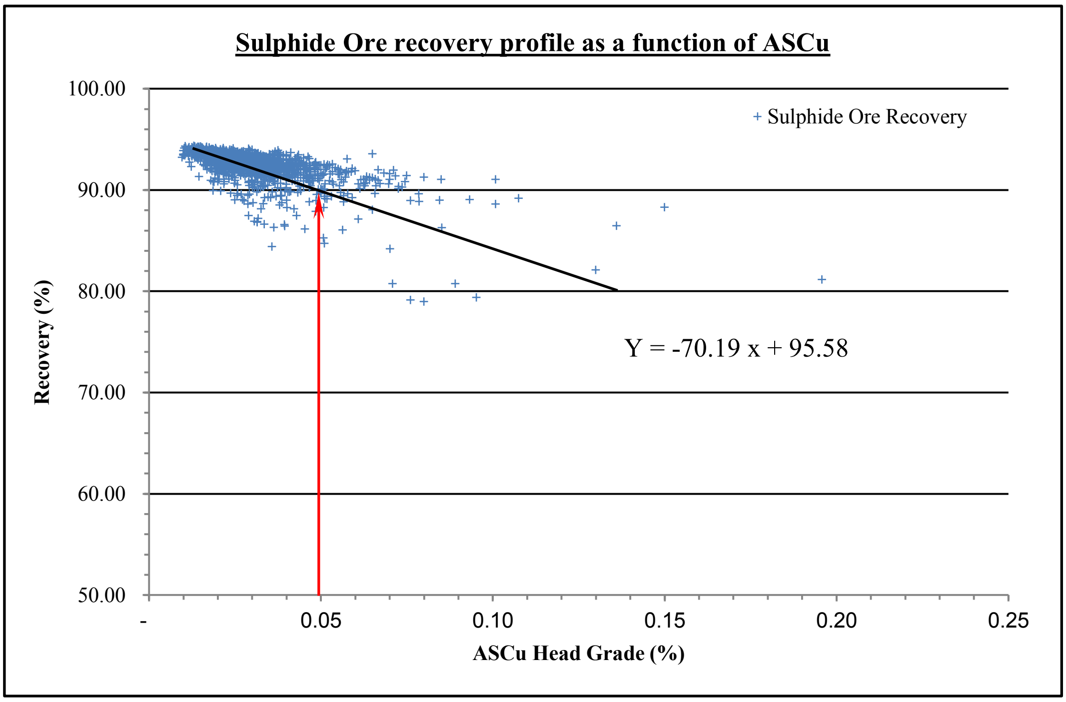


Figure 6.2 Sulphide ore flotation recovery as a function of increasing ASCu content in the mill head grade.

In order to quantify the benefit of improved copper recovery in sulphide ore by deferring ore with >0.05% ASCu, to mixed float, recovery data from the mixed ore process plant was also plotted as a function of ASCu head grade and is presented in Figure 6.3. This analysis shows that mixed ore copper recovery at a grade of 0.05% ASCu or more, would only average 78% total copper, compared to the +/- 90% total copper recovery attained by the sulphide ore process.

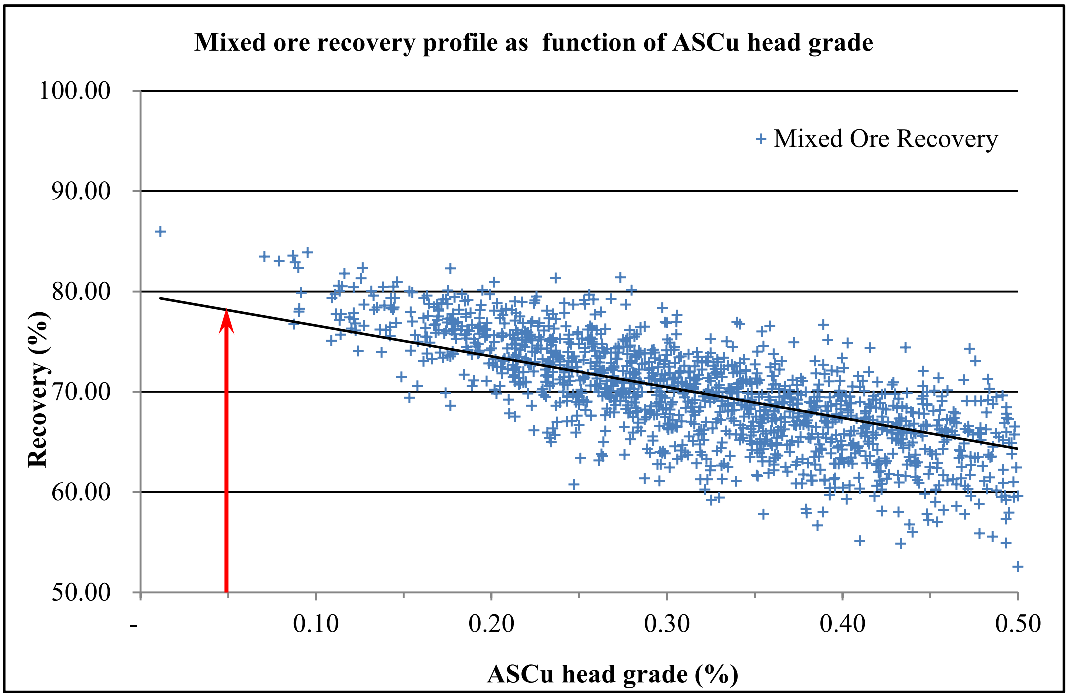


Figure 6.3 Mixed ore process copper recovery as a function of increasing ASCu content in the mill head grade.

By superimposing recovery profiles for the mixed ore and sulphide ore processing circuits onto the same chart it can be further shown that the mixed ore process generally attains much lower copper recoveries than the sulphide circuit. The analysis presented in Figure 6.4 demonstrates that the plant recovery, within the context of the quality control rules designed to support high performance is not appropriate. The loss of copper through poor mixed float recovery is far more detrimental to value creation than the negative impact that ASCu (within a range) will have on copper recovery in the sulphide process.

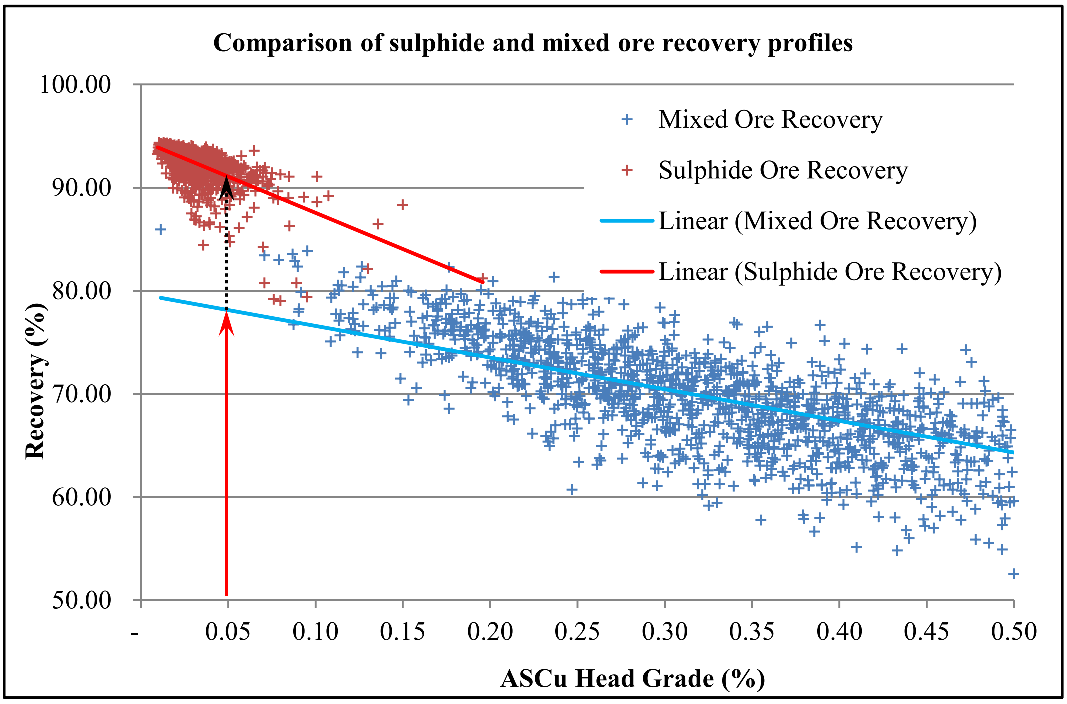


Figure 6.4 Comparison of copper flotation recovery profile as a function of ASCu mill head grade.

6.5.2 *Acid to ore ratio factor*

The "acid to ore ratio" (AOR) was a factor developed in 2009 as a method to group ore by its ASCu grade and GAC content. The method of calculating this ratio is presented in Equation 6.2.

$$AOR = \left(\frac{\left(\frac{GAC}{ASCu}\right)}{10}\right)$$

Equation 6.2

The AOR attempts to determine an economic grouping for oxide ore, the principle being that higher GAC values can be processed because the ASCu grade would also be correspondingly higher. The ore would therefore yield enough value to warrant the cost of acid consumed by the GAC minerals.

An AOR value of five was determined to be the highest limit that could be "economically" processed in the Leach SX/EW metallurgical plant. Three cut-off grade limits are defined in the Mat_Type classification system. These limits are presented in Table 6.5.

If the AOR value of an ore block in the geological model is above the threshold for processable oxide ore then that block is downgraded through a series of additional quality control bins to a "lower quality" category. Each bin is controlled by a mathematical "IF" statement, coded into the Mat_Type System macro. Reference is made to Appendix 1 and Appendix 2 which illustrate the application of AOR limits to the relevant ore bins by the Mat_Type "IF" statements, to classify oxide and mixed float ore types.

Table 6.5 AOR cut-off values applied to the Mat_Type ore classification system.

Lower AOR value	Upper AOR value	Name	Ore type and destination
0	5	AOR min	Ore type 22 and 24; Stockpile
5	15	AOR sub	Ore type, 32, 34, 54 and 62; Stockpile
15	>15	AOR max	Ore type 52, 54, 62 and 64; Ore with high GAC and high AOR values. Stockpiled or processed through the mixed float circuit.

At the time of undertaking this study there was no documentation that explains how the AOR threshold numbers were derived. Furthermore there was no information detailing the impact in changes to copper price, processing cost or acid cost on the AOR threshold values.

Due to a lack of supporting documentation for the Mat_Type system a misinterpretation of the ore bins that it created, led to ore types with higher AOR values being inappropriately stockpiled. Typically it was considered that if the GAC was very high then the ore would be classified as a mixed float ore product irrespective of its mineralogy. In some instances this misinterpretation led to high GAC ore being processed through the mixed float metallurgical process leading to very poor copper recovery despite the head grade. To illustrate this point a worked example is outlined below.

Consider a block of ore at 1.5% ASCu, which is currently the mean estimated ASCu grade for defined oxide ore in the deposit. This block is composed of primary oxide mineralisation in a weathered sap-rock matrix. The mean GAC value of the deposit from a geological view point, as noted in Chapter 5, is 79.35kg/t. This GAC value would sit within a non-carbonate bearing clastic lithology. It would therefore represent a “low” GAC value from a geological perspective. The AOR calculation using these parameters is presented in Equation 6.3.

$$AOR = \left(\frac{\left(\frac{GAC}{ASCu}\right)}{10}\right) \quad = \quad \left(\frac{\left(\frac{79.34}{1.5}\right)}{10}\right)$$

Equation 6.3

At an AOR of 5.28 this ore block of ore would be directed to a long term stockpile. As previously noted GAC and ASCu grade are independent variables. Creating a ratio between them does not take cognisance of important geological details which have been listed in this report as:

- Geological subdomains based on mineralogy formed by the interaction of oxidation processes in mineralised structures as presented in Chapter 4; and,
- The significance of the calcium carbonate content of a lithological unit which gives the GAC value, as presented in Chapter 5.

6.5.3 *Application of a GAC limit after the AOR factor*

As discussed in Section 6.5.2, gangue acid consumption is an integral part of the AOR calculation and subsequent ore classification. However GAC also plays a significant role as a standalone factor in the final stages of leachable ore classification.

Each of the oxide and mixed ore categories is further sub-divided into two final ore bins based on its GAC value. Ore will be placed in either of these bins if it is above or below a defined GAC threshold of 50kg/t. This sub-division is intended to act as a final quality control step with "higher quality" i.e. lower GAC, entering one bin and "lower quality" i.e. higher GAC, entering the other. The interaction of AOR and GAC statements for classifying oxide ore, are demonstrated in Figure 6.5. Again as noted in earlier sections, no documentation exists to explain why a value of 50kg/t was chosen.

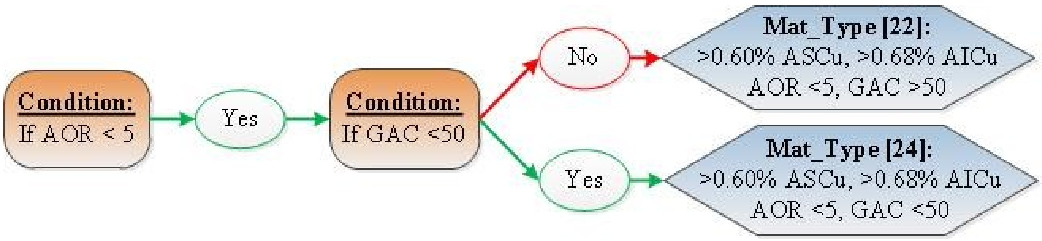


Figure 6.5 Conflicting AOR and GAC conditions used to classify oxide ore.

A worked example of the impact of using GAC as a standalone classification criterion has been presented in Equation 6.4, using the mean GAC value previously reported as 79.35kg/t and an ASCu grade of 2.00% which is considered to be high grade for the mineral processing plant. The resulting AOR is 3.97. It can be shown that after this block passes the AOR classification criteria it would go on be categorised as Mat_Type 22 and directed to a long term stockpile. This is due to the inappropriate application of GAC data in ore quality control criteria.

Equation 6.4

$$AOR = \left(\frac{\left(\frac{GAC}{ASCu}\right)}{10}\right) = \left(\frac{\left(\frac{79.34}{2.00}\right)}{10}\right)$$

6.5.4 *Mixed float acid insoluble copper ratio*

During the aftermath of the global financial crisis of mid-2008, work was conducted by the Kansanshi mine's metallurgical team in 2009 to improve copper recovery in the mixed ore processing circuit. Expansions to milling and floatation capacity created the opportunity to process mixed ore at a lower cost by recovering from stockpiles.

It was shown by Paquot (2009) that when using sodium hydrogen sulphide (NaHS) a mineral suite containing primary copper bearing sulphide minerals and secondary copper bearing sulphide minerals, with high degrees of surface tarnish could be more effectively recovered by a modified flotation method called "Controlled Potential Sulphidisation" (CPS) than compared to a standard flotation system. Key to controlling the degree of surface oxidation in the mixed ore product was the "Mixed Float Acid Insoluble Copper Ratio" (MF_AICu Ratio). This ratio is calculated using the formula presented in Equation 6.5 and is expressed as the percentage of a TCu assay that is in an AICu form.

$$MF\_AICu\ Ratio = \left(\left(100 - \left(\frac{AICu}{TCu}\right)\right) * 100\right)$$

Equation 6.5

From Paquot's work, a threshold of not less than 70% AICu was determined and applied as a quality control attribute for mixed float ore. The Mat_Type "IF" statement uses this ratio to separate mixed ore into bins for processing or stockpiling. This limit was based on metallurgical test work of a range of samples taken from ore stockpiles that had already been contaminated and mixed through a poorly understood and inappropriate grade control systems. Additionally, the ratio was based on AICu which is a function of TCu.

Given the improved geological understanding of the oxidation processes presented in Chapter 4 and improved understanding of the kinetics of NaHS on secondary sulphide minerals brought about by a continuation of Paquot's original work, the MF_AICu Ratio should be reviewed. Furthermore the ratio should be put into the context of ASCu with respect to TCu. This would bring alignment between the geologically calculated oxidation ratio data and the metallurgical process.

6.5.5 *Multiple criteria used to classify mixed float*

Due to the number of complex mathematical relationships in the Mat_Type process and the number of inappropriate factors already discussed, some ore is unable to be classified correctly. Mixed float ore has to pass through a range of criteria to be classified as such. These criteria AOR, GAC, MF_AICu_Ratio and a series of incorrectly calculated and applied cut-off grade conditions have been outlined in previous sections of this chapter. However during the ore classification process a series of conflicting criteria are simultaneously applied to the mixed float ore groups through different routes in the classification decision tree. The portion of the Mat_Type IF statement logic responsible for this ore categorisation is presented in Figure 6.6. It demonstrates that Mat_Type 54 and Mat_Type 62 can be created by following different routes through the system. As a result different criteria are applied allowing ore with very different and unrelated geological characteristics to be grouped together.

During daily mine production, Mat_Type 54 is directed to the mixed float metallurgical process. It is considered to be the highest grade and best quality of the various mixed float ore options, while Mat_Type 62 is directed to stockpile as it is considered to be the poorest quality mixed float ore. However from this analysis it can be seen that this assumption is incorrect. Both Mat_Type 54 and Mat_Type 62 have a range of ASCu grade between 0.6% and 0.4% and they are distinguished from one another by the MF_AICu ratio which is a floatation quality control factor.

To understand the impact of this misclassification, a block model report was created for all mixed float ore types mined from 1st January 2014 to 31st December 2014. This data was then further constrained by the specific criteria outlined in Figure 6.6. The results of the analysis are presented in Table 6.6. Note that Mat_Type categories 54 and 62 have been subdivided into "54_a", "54_b" and "62_a", "62_b", respectively, as per the annotation in Figure 6.6.

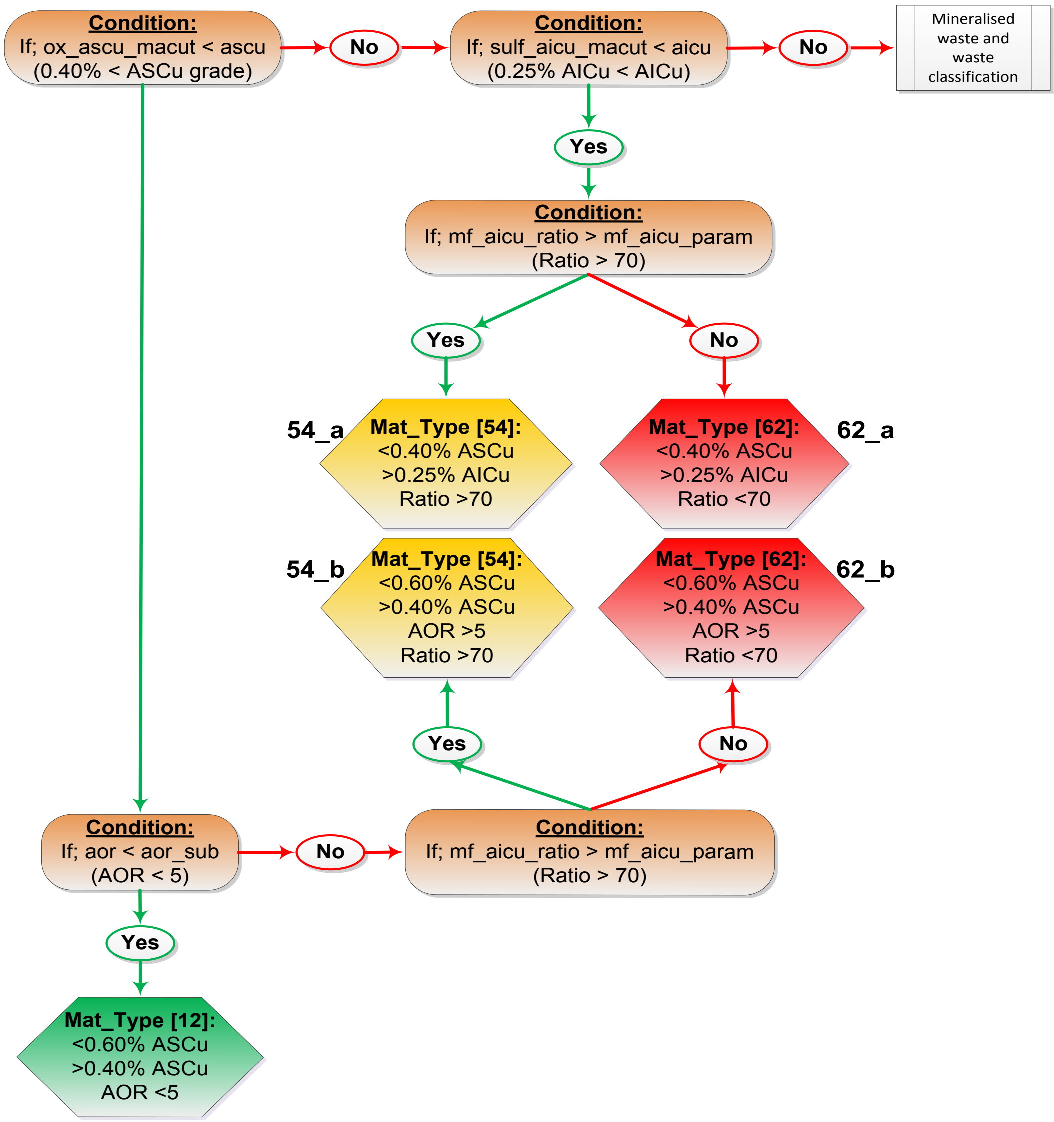


Figure 6.6 A portion of the Mat_Type IF statement used to categorise some of the mixed float ore types.

The analysis presented in Table 6.6 shows that there is very little consistency in TCu and ASCu grade or GAC values between the various sub-categories of Mat_Type 54 and 62. The ore has been inappropriately classified and the lack of consistency would have had a significant impact on mineral processing and copper recovery. Category 54 was directed to the crusher as a high grade high quality mixed float. However the overall TCu grade is only 0.94% yet this ore contains a portion with a TCu grade of 2.36%. Had this very high grade portion been separated correctly it may have been possible to feed it to the metallurgical plant as a more appropriate individual ore type thus realising a greater recovery of metal tonnes. Mat_Type 62 was directed to a low grade mixed float ore stockpile. However if the AOR factor did not play a role in classification, Category 62_b could be considered a low grade leachable oxide ore with a higher GAC component. Unfortunately it is now bulked together with Category 62_a which has a lower ASCu grade on a stockpile where it will be difficult to realise the contained value.

Table 6.6 Material mined in 2014 that was categorised as Mat_Type code 54 and 62.

Mat_Type	Tonnes	TCu	ASCu	GAC	AOR	MF_AICu Ratio
54_a	10,321,241	0.86	0.12	51.32	42.77	86%
54_b	580,937	2.36	0.48	56.89	11.85	80%
54 Total	**10,902,178**	**0.94**	**0.14**	**51.56**	**36.83**	**85%**

Mat_Type	Tonnes	TCu	ASCu	GAC	AOR	MF_AICu Ratio
62_a	6,541,625	0.53	0.22	51.55	23.43	59%
62_b	2,617,812	0.94	0.49	53.87	10.99	48%
62 Total	**9,159,437**	**0.65**	**0.30**	**52.17**	**17.39**	**53%**

6.5.6 *Multiple criteria used to classify mineralised waste.*

As with certain mixed float ore categories presented in Section 6.5.4, mineralised waste was also subjected to misclassification in the Mat_Type system. The portion of the Mat_Type decision process classifying mineralised waste is presented in Figure 6.7. If ore is unable to be classified in the main body of the system, it can be pushed into a category at the end of the Mat_Type decision process, which is designed to collect this material and assign it with the ore code 41. However the code 41 has also been used to specifically classify sulphide mineralised waste. Through the mine planning processes, material is mined and grouped together on a sulphide mineralised waste stockpile, despite having a range of different characteristics.

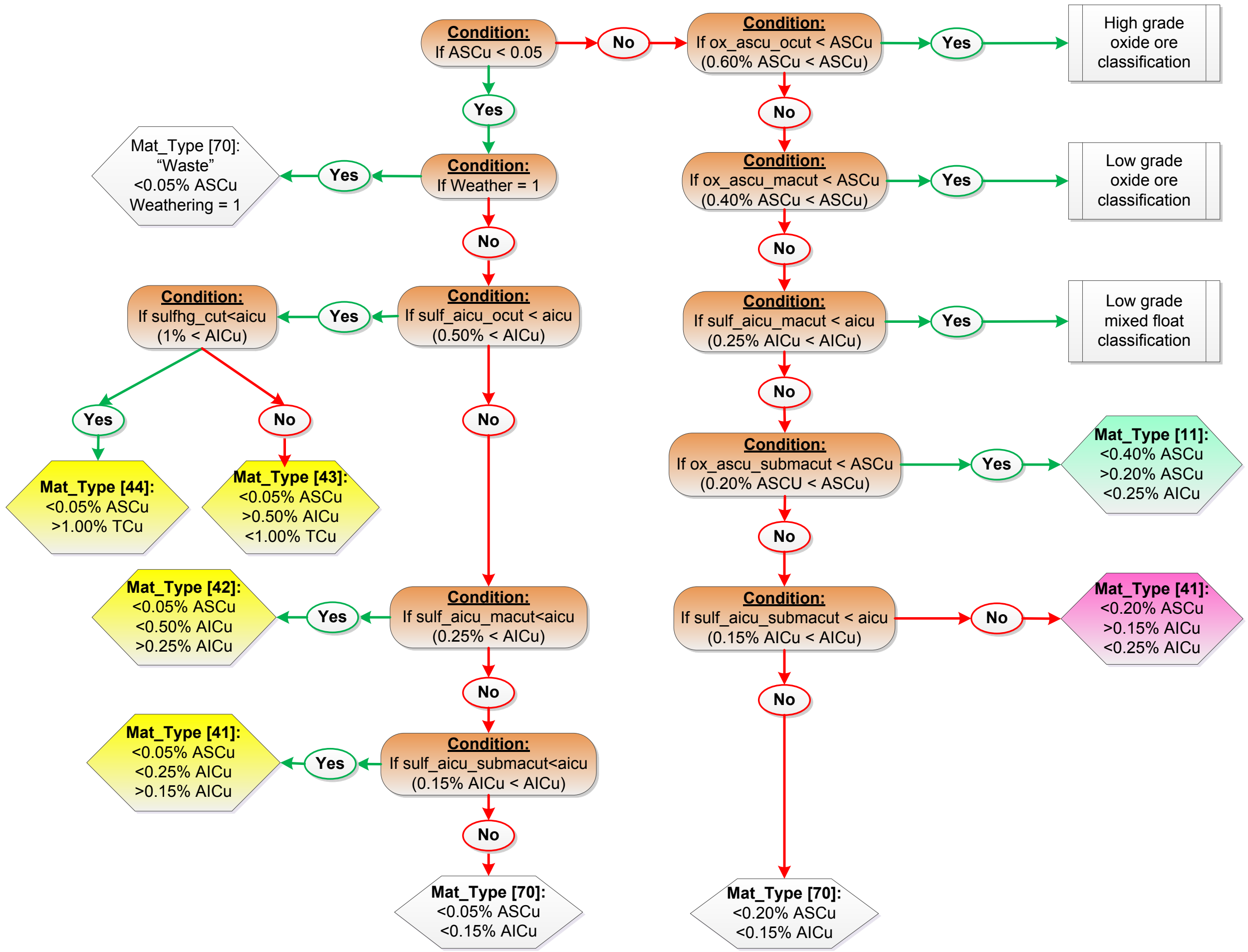


Figure 6.7 Mineralised waste and waste classification in the Mat_Type system.

To understand the quantity of ore that has been incorrectly classified as Mat_Type 41, data was extracted from the geological block model using Surpac™. Ore mined from 1st January 2014 to 31st December 2014 was reported using a series of constraints which are presented in Table 6.7. Note that the two Mat_Type 41 bins that exist in the Mat_Type system have been subdivided into Categories 41_a and 41_b, respectively for the purposes of this analysis. The results of this analysis are presented in Table 6.7.

Table 6.7 Constraint set used to extract data for Mat_Type category 41 from the geological block model.

Constraint order	Constraint
A	Below 31st December 2013 end of month pit survey
B	Above 31st December 2014 end of month pit survey
C	All blocks coded as Mat_Type 41
D	All blocks with TCu <= 0.25 = 41_a
E	All blocks with TCu > 0.25 = 41_b

Table 6.8 Material mined in 2014 that was categorised as Mat_Type code 41.

Mat_Type	Tonnes	TCu	ASCu
41_a	5,322,680	0.20	0.02
41_b	835,087	0.27	0.03
41 Total	6,157,767	0.21	0.02
Percentage of total	15%		

Of the total quantity of Mat_Type 41 (sulphide mineralised waste) that was mined in 2014, 15% was incorrectly categorised due to conflicting classification criteria. This material was mined and directed to a long term sulphide mineralised waste stockpile when it should have reported to another more appropriate ore category.

Assuming an average copper price of US$3.00/lb throughout 2014 approximately US$14.9 million of potential value was lost through this misclassification.

It is likely that as a result of this incorrect classification of mineralised waste a significant portion of the stockpiled material is now contaminated which may pose challenges if this material was to be reclaimed from the stockpile and processed in the future.

6.5.7 *Potential mineralised material that has been downgraded to waste*

In addition to the incorrect classification of mineralised waste, it is important to note that there are three "waste" bins in the Mat_Type system. Each of the three waste bins has been given the Mat_Type code "70" yet each bin has a unique set of defined criteria. The specific criteria for the three bins have been presented in Table 6.9. As with Mat_Type category 41, material entering waste bins can also have a range of TCu grades that are above the defined waste cut-off grade value of 0.15% AICu.

Table 6.9 The various conflicting criteria used to classify waste in the Mat_Type system.

Waste bin sub division	Criteria
70_a	Un-weathered material with no oxidation, <0.15% AICu, <0.05% ASCu.
70_b	Weathered and oxidised material, <0.15% AICu, <0.2% ASCu.
70_c	Any material with an ASCu grade <0.05% which is highly weathered.

Using a similar methodology to that which was presented in Section 6.5.6, block model reports were generated for material classified as waste by the Mat_Type system in 2014. The results of this analysis are presented in Table 6.10 and show that 7% of the total waste mined in 2014 had an average grade greater than the defined waste cut-off grade of 0.15% TCu. In total assuming an average copper price of US$3.00/lb throughout 2014 approximately US$65million of value was lost through inappropriate classification of potential ore into waste categories using the Mat_Type system.

Table 6.10 Tonnes of waste (Mat_Type code 70) mined in 2014.

	Tonnes	TCu	ASCu	AICu
Mat_Type category 70_a	61,990,622	0.07	0.01	0.06
Mat_Type category 70_b	4,680,865	0.21	0.05	0.16
Total Mat_Type category 70	66,671,487	0.08	0.01	0.07
Percentage of total	7%			

6.6 Inappropriate resource to reserve conversion process

As mentioned in Section 6.1 the Mat_Type system is a mineral resource to mineral reserve conversion system. The macro has been designed to apply economic factors to all blocks in a geological block model, thereby converting the entire mineral resource to a mineral reserve. This is of particular concern when using the system to determine ore value in areas of the model where the drill coverage is sparse. At the extremities of the geological block model there are areas without enough data to accurately inform the grade estimation process. These blocks typically have a background value of 0.00 assigned to the TCu and ASCu grade. In the geological block model a set of attributes are created as part of the estimation process that record the number of samples used to estimate an individual block, along with the average distance between those samples. By using these attributes the geology team is able to quickly identify areas of the geological block model that lack data and need follow-up RC drilling.

During the mineral resource to mineral reserve conversion process the Mat_Type system sets all blocks with AICu grade <0.15% as category 70 i.e. waste. Any subsequent mine plan created using this data will direct all blocks with a code of 70 to the waste dump. However, geostatistically speaking a portion of these blocks are actually “unclassified” i.e. with more data the block could be classified as ore or waste. Through a lack of understanding of how the Mat_Type system is applied to a geological block model in the mineral resource to mineral reserve conversion process, a significant risk has been presented whereby potential ore blocks have been incorrectly mined as waste and directed to a waste dump due to a lack of geological data.

The application of factors during the mineral resource to mineral reserve conversion process needs to be done within the guidelines stipulated by an appropriate reporting code. In Kansanshi mine's case the company must adhere to National Instrument 43-101 of the Toronto Securities Exchange where it has a primary listing. The Mat_Type system cannot be applied across the entire model. There must be a method to constrain the application of the ore classification system. A mineral resource to mineral reserve conversion process can only be applied to areas where there is a sufficient quantity of geological data and geo-statistical confidence to support a robust the estimation of the copper grade. Furthermore these areas must also have a suitable evaluation of mineral economics, mining method and mine design criteria to be considered as a either a proven or probable mineral reserve.

It is extremely difficult to quantify the potential loss in value of unclassified material being mined as waste. The geological model is a constantly evolving entity, as new data is gathered and blocks are re-estimated. Therefore any blocks that were previously "unclassified" and therefore mined as waste, may now have an estimated grade in the model. A block model report from the current model will only give the current TCu and ASCu value of the blocks. These updated values may not be the same as the value that these "unclassified" blocks had when they were mined. Attempts to quantify the loss in value using other sources of data have also proved inconclusive. One such example was a review of production statistics from the reconciliation process. This data does not have any spatial reference. It is therefore not possible to correlate any mined tonnage to an *in-situ* location or the sampling data that would have been used to estimate a grade in that block.

6.7 Defining refractory mineralisation

In the Kansanshi ore deposit a portion of the mineralisation is termed as refractory. Typically this was copper mineralisation that could not be recovered using the available metallurgical processes. As part of the publication of the 2012 Kansanshi ore resource update, work was completed to understand and define refractory mineralisation. Through analysis of available geological data including assays, logging and mineralogical studies combined with laboratory scale metallurgical test work on a series of samples, the following definition of refractory mineralisation was determined:

> *"Refractory mineralisation can be defined as elemental copper contained within a matrix of smectite clays, iron oxides and other hydroxide mineral which are found predominantly in the saprolite zone of the upper mixed clastics."*
>
> Gregory *et al.* (2012, Pp. 36.)

Further to the definition of refractory mineralisation a series of criteria was developed in order to determine spatially where this mineralisation was and separate it out into a unique geological domain. A set of criteria was designed to identify copper mineralisation with low ASCu values situated in heavily weathered and heavily oxidised zones. These criteria have been defined as:

- Within the saprolite weathering domain;
- TCu grade >0.25%;
- ASCu grade <0.4%; and,
- Oxidation ratio <0.1.

Through the development of this understanding it was decided that refractory ore should be defined separately from other mineralisation during the estimation process. An attribute called "Redoxn" was created in the geological model and numerical identification code was assigned to each domain. The identification code and its respective domain criteria are presented in Table 6.11. Weathering domains had been defined by Hassen *et al.* (2010) and numerical codes had been assigned to the geological model to identify each of these domains. These weathering domains were defined as:

1. Completely weathered;
2. Partially weathered; and,
3. Fresh.

Table 6.11 Criteria used to define redox zones in the geological model. (Adapted from Gregory *et al*., 2012).

Zone	Redoxn code	Grade	Oxidation Ratio	Mineralogy	Weathering
Refractory	4	>0.25% TCu <0.40% ASCu	*n/a*	Elemental copper locked in a silicate matrix	Saprolite
Oxide	3	>0.40% ASCu	>= 0.5	Cu bearing oxide minerals	Saprolite, Sap-rock
Transition	2	>0.10% ASCu	< 0.5	Supergene zone composed of oxide mineralogy plus primary and secondary sulphide mineralogy	Sap-rock, Fresh
Sulphide	1	<0.10% ASCu	< 0.1	Primary copper bearing sulphide mineralisation	Fresh

Although refractory mineralisation was already known during the development of the Mat_Type system, it was not clearly defined. An attempt to separate refractory mineralisation from primary sulphide mineralisation was made using the weathering attribute in the geological model. The “IF” statement decision tree that attempts to classify refractory ore in the Mat_Type system is presented in Figure 6.8.

At that time it was assumed that weathering was solely responsible for mineral oxidation. Therefore all mineralisation in the completely weathered zone should have an ASCu value that is similar to the TCu value indicating complete oxidation producing primary copper oxide minerals. It was believed that one could use this logic to determine refractory mineralisation in the block model. Any block of mineralisation with <0.05% ASCu and a TCu grade greater than the cut-off grade, located in the "completely weathered" zone could be defined as refractory. Any mineralisation meeting this rule would simply be defined as waste. The updated understanding of refractory mineralisation published in the 2012 Kansanshi mine technical report, has not been included in the ore classification system. As a result refractory mineralisation is not being correctly identified and classified.

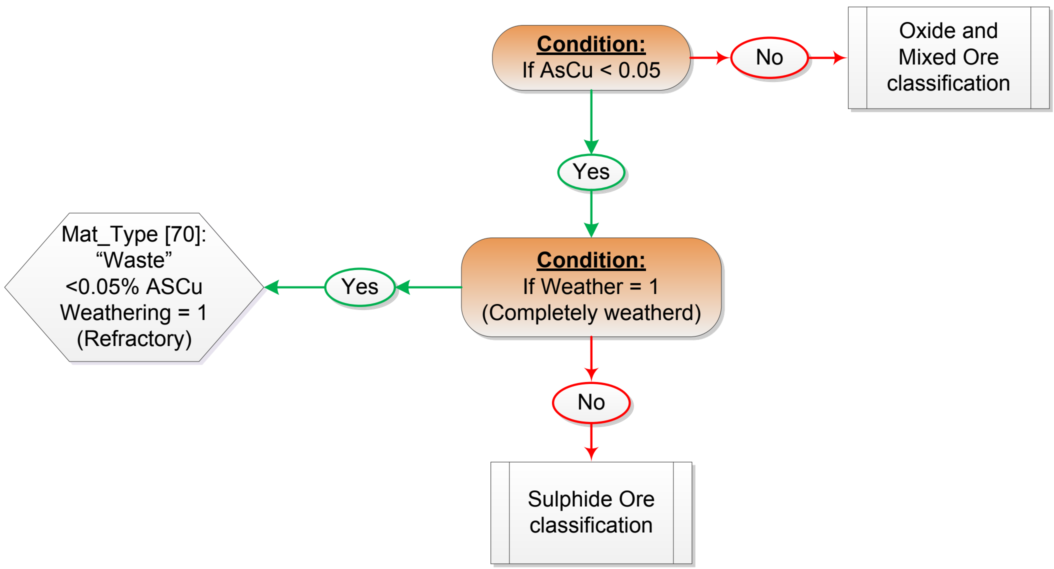


Figure 6.8 A portion of the Mat_Type decision tree that incorrectly uses weathering to classify refractory mineralisation.

6.8 Weathering zones

From 2010 to 2012 there was a change in terminology and coding method for weathering domains in the geological models. A new standard was introduced and all domains were updated accordingly. Until this review of the ore classification method at Kansanshi mine was completed, it was not understood how changes in the resource model would impact other aspects of geological data. Thus a significant disconnect now exists between the methodologies used to populate data in to the mineral resource model or grade control model. To illustrate the changes that have occurred from 2010 to 2012 in the definition of weathering and the associated coding used in the block models, a summary has been provided in Table 6.12.

As reported in Section 6.7 the Mat_Type system uses the weathering domains to determine refractory mineralisation. It was further reported that this was an outdated methodology which has been superseded by a more appropriate definition of refractory mineralisation published by Gregory *et al.* (2012). However from the data presented in Table 6.12 it can be seen that where the Mat_Type system was using the weathering code for completely weathered material as per the 2010 methodology, it is now using a weathering code for fresh rock due to the changes made in 2012. As a result there is a significant risk of sulphide ore being incorrectly hauled to waste dumps, while refractory mineralisation may have been hauled to the mineral processing plant.

Table 6.12 A summary of changes made to the weathering domains and associated coding in the Kansanshi mine mineral resource model.

2010 Resource model update		2012 Resource model update	
Code	**Description**	**Code**	**Description**
	n/a	4	Laterite
1	Completely weathered	3	Saprolite
2	Partially weathered	2	Sap-rock
3	Fresh	1	Fresh

6.9 Application of the Mat_Type ore bins in daily production

During the mine planning process, the 18 bins that Mat_Type creates, are grouped into five broad categories namely; sulphide ore, oxide ore, mixed float ore, mineralised waste and barren waste. A comparison of Mat_Type ore categories to the mine planning grouping is presented in Table 6.13. This method of grouping Mat_Type ore bins together does not represent a consistent application of the ore quality parameters used to divide the ore into 18 initial bins. The only categories that are consistently grouped by mine planners as per the intended original classification are mineralised waste and sulphide ore.

With reference to oxide ore, where it is believed that the planning ore groups are simultaneously separating out high GAC ore and high and low copper grade, it can be seen that these ore characteristics are actually being mixed together creating no definition between the desired quality control attributes. This observation can also be applied to mixed float ore where it is believed that combining ore bins in the mine plan indicates a simple difference between high grade and low grade mixed ore. The reality is a substantial mixing of distinct ore types with differing processing characteristics, undoubtedly leading to the erratic metallurgical performance typically seen in the mixed ore circuit.

Table 6.13 Grouping of individual Mat_Type ore codes for mine planning purposes.

Bin number	Mat_Type ore quality attributes	Bin number	Mine planning ore groups
11	Oxide mineralised waste	11	Oxide mineralised waste
12	Low grade oxide ore, AOR <5, GAC <50kg/t	12, 22	Low grade oxide ore, "low" GAC
13,	High grade oxide ore with "high" GAC	13, 22, 24	High grade oxide ore, GAC >50kg/t
14, 24	High grade oxide ore, >0.60% ASCu, AOR <5, GAC <50kg/t	14,	High grade oxide ore, >0.60% ASCu, AOR <5, GAC <50kg/t
22	High grade oxide ore, >0.60% ASCu, AOR <5, GAC >50kg/t	N/A	N/A
24	High grade oxide ore, >0.60% ASCu, AOR <5, GAC <50kg/t	N/A	N/A
32	High grade oxide ore, >0.60% ASCu, AOR >5, GAC >50kg/t	N/A	N/A
34	High grade oxide ore, >0.60% ASCu, AOR >5, GAC <50kg/t	N/A	N/A
41	Sulphide mineralised waste	41	Sulphide mineralised waste
42	Low grade sulphide	42	Low grade sulphide
43	Medium grade sulphide	43	Medium grade sulphide
44	High grade sulphide	44	High grade sulphide
52	High ASCu grade, good quality mixed float ore, AOR >15, MF_AICu_Ratio >70	N/A	N/A
54	Low ASCu grade, good quality mixed float ore, AOR >15, MF_AICu_Ratio >70	54	High grade high quality mixed float ore to be processed
62	Low ASCu grade poor quality mixed float ore, AOR >15 MF_AICu_Ratio <70	32, 34, 52, 62, 64	Low grade low quality mixed float ore to be stockpiled
64	High ASCu grade, poor quality mixed float ore, AOR >15, MF_AICu_Ratio <70	N/A	N/A
70	Waste	70	Waste

6.10 Chapter summary

This chapter has presented a review of the current grade control methodology employed at Kansanshi mine. This methodology is called "Mat_Type" and is controlled by an automated macro process in Surpac. Through this review it was found that the Mat_Type system is no longer appropriate in the current business environment.

The Mat_Type system was designed to apply modifying factors to a mineral resource model in order to convert it into a mineral reserve. To do this it used a number of data inputs to calculate cut-off grades. These were applied along with a series of quality control rules to ore *in-situ*. These ore categories were then used to guide the mine planning process by determining the most effective metallurgical process to recover copper from the ore. The system has been in use at Kansanshi mine for five years and in that time no review of the data inputs, processes or data outputs has been carried out, despite a number of significant developments in the business and increased understanding of geological and metallurgical processes.

A systematic review of the Mat_Type macro has shown that a number of key sets of technical knowledge have not been integrated into the ore control system, posing significant risk in four key areas of the business. These three areas of risk can be summarised as:

i. <u>Cut-off grades and mineral resource to mineral reserve conversion process</u>
   Out-dated financial data, incorrectly applied to cut-off grade calculations creates artificially high cut-off grades. These grades are then applied inappropriately to acid soluble copper or acid insoluble copper, further increasing the cut-off grade with respect to total copper. This is downgrading economic ore to mineralised waste and barren waste categories, thus destroying value;

ii. Lack of ownership of the system

A series of undocumented manual data inputs that override calculated cut-off grades, demonstrates a serious lack of security and integrity in the system. A lack of ownership by the relevant technical section to ensure that the system is regularly audited, further contributes to the security concern;

iii. Inappropriate use of outdated geological knowledge

From 2010 to 2015 significant improvements have been made in the geological understanding of the deposit. Most notable are the changes to weathering domain definition and the improved understanding of GAC and refractory mineralisation. The Mat_Type system has not been reviewed and updated to include these developments leading to misclassification of various types of mineralisation; and,

iv. Application of inappropriate quality control factors

The inappropriate use of AOR and MF_AICu_Ratio as "quality control" factors does not take cognisance of geological or mineralogical data. As a result different ore types are bulked together into "poor quality" categories and stockpiled, destroying potential value.

It can be concluded that, Kansanshi mine's current ore classification system does not integrate detailed knowledge across key technical silos. This is leading to the destruction of value. As such the Mat_Type system is no longer relevant or appropriate, in the context of the current business strategy. Such significant disconnects between the development of "ore quality" parameters and the practical application of these parameters to mine planning and mineral processing further demonstrates that the system can no longer be used effectively.

The ore classification system and overall grade control strategy must be updated and brought in line with the collective developments in understanding of geology, mineralogy, metallurgy and mineral economics. A cross discipline approach is required to develop a robust process that converts mineral resources to mineral reserves. This approach needs to be communicated effectively to all technical sections; geology, mine planning, mineral processing, in order to maintain the integration needed to drive the strategy forwards in the correct context. Finally, there needs to be a formalised systematic review process for all data inputs in the mineral resource to mineral reserve conversion process. This review must be part of the annual business review and must feed into the annual mine planning and budgeting processes.

The following chapter presents a review of the mineral processing systems at Kansanshi mine. Over the years the plant has developed into a substantial entity with a high degree of flexibility. Management of this flexibility and the understanding of what is considered practical is still not integrated into the larger decision making process of the business. This review is intended to provide an understanding of the key aspects of the various mineral processing systems employed at the mine. This will aid in the integration of key technical knowledge such as geology, mineralogy and mine planning into the mineral processing strategy.

7.0 A REVIEW OF THE MINERAL PROCESSING SYSTEMS AT KANSANSHI MINE

7.1 Introduction

Kansanshi mine has a large and complex metallurgical plant capable of recovering copper from a variety of copper bearing ore types. A simplistic high level division of the plant infrastructure can be made based on the types of metallurgical process. There are three main processes all of which are served by a dedicated crusher and set of mills. These metallurgical processes can be listed as:

- Sulphide ore flotation;
- Mixed ore flotation; and,
- Oxide leach, solvent extraction and electro-winning.

On all three circuits, Kansanshi mine utilises an open circuit semi-autogenous grinding (SAG) mill with a closed loop scats crusher and ball mill to maximise throughput. A process flow diagram for the each of the Kansanshi metallurgical circuits is presented in Appendices 3, 4 and 5 respectively.

This chapter presents a review of the metallurgical processing systems employed at Kansanshi mine. The primary purpose of this review is to develop an understanding how each circuit works. Further to this is the need to understand what ore characteristics can positively or negatively impact copper recovery. Through this review a hypothesis of *in-situ* ore characteristics was developed between the geology and metallurgy teams. This hypothesis would link *in-situ* geological features to copper recovery, the purpose being to define more appropriate copper recovery models. Work conducted to test this hypothesis is also presented in this chapter.

7.2 Sulphide ore

The current sulphide ore processing circuit at Kansanshi mine was commissioned in August 2008. It is considered to be the simplest metallurgical circuit used at Kansanshi mine. It comprises of a gyratory crusher followed by a secondary crusher which produces a crushed product from 65mm down to fines. A surge stockpile sits between the crushing and milling circuit. From this stockpile ore is fed by a conveyor to an open circuit grinding system. All ore reports to a semi-autogenous grinding mill (SAG) first. Scats are ejected from the mill at approximately 40mm diameter. They are directed to a pebble crusher that breaks them down to <10mm and they are then ground in a ball mill.

This circuit has the capacity to crush and grind 12 million tonnes of ore per annum. After grinding the ore slurry is pumped to a floatation system that uses sodium isobutyl xanthate (SIBX) as the main collector reagent. The floatation circuit comprises of a rougher bank of six $300m^2$ flotation cells. Secondary cleaning and tertiary scavenging occurs downstream before concentrate is thickened and filtered ready for transportation.

Reported copper recovery on this circuit is generally above 90%. This is attributed to the 0.05% ASCu limit on ore feed. This limit is designed to ensure that no tarnish or oxidation is present on the mineral surface. By ensuring the mineral surfaces are "clean" SIBX is able to bond effectively making the mineral surface hydrophobic. It then readily attaches to an air bubble and floats upwards to be recovered.

7.3 Mixed float ore

During the early stages of the mine's life a significant portion of mineralisation was directed to stockpile due a number of quality related aspects. Mixed float ore was defined as any sulphide ore that had a high degree of weathering. In August 2008 a new sulphide floatation circuit complete with gyratory crusher and mills was commissioned at Kansanshi mine. This new circuit was used to process sulphide ore. The original sulphide ore crushing, milling and floatation circuit was used to carry out metallurgical trials of different reagents in an effort to understand how best to process the stockpiled mixed float ore.

Through work by Paquot (2013) a method of effectively treating mixed float ore using a flotation process was determined. By adding sodium hydrogen sulphide (NaHS) at the early stage of flotation, a suite of secondary copper sulphides and some primary copper oxides could be recovered by floatation methods to produce a concentrate. As was reported by Freeman *et al.* (2000) after Orwe *et al.* (1997, 1998), the main purpose of the hydrosulphide ion was to re-sulphidise secondary sulphide minerals such as bornite, digenite and chalcocite, that had undergone partial oxidation. This re-sulphidisation resulted in a strongly hydrophobic mineral surface in the presence of a collector reagent.

Through the trials and subsequent phases of optimisation it was found that mixed ore floatation using the NaHS reagent was only effective if the ratio of AICu to TCu remained above 70%. To maintain a consistent feed to the mixed float circuit the MF_AICu ratio factor was incorporated into the Mat_Type system. However this factor was not applied to the ore classification process in an appropriate manner. From the review of the Mat_Type ore classification system presented in Chapter 6, it is evident that mixed ore is composed of any mineralisation that cannot be directed to the sulphide flotation or oxide leach circuits. Mixed ore on stockpiles and mixed ore that is being processed is composed of a range of different copper bearing minerals each with different metallurgical processing properties. As a result the MF_AICu ratio was ineffective and copper recovery in the mixed float ore circuit rarely exceeded 70%.

In addition to the poor recovery, the copper grade of the mixed float circuit concentrate is often below 20% Cu. Significant amounts of free floating gangue minerals are present in the ore. Typically the free floating gangue is composed of micaceous and carbon bearing clay minerals from the saprolite zone. These have formed from the weathering of the carbonaceous phyllite a lithology in the upper mixed clastics stratigraphic package.

Through the data reported in Chapter 4 it has been demonstrated that weathering processes and oxidation processes are not directly correlated. The mixed float ore product could be significantly improved through the combined use of oxidation domains and weathering domains as well as oxidation ratios, to define the ore product.

7.4 Oxide leach ore

The initial oxide ore processing circuit was designed to treat eight million tonnes of ore per annum. In 2012 this capacity was increased to 12.5 million tonnes per annum. A further expansion phase was commissioned in 2014 which increased the leach plant capacity to 14.5 million tonnes per annum. The third expansion phase was designed to allow tailings from the mixed float process which are generally high in ASCu to be treated through the leach. This would achieve two goals; primarily the need to use or neutralise the large quantity of sulphuric acid that would be produced from the soon to be commissioned smelter. Secondly, to create opportunity to realise additional revenue from leaching ASCu that was previously lost in the mixed float ore tails.

The oxide leach circuit is an extremely complex entity. Its primary function is to recover copper oxide mineralogy. At Kansanshi mine this mineralogy is typically a suite containing malachite, chrysocolla, tenorite and cuprite. All of these minerals are soluble (to one degree or another) in sulphuric acid. Once the mineral has dissolved a copper rich solution is formed. This solution is amenable to solvent extraction and electro-winning (SX/EW) processes. As was reported in Chapter 4, typical oxide ore mineralogy is not in a 100% ASCu form. That is to say that the mineral still contains a portion of AICu. This remnant AICu portion is generally found in the central area of the primary oxide mineral. This area at the centre has only been partially affected by the oxidation process to form secondary sulphide minerals.

In order to recover copper from this more complex mineralogical suite the oxide ore process contains an upfront floatation process. This process is designed to recover any remnant AICu bearing mineral grains that are exposed after milling. Any copper recovered in this process is directed to the final concentrate handling area where it is blended with concentrate from the other circuits before being dispatched to a smelter.

The tailings of the oxide floatation system are directed to the sulphuric acid leach plant which marks the start of the oxide leach ore processing phase. After leaching the ore in sulphuric acid a decantation process is used to separate the copper rich acidic solution from the tailings slurry. Further leaching can occur in the decantation process improving overall copper recovery. The copper rich acid solution is termed pregnant liquor solution (PLS).

The next phase in the oxide ore metallurgical process is solvent extraction. Copper contained in PLS is transferred to an electrolyte phase and in this process copper in concentrated from ~10g Cu per litre of H_2SO_4 to ~50g Cu per litre of H_2SO_4. The electrolyte is pumped through the electro-winning process where copper is extracted from the electrolyte by electroplating onto a stainless steel starting plate. The resultant cathode sheet is composed of pure copper at a grade of 99.99% Cu.

7.5 Recovery models

A key component of the mine planning process is the prediction of copper metal that will be made in a certain time frame. Traditionally the metallurgical department have provided recovery models which are applied to the outputs of the mine plan to calculate overall expected revenue. The actual metallurgical performance has rarely matched the model prediction. Copper recovery is generally lower than the metallurgical models would suggest. The standard metallurgical recovery model used at Kansanshi mine is based on a two product formula.

The copper head grade is measured at the mill cyclone overflow. The copper concentrate grade is measured after filtration and the tailings grades are measured by on-stream analysers. Recovery models for the sulphide floatation, mixed ore flotation, oxide floatation and oxide leach metallurgical processes have been derived by Van Der Merwe (2011) and are presented in Equation 7.1, Equation 7.2, Equation 7.3 and Equation 7.4.

Sulphide ore floatation recovery: **Equation 7.1**

$$TCu\ recovery = \left(\frac{(\ TCu\ feed - TCu\ tails\ )}{(\ 27 - TCu\ tails\ )}\right) * \left(\frac{27}{TCu\ feed}\right) * 100$$

Where: TCu recovery is the recovery of copper to concentrate.

TCu feed = Assayed TCu head grade at the mill cyclone overflow.

ASCu feed = Assayed ASCu head grade at the mill cyclone overflow.

TCu tails = $(0.0329\ *\ TCu\ feed)\ +\ (1.5132\ *\ ASCu\ feed)$.

Mixed ore floatation recovery: **Equation 7.2**

$$TCu\ recovery = \left(\frac{(\ TCu\ feed - TCu\ tails)}{(\ 27 - TCu\ tails\ )}\right) * \left(\frac{27}{TCu\ feed}\right) * 100$$

Where: TCu recovery is the recovery of copper to concentrate.

TCu feed = Assayed head grade at the mill cyclone overflow.

ASCu feed = Assayed ASCu head grade at the mill cyclone overflow.

TCu tails = $(0.0688\ *\ TCu\ feed)\ +\ (0.8\ *\ ASCu\ feed)$.

Oxide ore floatation recovery: **Equation 7.3**

$$TCu\ recovery = \left(\frac{(\ 0.04 *\ ASCu\ feed + 0.45 * AICu\ feed)}{(TCu\ feed)}\right) * 100$$

Where: TCu recovery is the recovery of copper to concentrate.

TCu feed = Assayed head grade at the mill cyclone overflow.

ASCu feed = Assayed ASCu head grade at the mill cyclone overflow.

AICu feed = Assayed AICu head grade at the mill cyclone overflow.

TCu tails = $(0.0688\ *\ TCu\ feed)\ +\ (0.8\ *\ ASCu\ feed)$.

Oxide leach (SX/EW) recovery: **Equation 7.4**

$$TCu\ recovery = \left(\frac{(0.6091 * 0.4066 + 120.992\ ASCu\ feed^{0.4043})}{(0.4066 + ASCu\ feed^{0.4043})}\right) + (AICu\ feed * 0.4)$$

Where: TCu recovery is the recovery of copper to cathode.

ASCu feed = Assayed ASCu head grade at the pre-leach thickener, after oxide floatation.

AICu feed = Assayed AICu head grade at the pre-leach thickener, after oxide floatation.

Through the geological and mineralogical data presented in Chapter 4 it is understood that the ore feed is composed of minerals that are dominated by either an ASCu portion or an AICu portion. From this conclusion a hypothesis for metallurgical recovery was proposed. This hypothesis was stated as:

> *If ore mineralogy is composed of minerals that are dominated by either AICu or ASCu and an in-situ geological grouping of mineralogy can be determined by oxidation ratio, can more appropriate metallurgical recovery models be generated that consider these three components of the ore feed?*
>
> (Dawson and Beaumont, 2014)

To test this hypothesis, historic mill head grade and recovery data for the period from 1st January 2009 to 31st December 2014 was captured. Point data for each metallurgical circuit was composited to produce daily head grade and associated copper recoveries. The data for each circuit was plotted as a Gaussian distribution to understand the mean and modal recovery points. A standard deviation was calculated for each data set and using two standard deviations about the mean outlying anomalous data was removed. Using the Python™ scripting language, a mean least squares regression model was derived to plot the ratio of ASCu to TCu and the ratio of AICu to TCu as a function of overall TCu recovery. The regression model is presented in Equation 7.5.

$$Recovery = \left(\left(a * \left(\frac{ASCu}{TCu}\right)\right) + \left(b * \left(\frac{AICu}{TCu}\right)\right)\right)$$ **Equation 7.5**

Where a and b represent coefficients of regression derived from the model.

7.5.1 *Recovery model for the sulphide flotation circuit*

The relationship between TCu grade and recovery a five-year time frame, from January 2009 to December 2014, is presented in Figure 7.1. There is a very broad general correlation between higher or lower TCu head grade and recovery. However the spread of data is so wide that periods of high grade, low recovery or low grade, high recovery can be identified along with the normal relationship that is expected of low grade, low recovery or high grade, high recovery.

The data set displayed in Figure 7.1 has been presented in Figure 7.2 to show the relationship between TCu and recovery. Typically at higher TCu grades the recovery is also high. It has been noted that the within the data analysed there is some inconsistency with this statement. It was found that the anomalous data points showing high TCu grade but poor recovery were a function of the ASCu content of the mil feed on that day.

Using the Python™ interactive development environment this data set was modelled using Equation 7.5. This allowed recovery to be plotted on a three dimensional axis as a function of ASCu and AICu. From this analysis two important trends have been identified which are presented in Figure 7.3.

Firstly; an increase in AICu content leads to small increases in recovery. Secondly; ASCu content has a major negative impact on recovery. Through the application of Equation 7.5 to this data analysis a regression plane has been created.

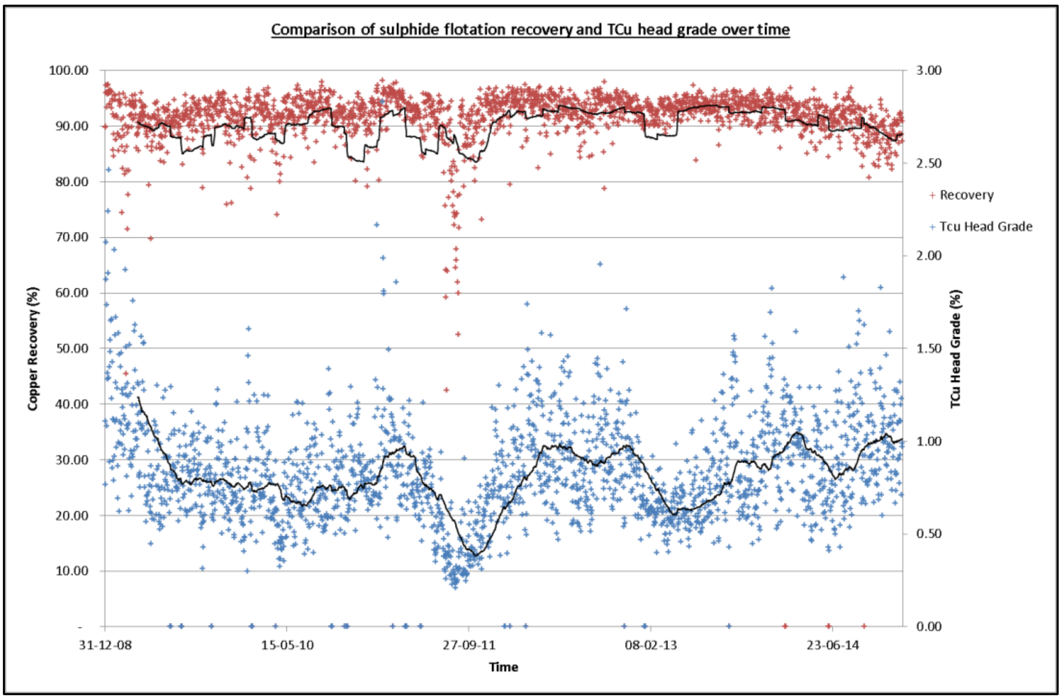


Figure 7.1 Comparison of TCu head grade to recovery for the sulphide ore processing circuit.

This regression plane gives the factors that can be used to predict copper recovery in the sulphide metallurgical plant for any range of ASCu and AICu grades. The recovery model for the sulphide circuit is presented in Equation 7.6.

Equation 7.6

$$Recovery = \left(\left(38 * \left(\frac{ASCu}{TCu}\right)\right) + \left(95 * \left(\frac{AICu}{TCu}\right)\right)\right)$$

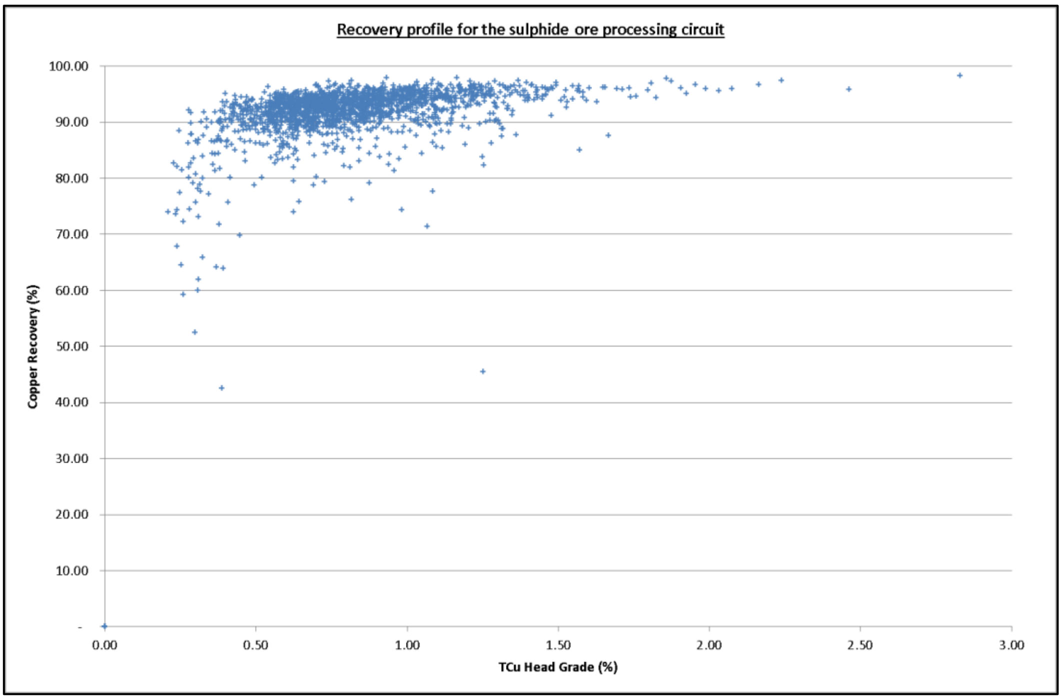


Figure 7.2 Recovery of copper as a function of the TCu head grade for the sulphide ore processing plant.

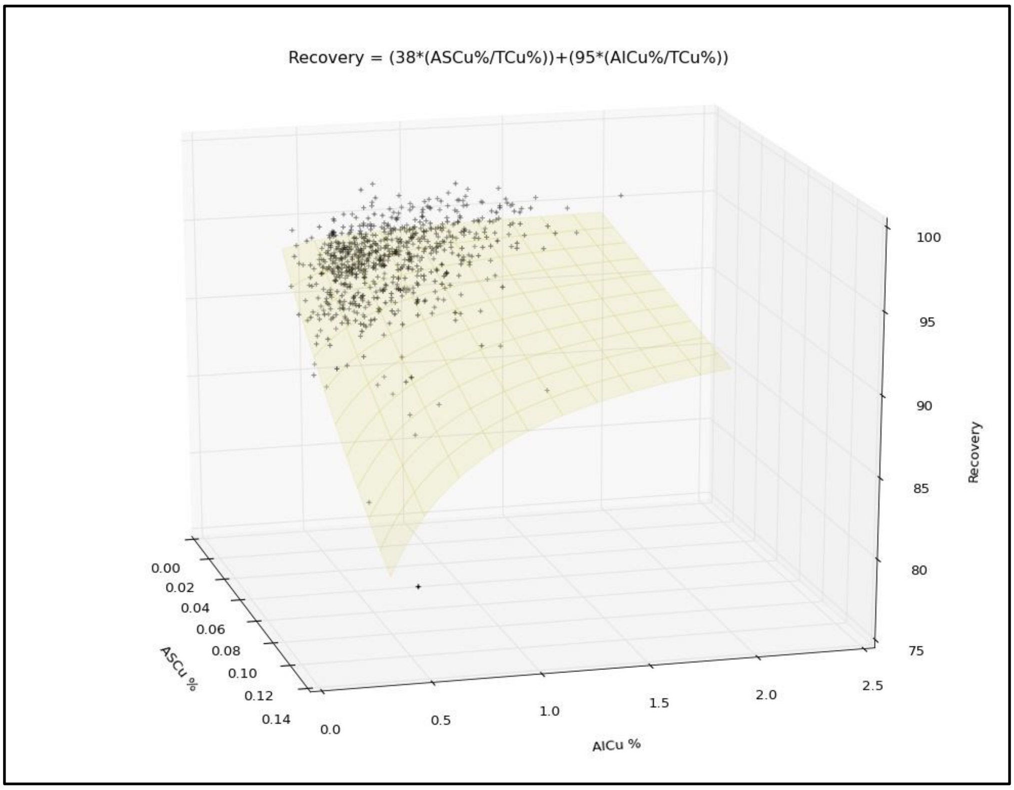


Figure 7.3 Recovery model for the sulphide float circuit based on ASCu and AICu content of the overall ore feed.

7.5.2 *Recovery model for the mixed float circuit*

As with the sulphide floatation circuit data presented in Section 7.5.1 a similar analysis was carried out for the mixed ore floatation circuit. Historical data for 2009 was not available as during this period the mixed ore floatation process was undergoing testing and commissioning. However data from January 2010 to December 2014 was available and used for this analysis. A graph of recovery and corresponding total copper head grade over time from the mixed float ore circuit is presented in Figure 7.4. Comparison of the moving average trend line for recovery and TCu show very little correlation. Overall the plant recovery is very erratic indicating that grade is not the main factor controlling recovery. In addition a noticeable change in recovery occurred between Quarter 2 and Quarter 3 of 2012. It is unclear what was responsible for this change.

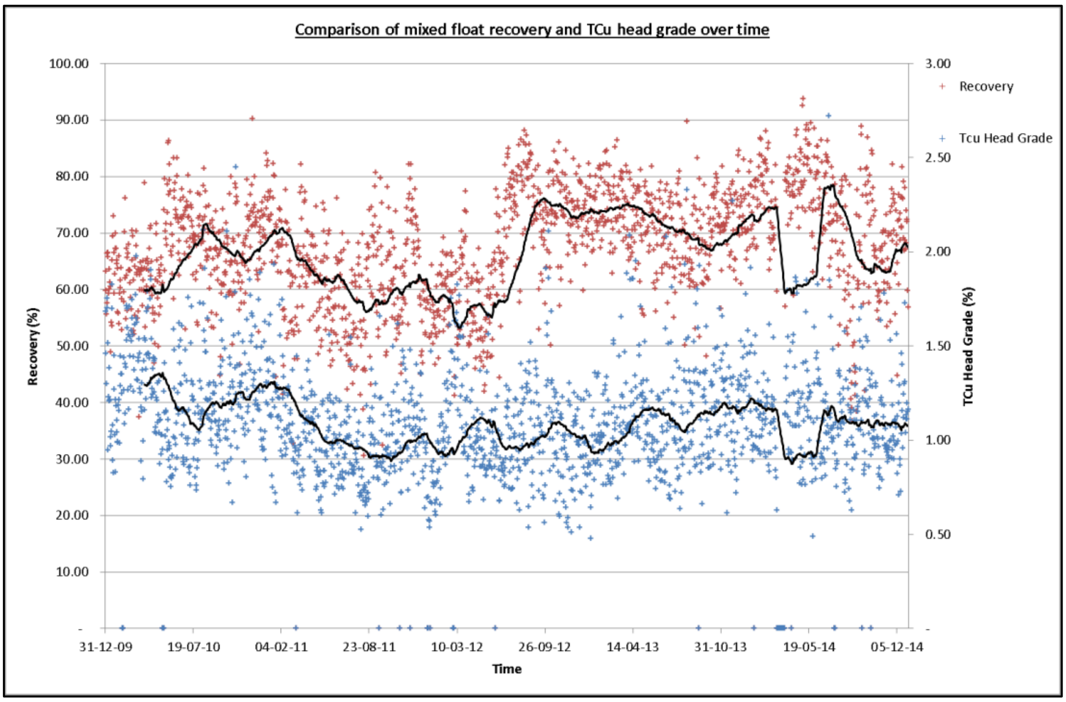


Figure 7.4 Comparison of TCu head grade to recovery for the mixed float ore processing circuit.

By modelling the recovery as a function of the AICu and ASCu components of the TCu head grade using Equation 7.5 a three dimensional recovery model was created. This model has been presented in Figure 7.5. A second view of the model to show the impact of increasing ASCu grade on recovery has been presented in Figure 7.6. A recovery model using the regression coefficients from this analysis is presented in Equation 7.7.

$$Recovery = \left(\left(30 * \left(\frac{ASCu}{TCu}\right)\right) + \left(87 * \left(\frac{AICu}{TCu}\right)\right)\right) \qquad \textbf{Equation 7.7}$$

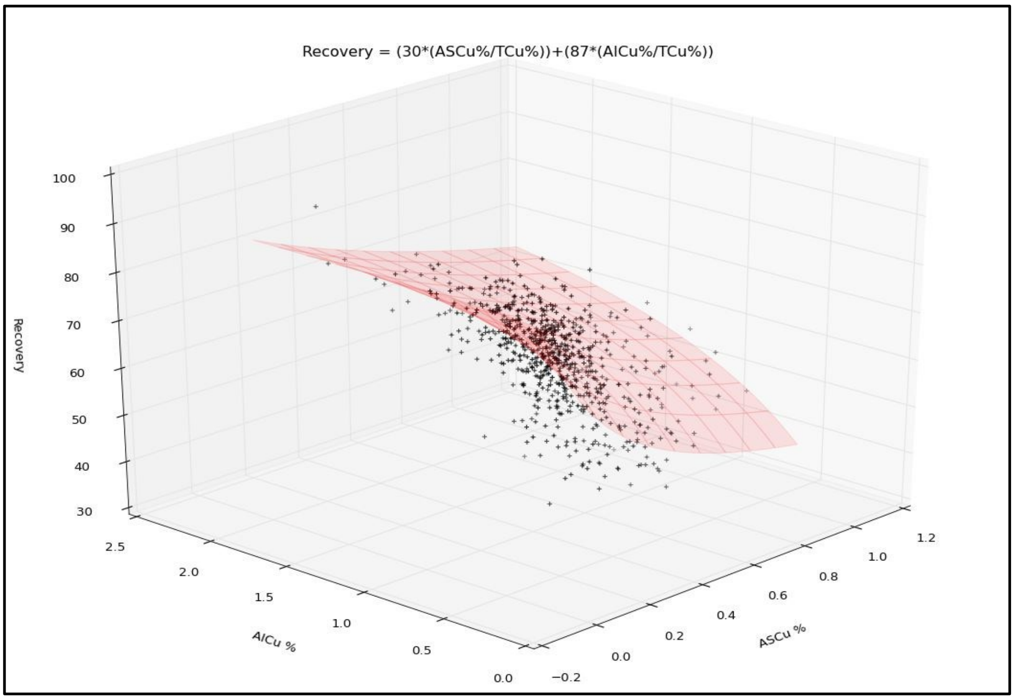


Figure 7.5 Recovery model for the mixed float circuit based on ASCu and AICu content of the overall ore feed.

It can be seen that ASCu plays a major role in determining copper recovery for the mixed float circuit. However increasing AICu grade only marginally impacts the recovery. There is a wide scatter of data points indicating that in addition to ASCu and AICu there are other factors impacting the recovery in this circuit. Work by Paquot and Nugulbe (2015) stated that iron oxides and hydroxides on the surface of chalcopyrite and chalcocite particles negatively impacted their ability to float. As was noted in the microscopy work presented in Chapter 4 gangue Fe oxides will have formed as a result of *in-situ* oxidation processes.

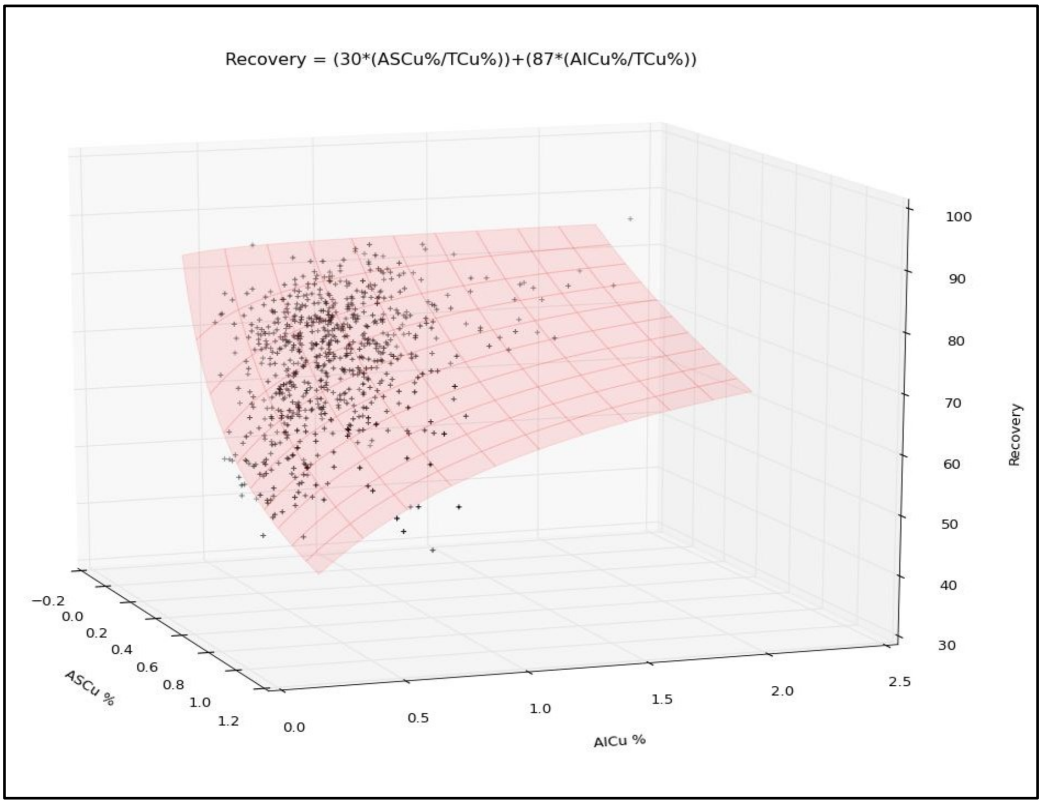


Figure 7.6 Mixed float recovery model rotated to show the impact of increasing ASCu grade on recovery.

7.5.3 *Recovery model for the oxide float circuit*

The oxide float circuit is designed to recover any portion of AICu that may remain in the oxide ore. As it is not the main focus of the oxide ore processing methodology the residence time is often reduced significantly in order to quickly move the ore feed through to the leaching part of the process. Recovery in this circuit is extremely erratic. It was not possible to derive a meaningful data set that could be used to model recovery.

A note is made that significant value can be obtained from the AICu portion of the oxide ore. Ensuring the oxide float circuit is well operated will assist in recovering additional value.

7.5.4 *Recovery model for the oxide leach circuit*

The oxide leach plant recovery was modelled in a similar manner to the floatation plants. The recovery coefficients used are presented in Equation 7.8 while the resulting recovery model is presented in Figure 7.7. The ASCu grade is the dominant variable in controlling the recovery profile of this circuit, while an increase in AICu grade presents only a marginal decrease in recovery.

$$Recovery = \left(\left(90 * \left(\frac{ASCu}{TCu}\right)\right) + \left(58 * \left(\frac{AICu}{TCu}\right)\right)\right) \qquad \textbf{Equation 7.8}$$

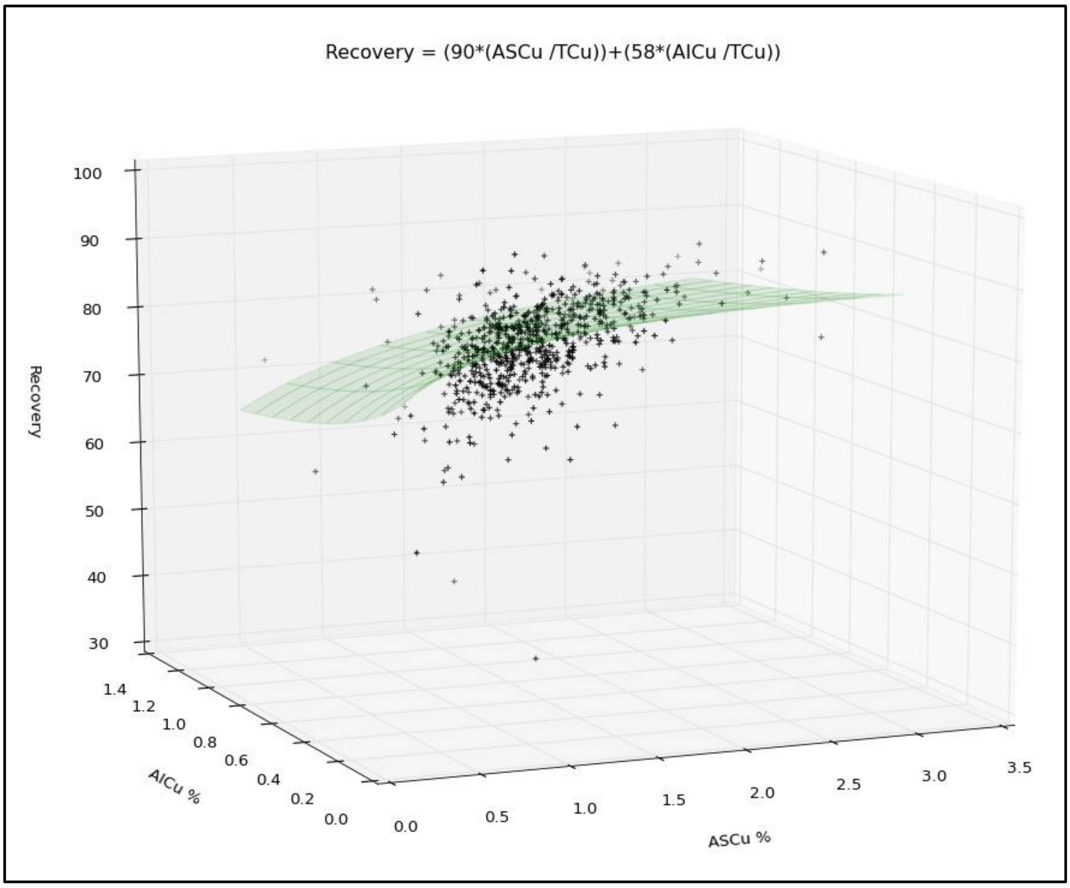


Figure 7.7 Oxide leach plant recovery model.

There is a broad spread of data points in the oxide leach plant recovery model. Work by Paul (2015) has highlighted that a number of copper oxide minerals present in the leach feed are not amenable to leaching processes most notably the copper silicate chrysocolla. Additionally, chalcocite can leach under certain conditions. The traditional mineralogical division has always been based on the assumption that all primary oxide copper minerals are acid soluble. Recent mineralogical work has indicated that secondary sulphide and primary oxides have a range of "leach-ability". That is to say that the ability of sulphuric acid to leach Cu from the mineral ranges from very fast leaching to slow leaching to un-leachable.

To accurately determine copper recovery in the leach circuit it is important that a quantitative understanding of the mineral species reporting to this circuit is made available. The required level of mineralogical detail is not available at a geological scale. However, as reported in Chapter 4 domains of malachite dominant or chrysocolla dominant mineralisation can be determined but this is a more qualitative data set that can be used to communicate to downstream stakeholders a suitable expectation for this ore's metallurgical performance.

7.6 Data integrity

Through the work presented in Section 7.5 it was found that the models did not correlate well with the existing recovery models or the historic calculated recoveries. In general the lower quartile of the data range for each model tended to over predict recovery when compared to the actual recovery. Similarly in the upper quartile of the data range the model tended to under predict recovery when compared to the actual recovery. In the interquartile range, correlation between predicted and actual recovery improved towards the median point. An investigation was carried out into why this would be so. This investigation highlighted three important points:

- Firstly; the size of the range of data used to create the predictive model had a major impact on the coefficients of regression.

Smaller ranges (covering shorter time frames) could be used to more accurately predict a recovery within that specific range. As the time frame that the data population covered was increased, the upper and lower ends of the recovery prediction became more diverse from the actual calculated value.

- Secondly; the plant data used in the analysis had been subjected to a number of internal factors and back calculations these included but were not limited too; moisture factors, density conversion factors for mill feed, concentrate production and tailings slurry. The data available had been modified and did not provide a true representation of the entities that were being modelled.
- Thirdly; the highly variable nature of the ore feed as a result of inappropriate ore classification added another dimension to the variability of the data. Isolated cases where a lack of correlation between head grade, oxidation ratio and recovery would skew the models' ability to accurately predict recoveries based on the hypothesis.

Going forwards if these models were to be implemented further testing is required. Such testing will need to be completed over a suitable time frame to build up a meaningful population of data that is statistically relevant. This data population will need to be gathered in such a way as to ensure that the actual feed and recovery data collected is in a raw format, which has not been subjected to any modification. To achieve this, a more robust metal accounting and reconciliation system is required. Finally if practically possible, the test phase must be based on an ore feed that is geologically consistent. By minimising variability in the ore feed greater consistency can be achieved that will narrow the data range in the predictive models.

7.7 Chapter summary

Kansanshi mine has a large and complex metallurgical process plant that is capable of recovering copper from a broad range of ore types and mineralogy. Many phases of expansion and optimisation have occurred in this plant but they have often been done insolation of the overall mine value chain. This has led to the development of inappropriate assumptions and factors that have then been applied to other areas of the business without a suitable level of understanding.

A hypothesis was proposed that more appropriate predictive recovery models could be developed that would determine a recovery based on the ASCu, AICu and TCu content of the ore feed. This hypothesis has been proven in part but can be refined through the implementation of a more robust metal accounting and reconciliation system that removes a number of factors and back calculation steps which modify plant data.

In Chapter 8, a discussion of the key findings presented in pervious chapters of this report is given. This discussion shows where more appropriate use of geological economic and metallurgical data can be achieved. The result of this discussion is the creation of a more appropriate and robust ore classification system which incorporates a geometallurgical strategy. This system will drive integration between mine planning, grade control and mineral processing.

8.0 INTEGRATION OF GEOLOGICAL AND METALLURGICAL DATA.

8.1 Introduction

Through a systematic review of data from the three main technical silos at Kansanshi mine it has been demonstrated that a number of decisions and processes that are made on a daily basis are no longer appropriate. Given the number of errors and modifications that are required to make the Mat_Type system classify ore more appropriately, one must consider if it is practical to continue with its use. Through the work presented in this report there is an opportunity to develop a much more relevant ore classification system that is supported by the analysis of robust data.

Improved understanding of geological processes that control oxidation and weathering, along with better understanding of the complex mineralogical associations, need to be integrated into the overall business strategy. Through this integration it will be possible to group mineralisation into more meaningful ore types for metallurgical processing. These groupings would be based on similar geological and mineralogical characteristics, as well as grade ranges that can be used to blend and maintain head grade consistency in the metallurgical process. An appropriate ore classification system must have a functionality to complete the following tasks:

1. Identify all blocks with sufficient geological data to be accurately estimated with a copper grade;
2. Identify blocks without sufficient geological data to be accurately estimated with a copper grade;
3. Apply grade limits to group ore blocks based on an economic evaluation of the mining and processing costs as well as copper prices mineral royalty taxation and downstream beneficiation charges;

4. Determine a suitable metallurgical processing system for all of the ore blocks, based on a combination of geological and metallurgical factors;
5. Determine a suitable destination for all blocks that cannot report to the processing plant; and,
6. Assign an appropriate code to each block in the geological model thus making it easy to identify the factors listed above.

This chapter presents a method whereby data and understanding can be integrated effectively and communicated across the key departmental silos so far encountered. This method is designed to break down the silo walls and place integrated decision making at the core of the business. When implemented it would be the central strategy to guide all mine planning decisions, in both short term daily production and longer term mine planning.

8.2 Understanding what is ore and what is waste

Kansanshi mine utilises a grade control model for short term mine planning (<18 months). This grade control model is based on a combination of diamond drilling and close spaced RC drilling. It is a very detailed model with a high density of data but only covers small areas of the deposit which are currently active.

As noted in Chapter 6, not every block in the grade control model has sufficient data to be given an estimated copper grade. Blocks that do not have sufficient data for estimation are assigned a background TCu value of -99. This background value distinguishes these areas from those that do have sufficient data for an estimated grade. The geo-statistical parameters used in the estimation process for the Kansanshi deposit determine that a minimum of five samples, within the dimensions of the search ellipsoid, are needed to confidently estimate a TCu or ASCu grade. During the estimation process an attribute called "tcu_nsamp" is created in the geological model that records the number of geological samples used to estimate that block. The geological models are updated weekly with new assay and logging data. During each data update a re-estimation of the available data is carried out. From this re-estimation the tcu_nsamp attribute is also automatically updated in the appropriate blocks. By using the tcu_nsamp attribute in the first stage of the ore classification process, a level of confidence can be applied, which will ensure that only those blocks with sufficient geological data are classified with an ore category. Any block in the geological model that does not meet the tcu_nsamp criterion would be coded as "0". This would indicate that the block is currently "unclassified". Unclassified blocks cannot be mined until further geological work has been completed. Mine planners and geologists would be able to use this data for two key functions.

Firstly, they can ensure that potential ore is not thrown away by simply assuming it is waste. This will provide a mine planner with confidence that potential value is not lost through poor decision making. Secondly, the attribute highlights areas in the geological model where further geological work needs to be planned to gather sufficient data to support the estimation process. This will improve collaboration between mine planners and geologists in developing plans to access working areas of the pit to gather the necessary data to update the geological model in a timely fashion and ultimately support the mine plan.

Incorporating the tcu_nsamp attribute data into the ore classification system can be achieved with an IF statement that is coded into the geological model. An example of which is presented in Figure 8.1. The IF statement would query the tcu_nsamp attribute in a particular block to determine how many samples have been used to estimate the TCu or ASCu grade. If the number is less than five then the block is considered to be unclassified while greater than five would mean that block can pass through to subsequent stages of the ore classification process.

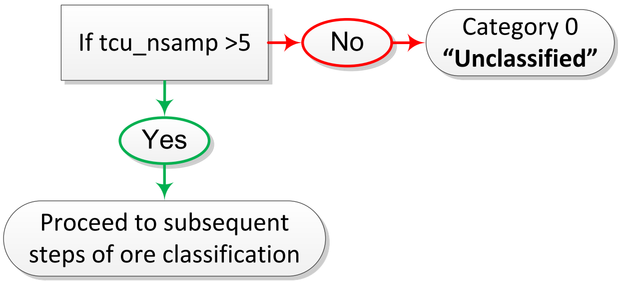


Figure 8.1 Using the "tcu_nsamp" block model attribute to give confidence in the ore classification process.

8.3 Using oxidation ratios to assist in defining ore groupings with similar metallurgical characteristics

The current Mat_Type ore classification system does not take into account the impact of mineralogy on mineral processing. Chapter 4 discussed the ore mineralogy found in the deposit. Mineralogical groupings that naturally occur can be identified by the oxidation ratio. Chapter 7 discussed the metallurgical process plant and the various mineral processing methods available to treat the wide variety of mineralogy (both gangue and economic) that is present in the Kansanshi ore. By using the oxidation ratio ranges reported in Chapter 4 as a second step in the ore classification methodology it would be possible to direct any block of ore in the geological model to an appropriate mineral processing circuit. An example of a simple IF statement which uses oxidation ratio to determine a suitable mineral processing method is presented in Figure 8.2.

By using the oxidation ratio in ore classification, the current MF_AICu_Ratio would become defunct. The MF_AICu_Ratio currently only applies to sub categories in the mixed float ore group. The oxidation ratio would apply to all ore which is much more appropriate given current geological and metallurgical understanding.

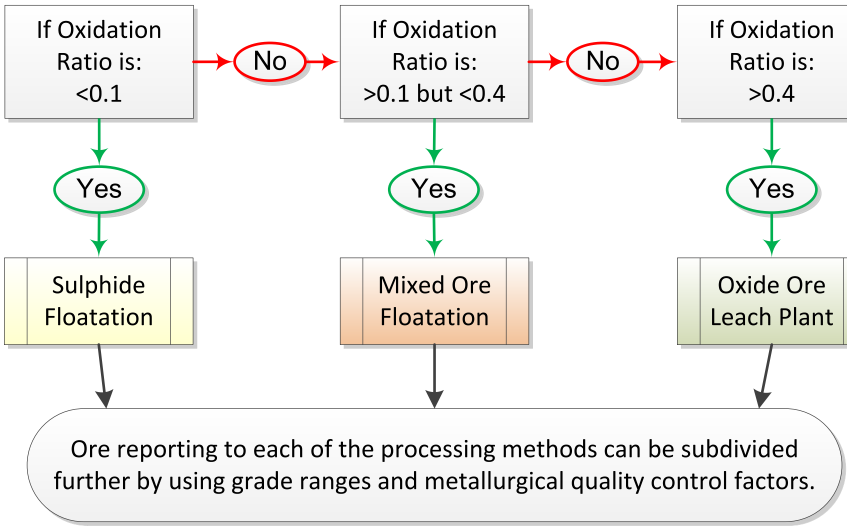


Figure 8.2 Using oxidation ratio to determine a suitable metallurgical process.

8.4 Correct application of cut-off grades

As reported in Chapter 6 cut-off grades created and used by the Mat_Type system have a number of errors. As a result the calculated cut-off grades are overwritten by a set of manual inputs. The errors that have been identified can be summarised as:

- Out of date data;
- Inadequate data;
- Inappropriate use of data in the cut-off grade calculations; and,
- Inappropriate application of the calculated cut-off grade to AICu or ASCu independently of TCu.

The creation of a geometallurgical strategy to drive ore classification and mineral processing efficiency must have well defined grade thresholds that assist in determining the mineralisation that is processable and the mineralisation that should be stockpiled. It is these financial data inputs that form the foundation of a mineral resource to mineral reserve conversion process. As a result of the findings presented in Chapter 6 a review of the financial inputs to the ore classification process was completed.

8.4.1 *A review process to understand the available economic data*

Through collaboration with Kansanshi mine financial department a cut-off grade calculation model was built using the Microsoft Excel software. To populate the cut-off grade model with suitable data the Kansanshi mine five-year operating budget was interrogated to determine appropriate fixed and variable costs. This data is presented in Table 8.1. Further interrogation of the five-year operating budget provided net return data that could be used to calculate the cost of producing the final copper product from a particular metallurgical process. The SX/EW process produces copper cathode which is considered to be a finished product and does not need any downstream beneficiation. It therefore has a different net return value to the concentrate produced by the other metallurgical processes. Net return data for the three metallurgical process routes is presented in Table 8.2.

The sulphide ore flotation process and mixed ore floatation process both produce a copper concentrate that is blended together and sold to an off-site smelter for further beneficiation. However the two metallurgical processes operate at different throughput rates and have different average recoveries. These differences lead to variations in the fixed and variable costs and differences in the calculated cut-off grades.

The level of detail presented in Table 8.1 and Table 8.2 is higher when compared to the cost data inputs presented in Figure 6.1. As a result of this data analysis a set of calculated cut-off grades was produced by applying this data to Equation 6.1. A comparison of the correctly calculated cut-off grade and the original Mat_Type cut-off grades are presented in Table 8.3. A further comparison between the cut-off grades developed from the 2015 budget review data and the cut-off grades created by the life of mine optimisation process are presented in Table 8.4. Both comparisons presented show that the cut-off grades calculated using the current financial data set are higher than the historic grades applied to the Mat_Type system and also higher than the grades created by the life of mine optimisation scenario.

Table 8.1 Kansanshi mine cut-off grade model, fixed and variable cost data entry form.

Oxide ore leach, solvent extraction and electro-winning		
Fixed Costs:		
Mining	$/t process	0.30
Engineering	$/t process	3.17
Other Direct	$/t process	0.28
Administrative Site	$/t process	0.24
Labour	$/t process	2.95
C&M	$/t process	0.01
Gold	$/t process	0.01
Sub-total fixed	**$/t process**	**6.97**

Variable Costs: Crushing milling and leach		
Ore mining differential/GRADE CONTROL	$/t process	0.02
Crusher Feed Cost (FQMO)	$/t process	1.66
Rehandel	$/t process	-
Plant - direct consumables	$/t process	1.35
Plant - power (Crushing & Milling)	$/t process	1.24
Plant - other	$/t process	-
Sub-total variable	**$/t process**	**4.26**
	Total fixed + variable	**11.23**

Mixed ore floatation		
Fixed Costs:		
Mining	$/t process	0.33
Engineering	$/t process	1.23
Services	$/t process	0.31
Administrative	$/t process	0.27
Labour	$/t process	2.10
C&M	$/t process	0.02
Gold	$/t process	0.01
Sub-total fixed	**$/t process**	**4.27**

Variable Costs:		
Ore mining differential	$/t process	0.02
Crusher Feed Cost (FQMO)	$/t process	1.82
Rehandel	$/t process	-
Plant - direct consumables	$/t process	2.16
Plant - power (Crushing & Milling)	$/t process	0.52
Plant - other	$/t process	-
Sub-total variable	**$/t process**	**4.52**
	Total fixed + variable	**8.79**

Sulphide ore floatation		
Fixed Costs:		
Mining	$/t process	0.49
Engineering	$/t process	1.23
Services	$/t process	0.47
Administrative	$/t process	0.41
Labour	$/t process	3.21
C&M	$/t process	0.02
Gold	$/t process	0.02
Sub-total fixed	**$/t process**	**5.85**

Variable Costs:		
Ore mining differential	$/t process	0.03
Crusher Feed Cost (FQMO)	$/t process	2.75
Rehandel	$/t process	-
Plant - direct consumables	$/t process	1.07
Plant - power (Crushing & Milling)	$/t process	1.23
Plant - other	$/t process	-
Sub-total variable	**$/t process**	**5.08**
	Total fixed + variable	**10.92**

Table 8.2 Kansanshi mine cut-off grade model, net return data entry form.

Return: Cathode		
Labour	$/t Cu cathode	36.94
Engineering	$/t Cu cathode	46.49
Plant	$/t Cu cathode	192.55
EW power	$/t Cu cathode	239.32
HPL	$/t Cu cathode	43.35
Transport and Freight	$/t Cu cathode	238.42
Premium cathode	$/t Cu cathode	(91.70)
Recovery	%	88.00
Gold credit dore (c per lb of final product)	c/lb	(0.14)
Cathode Royalty	$/t Cu Metal	476.20
Sub-total - (KMP)	$/t Cu Cath	1,178.56

Metal price and Royalty data:

copper price ($/t)		5,291.09
copper price ($/lb)		2.40
Royalty %age		9.00
Pounds to Metric tonne conversion		2,204.62
Mine Dilution Factor	%age	110.00

Return: Concentrate		
Concentrate Grade	% Cu	23.21
Anode Grade	% Cu	99.76
Moisture Content	%	8.00
Final Con product handling	$/t conc	8.23
Anode Freight	$/t anode	49.11
Smelting charge	$/t conc	70.94
Refining charge	c/lb	6.89
Recovery	%	70.00
Gold credit (c per lb of anode sold)	c/lb Cu Metal	(0.15)
Gold credit dore (c per lb of final product)	c/lb Cu Metal	(0.14)

Concentrate Transport (wet)	$/t Cu in conc.	38.52
Treatment Cost (wet) [Smelting]	$/t Cu in conc.	332.24
Refining Charges	$/t Cu in conc.	0.62
Realisation and Freight (ANODE)	$/t contained Cu	49.23
Anode Royalty	$/t Cu Metal	476.20
Au in Anode: Credit per tonne of anode	$/t Cu Metal	(3.32)
Au in Dore: Credit per tonne of anode	$/t Cu Metal	(3.01)
Subtotal - (*smelter return*)	**$/t Cu in conc.**	**890.48**

Return: Concentrate		
Concentrate Grade	% Cu	23.21
Anode Grade	% Cu	98.00
Moisture Content	%	8.00
Final Con product handling	$/t conc	8.23
Anode Freight	$/t anode	49.11
Smelting charge	$/t conc	70.94
Refining charge	c/lb	6.89
Recovery	%	92.00
Gold credit (c per lb of anode sold)	c/lb Cu Metal	(0.15)
Gold credit dore (c per lb of final product)	c/lb Cu Metal	(0.14)

Concentrate Transport (wet)	$/t Cu in conc.	38.52
Treatment Cost (wet) [Smelting]	$/t Cu in conc.	332.24
Refining Charges	$/t Cu in conc.	0.62
Realisation and Freight (ANODE)	$/t contained Cu	49.11
Anode Royalty	$/t Cu Metal	476.20
Au in Anode: Credit per tonne of anode	$/t Cu Metal	(3.32)
Au in Dore: Credit per tonne of anode	$/t Cu Metal	(3.01)
Subtotal - (*smelter return*)	**$/t Cu in conc.**	**890.37**

Table 8.3 A comparison between the Mat_Type cut-off grades and the grades produced by a review of the 2015 five-year operating budget.

Metallurgical Process	Mat_Type	2015 Review
Oxide leach SX/EW	0.4%	0.34%
Mixed float	n/a	0.33%
Sulphide float	0.25%	0.31%

Table 8.4 A comparison between the short term and life of mine cut-off grade.

Metallurgical Process	2015 Review	LOM Optimisation
Oxide leach SX/EW	0.34%	0.28%
Mixed float	0.33%	0.26%
Sulphide float	0.31%	0.19%

In the past cost data applied to the Mat_Type system had been left for many years without any form of a review. As a result of the process outlined in this section a cross functional relationship between the mine technical services department and financial department has been forged. Periodic review of available financial data within the context of reviewing the cut-off grade can now become part of the annual budgeting and mine planning process. Due to the current geo-economic and geo-political situation, the mining industry is subject to ever increasing levels of volatility in operating costs, taxation and commodity prices. To continue to operate effectively and profitably it is essential that the economic foundation of the business plan is clearly established, understood by all stakeholders and reviewed as and when a major change occurs.

8.4.2 *Application of cut-off grade data in an ore classification system*

The cut-off grade model that has been developed in Microsoft Excel can be applied to a geological model in Surpac by developing a .tcl script which recreates the data inputs and calculations. By creating a cut-off grade generator that can be applied to any geological model ownership of mineral resource to mineral reserve conversion process remains within the company and removes the need for external consultants to do the work on the company's behalf. However in order to apply cut-off grades to the geological model appropriately a choice must be made as to what grades should be used.

The data made available from the review of the 2015 operating budget can be considered to be more realistic of the mine's operating costs. In the life of mine optimisation scenario certain cost elements are considered to be optimistic and based on analysis of potential future risks and rewards or future business developments. As a result the calculated life of mine cut-off grade for each ore stream is much lower than the five-year budget calculated cut-off grades.

It is possible to use the two sets of data and their respective calculated cut-off grades to define economic mineralisation as well as marginal grade mineralisation and waste. If one considers the life of mine cut-off grade to be the absolute minimum grade at which any ore can be considered to be economic then this grade can be used to determine the boundary between waste (below) and potentially economic mineralisation (above).

One can then consider the cut-off grade calculated from the five-year budget as the minimum grade that mineralisation in the deposit is economic under current circumstances. This cut-off grade would mark the boundary between currently economic ore (above) and marginal ore (below). Marginal ore would have an upper grade set by the five year budget calculated cut-off grade and a lower grade defined by the life of mine cut-off grade. Marginal ore could therefore be defined as mineralisation that is currently uneconomic, however over the life span of the mine this mineralisation may have the potential to be economic, given a change in financial assumptions or cost data inputs.

Due to the variability of grade in the Kansanshi deposit it would be necessary to evaluate the range of grade available that is above the five-year budget calculated cut-off grade. In order to maintain a level of practicality in the daily mining and plant feed a further cut-off grade would be needed in this range to distinguish higher grade ore from lower grade ore. Through analysis of grade tonnage curves one could ascertain the average grade above cut-off and use this as a high grade to low grade ore boundary. This would allow the mine planner to schedule blending of higher grade and lower grade ore during daily operations to meet a specific grade range determined by the metallurgical process plant. A process of periodic review of all data used in the creation of the three cut-off grades proposed would be essential to ensure that they remain current and valid. An example of the application of this cut-off grade methodology for the sulphide ore portion of the deposit has been presented in Figure 8.3.

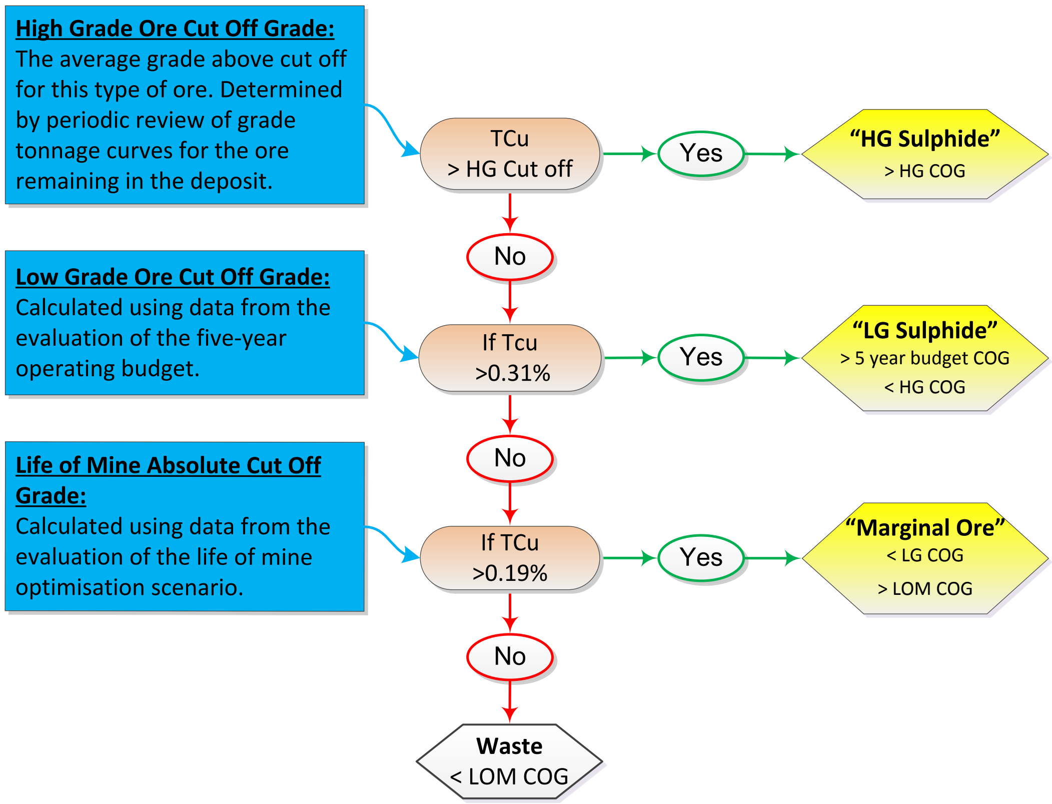


Figure 8.3 An example showing the application of short term and long term calculated cut-off grades in the ore classification process.

8.5 Improve the application of GAC data to leachable ore classification

Through the analysis presented in Chapter 5 and Chapter 6 it is clear that GAC data is not being used appropriately in the current ore classification process. This is apparent both in the use of GAC data as part of the calculation of the AOR factor and in the use of GAC as a standalone ore classification entity. As mentioned in Chapter 1 an abundant source of cheap sulphuric acid will soon be available from the Kansanshi smelter. It is therefore necessary that the mine now develops a strategy to consume this acid. Furthermore a strategy is needed to effectively utilise the expanded oxide leach plant capacity that has been recently installed.

Improved communication of the available GAC data and the geological context that controls this data across silo walls is critical. By developing the understanding of what GAC is, improvements can be made as to how GAC data is applied and interpreted in subsequent parts of the mine value chain. This will allow for optimisation of acid consumption in the mine plan that will ultimately drive aspects of the leachable ore mineral processing strategy. The opportunity to process higher GAC value leach ore rather than stockpiling it achieves a number of strategic goals. These goals can be listed as:

- Realisation of value that is currently lost to stockpile;
- The consumption of smelter acid;
- A reduction in overall leachable ore processing costs through the use of cheaper smelter acid; and,
- A reduction in unit cost per tonne (US$/t) of leachable ore processed through increased plant throughput.

As a first step to realising these benefits the AOR factor should be removed from the current ore classification method. Furthermore the GAC limits currently applied in the Mat_Type system should be modified from the current fixed limit of 50kg/t to a limit that is more representative of the *in-situ* geology. A high GAC ore category would consist of leachable ore that is in proximity to marble lithologies, while a low GAC ore category would consist of leachable ore that is in proximity to clastic units.

This will create a much more simple classification system that can support the mine planning process to determine suitable leachable ore feed. GAC can then be used as a true quality control factor on leachable ore to assist in the planning and management of acid consumption. An example of a more appropriate leachable ore classification using GAC as a quality control factor is presented in Figure 8.4.

By removing the AOR factor, four ore bins in the current Mat_Type system would no longer be necessary (32, 34, 52 and 64), while a further two (54 and 62) would be modified. As a result leachable ore that is currently being downgraded to long term stockpiles due to its AOR value would report to the remaining five possible leachable ore bins; 12, 13, 14, 22 and 24.

Through the data analysis presented in Chapter 5 there is an opportunity to modify the GAC threshold used to classify leachable ore by high and low GAC content. By applying the GAC cut-off more appropriately it would be possible to remove a further bin (13) leaving only four possible ore bins for leachable ore. This is a significant and practical simplification from the current options available for ore classification. In total this method would create four leachable ore categories to assist in head grade management and acid management. These categories can be listed as:

- High grade, good quality low GAC ore - 14;
- Low grade, good quality low GAC ore - 12;
- High grade, poor quality, high GAC ore - 24; and,
- Low grade, poor quality, high GAC ore - 22.

This classification would align the application of GAC data in ore classification to the application of GAC data in the geological model. Through this, the mine geology team would be able to use mapping and logging data to better inform the geological model and the mine plan of areas of high or low GAC leachable ore. This would then inform the mine plan and translate to a practical set of ore types for mineral processing.

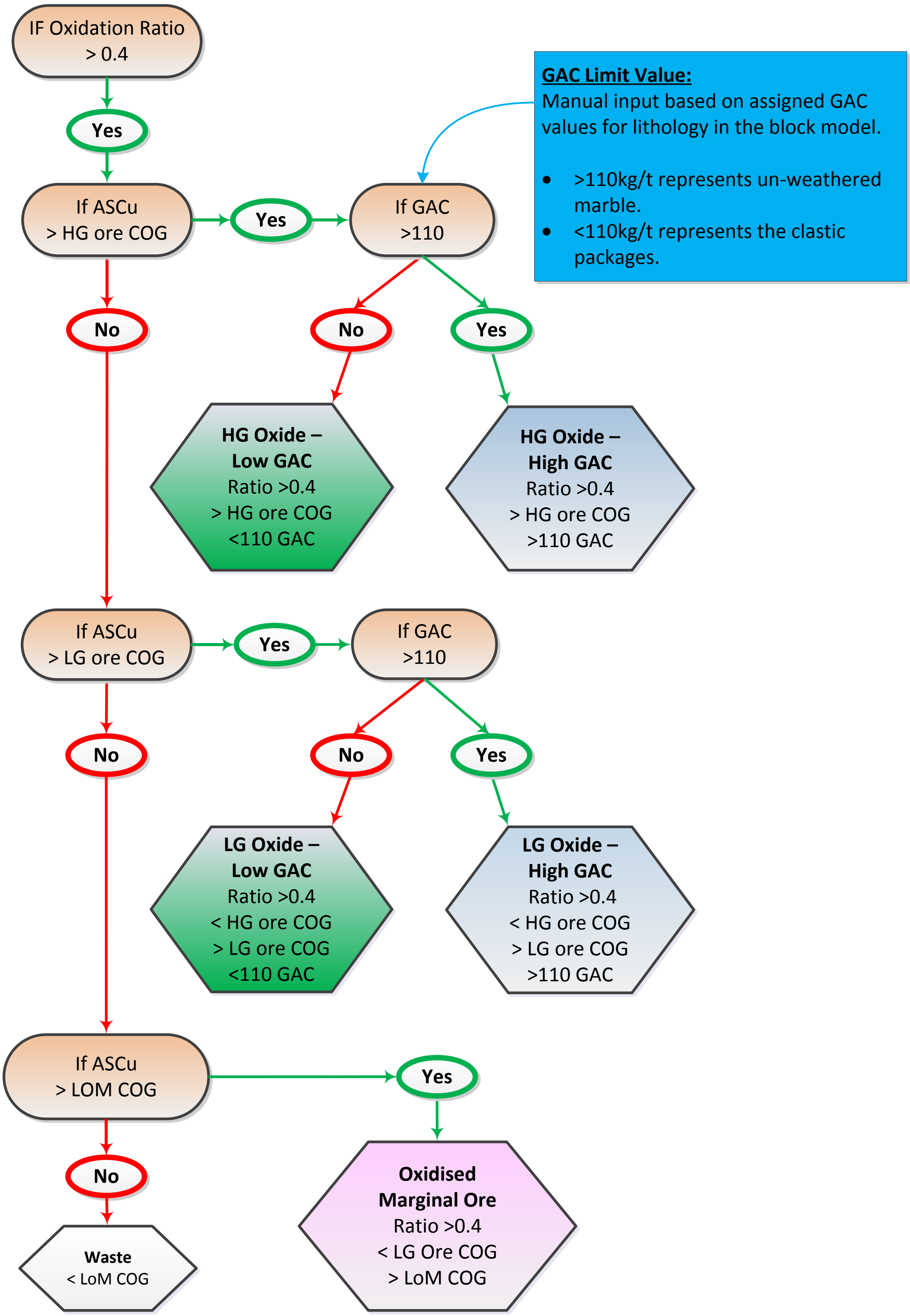


Figure 8.4 Using GAC data to appropriately to sub-divide leachable ore categories.

8.6 Classifying and understanding refractory mineralisation

It was noted in Chapter 6 that substantial work has been completed over the past five years to define refractory mineralisation. It was further noted that the associated developments in understanding of refractory mineralisation have not yet been incorporated into the ore classification process. The Redoxn attribute is now a standard data set in all geological models. The domain is well defined through geological mapping, logging, assaying and application of the criteria listed in Section 6.7. Any mineralisation that falls within a refractory domain is identified by the Redoxn attribute with a numerical code of four (4) in the geological model.

As with the tcu_nsamp attribute it is possible to apply the Redoxn data to ore classification. Again IF statements can be used to query data in the geological model and determine if mineralisation within a refractory domain is suitable for a specific mineral processing system. Typically refractory mineralisation affects the sulphide floatation system. An example of an appropriate IF statement to identify refractory mineralisation and separates it away from any ore category is presented in Equation 8.1.

Equation 8.1

$$If\ (oxidation\ ratio < 0.1\ and\ redoxn = 4)\ \ set\ ore\ code\ to\ 71\ (refractory)$$

8.7 Developing a new geometallurgical ore classification system

Throughout Chapter 8 a number of improvements have been suggested to simplify or optimise the ore classification system. However each of these improvements has so far been presented as a standalone entity. It has been demonstrated through the work presented in earlier chapters of this report that the entire mine value chain at Kansanshi mine is often composed of a set of defined silos that do not integrate upstream or downstream. Ore classification systems are integral to driving value along the mine value chain. As such it is imperative that an ore classification system is based on a holistic approach that ensures the flow of value from its *in-situ* location to the final finished product. Using geometallurgical principles, knowledge from across the entire mine value chain can be integrated to create a robust and appropriate ore classification system to assist in mine planning and mineral processing. A method is being presented whereby faults within the current Mat_Type system are removed and all of the improvements listed in Chapter 8 can be brought together to create an integrated holistic ore classification system.

This new method presents several advantages over the current Mat_Type system. Ore mineralogy is grouped more consistently by its mineral type. This grouping can be determined spatially in the ore deposit through coding in the block model. These groupings of ore mineralogy relate to certain processing methods employed at Kansanshi mine. Therefore an appropriate metallurgical process can be selected for each of the three main ore mineral groups. Through a process of regular review and analysis of cost data a range of cut-off grades can be defined that will sub divide ore groupings to assist in mine planning and head grade management at the mill. Finally, a set of metallurgical quality control factors supported by geological data can be applied to the sub-categories. These quality control factors will assist in the management of acid consumption and stop refractory mineralisation from reporting to the mill resulting in improved recovery.

This method is presented in Figure 8.5 and follows a similar principle to the Mat_Type system in that it classifies ore by way of a series of IF statements. These IF statements are simple questions that place upper and lower limits on a range of data. At each level of decision making the IF statement poses a question that allows a range of data to be broken down into two or more smaller ranges. Each smaller range has greater detail than their parent range. Each question is based on a systematic analysis of the available data. After passing a series of questions the data range is trimmed down to leave only that material with the appropriate characteristics to be classified as a single ore type. Each ore type is given a unique numerical code that can be stored in the block model. This code would allow a user to query a geological model and quickly see and understand three key pieces of information. This information can be listed as:

- The spatial location of ore and waste;
- The grade range of a particular ore type; and,
- The most appropriate metallurgical process to direct that ore type too.

In order to implement such a system some of the numerical codes previously used in the Mat_Type system could be re-used in this system. By doing so the potential impacts of confusion caused by a transition between ore classification systems could be minimised. The Mat_Type ore codes that can be reused in a new ore classification system are presented in Table 8.5.

Table 8.5 List of Mat_Type ore codes that could be reused in a new ore classification method.

Code	Ore Type Description
11	Oxidised mineralised waste
12	Low grade oxide ore, low GAC
14	High grade oxide ore, low GAC
22	Low grade oxide ore, high GAC
24	High grade oxide ore, high GAC
41	Sulphide mineralised waste
42	Low grade sulphide ore
44	High grade sulphide ore
52	Low grade mixed float ore
54	High grade mixed float ore
70	Waste
71	Refractory

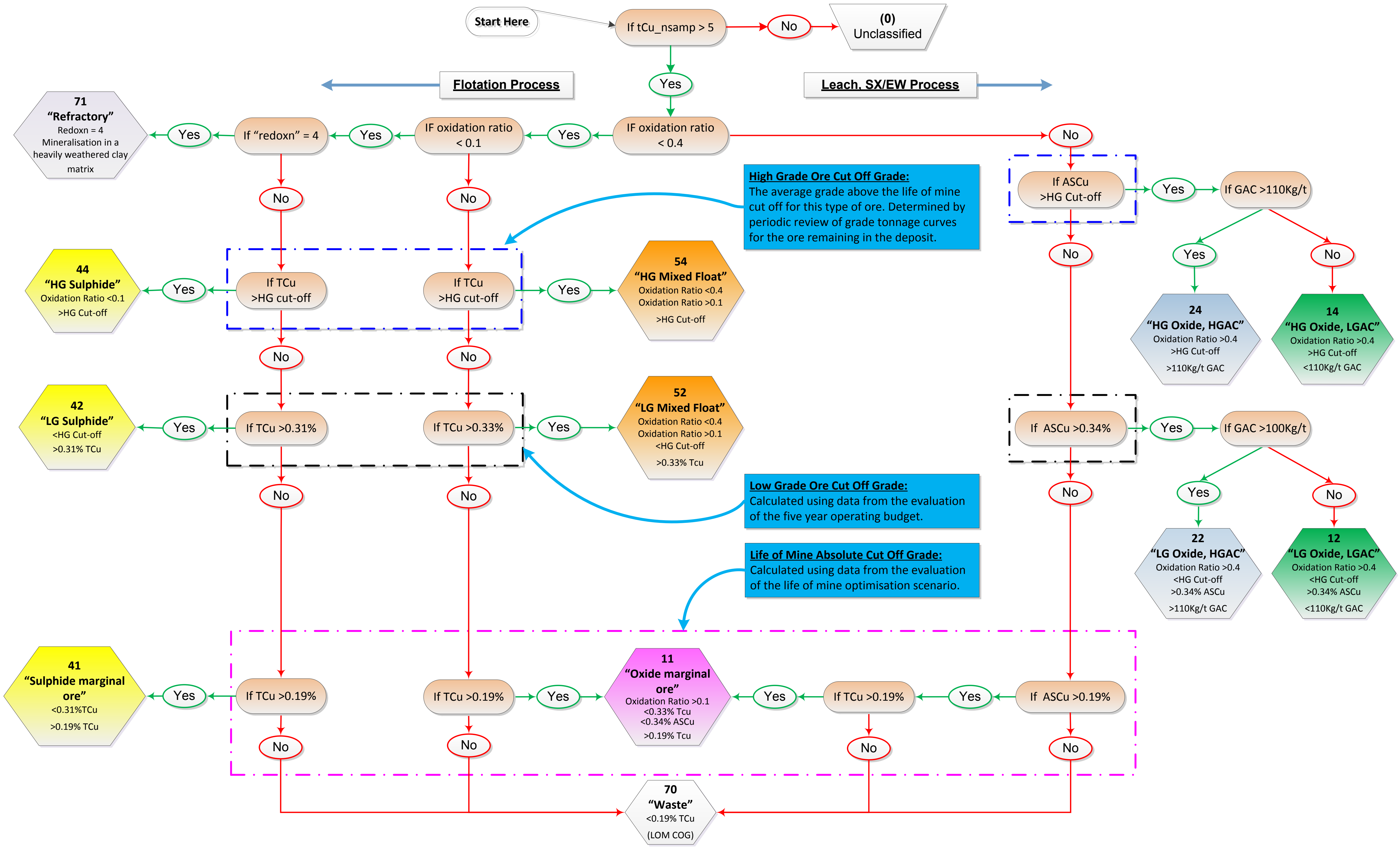


Figure 8.5 A more suitable integrated ore classification system for Kansanshi mine.

8.8 Chapter summary

Through the research presented in this report it has been demonstrated that the current ore classification system used at Kansanshi mine is inappropriate. This chapter has presented a number of methods whereby current geological and metallurgical understanding can be combined into a more appropriate ore classification system. This new system would be supported by the available data and ensure that mineralisation is grouped into ore types that are representative of the ore deposit and metallurgical process. The improvements in ore classification can be listed as:

- Continuous review and update of applicable cut-off grades, to ensure the business operates profitably;
- Defining mineralisation based on its mineralogy and determining an appropriate metallurgical process for that grouping; and,
- Application of appropriate quality control factors based on robust geological and metallurgical data.

By developing this system in-house the company's intellectual property remains in the custody of the company. This brings additional benefits to those already outlined. These additional benefits can be listed as:

- The detailed collective technical knowledge of all technical departments can be reviewed and applied to the system; and
- The risk created by using consultants to make critical business decisions without understanding the full context of the available data is removed.

A set of recommendations regarding the use of the data and ideas that have been reported thus far is presented in Chapter 9. From these recommendations a series of conclusions have be drawn with reference to the problem statement presented in Chapter 1.

9.0 CONCLUSIONS AND RECOMMENDATIONS

9.1 Introduction

This research project set out to determine if a geometallurgical strategy could be developed to integrate technical knowledge across corporate and site level departmental silos. The intention of this strategy was to improve the flow of value from an *in-situ* location in the deposit, along the mine value chain, to a final saleable product, thereby reducing the need for stockpiling and improving the company's revenue stream. This research report has covered a wide range of technical disciplines in the mine value chain. Data has been gathered and analysed to understand the main constraints within a particular silo. From this analysis opportunities for improved integration knowledge have been identified. Through Chapter 9 a series of recommendations are presented that highlights these opportunities and how best to use them to improve the flow of value along the company's mine value chain.

9.2 Systems development

The work that has been documented in this report represents the first full and systematic investigation of the mine value chain at Kansanshi mine. The mine has been operating for ten years and as the business has grown data management systems and integration of technical knowledge has not kept pace with the expansion. This poses a serious risk to the company as technical departments work in isolation of each other. This isolation or silo approach does not allow for the effective transfer and preservation of value along the mining value chain.

To continue to operate effectively in this highly volatile economic environment it is imperative that decisions are not made by individuals in isolation of the overall mine value chain. The management team needs to encourage and develop a technical forum where strategy data and decision making can be discussed by all stake holders. This forum based approach will ensure that any decision made does not benefit one silo, at the cost of all others. All decision making should be guided by compromises which apply both upstream and downstream of the area of concern. Such a compromise must be objective and in line with the goals of the business and not the individual silo manger's key performance indicators.

The information presented in this report represents a first pass attempt at developing a level of practical integration between geology, mine planning, financial and metallurgical departments. Continued development of this integration through systematic data review will strengthen the ties that have been created. Encouraging all stakeholders to think about the process of mineral definition, the mineral resource to mineral reserve conversion process and the metallurgical process will assist in developing greater levels of detail in data analysis. These improvements will provide greater confidence in any decision that is ultimately made.

9.3 Implement a more robust ore classification methodology

From the data presented in Chapter 6 it is clear that the Mat_Type system is an inappropriate method to classify ore at Kansanshi mine. With the business developments that are coming online in the near future there is an opportunity to develop an ore classification system that is more geologically and metallurgically appropriate. An added benefit of this will be a simplification of the mine planning process. The methodology presented in Chapter 8 is based upon the analysis of multiple sources of data which have been presented in this report.

Through more appropriate classification of mixed ore and high GAC oxide ore there is a potential to reduce the amount of ore that is currently directed to stockpile. This will have three significant benefits which are listed as:

1. Greater realisation of value as more ore will be treated rather than stockpiled;

2. Improved metallurgical plant performance through a more appropriate grouping of ore types based on a combination of mineralogy and grade; and,
3. Reduced pressure on the mining fleet as more exposed ore would be immediately directed to a suitable metallurgical process.

9.4 Periodic metallurgical test work

As the mine continues to produce, it is necessary that new areas of the ore deposit are stripped to expose more ore. In conjunction with this, new technology continues to present opportunities for improvement in mining and metallurgical processing. A formalised systematic process needs to be developed that allows periodic metallurgical and mineralogical test work to be completed on ore and waste samples. This test work should be completed at least on an annual basis. Sample selection and the type of test work that needs to be carried out should be determined through cross departmental integration between geology and metallurgical staff at the mine. Integration is key to ensure that samples are selected to be representative of the geological domains, ore grade ranges and metallurgical processes that would be employed to process the ore. Once suitable samples have been selected, the following test work should be carried out:

- Detailed optical microscopy;
- Qualitative and quantitative mineralogical analysis;
- Grind size and mineral liberation tests; and,
- A range of metallurgical recovery tests that mimic the mineral processing systems employed at the mine.

By formalising a regular metallurgical test work program an integrated collaborative approach develops that allows the team to share knowledge more effectively. This develops an understanding of the practicality that needs to be applied to any decision making process regarding how best to process an ore type found in a specific domain.

9.5 Chapter summary

Through the research and analysis presented in this report a number of improvements have been suggested that can be applied at Kansanshi mine to improve ore classification. These improvements will be realised through the integration of knowledge and understanding that sits in different technical silos. This chapter has presented a series of conclusions and recommendations that would allow the improvements suggested to be implemented along the mine value chain. This would break down traditional technical silo walls and improve the flow of value from its *in-situ* location through the various stages of the mining process until it produces a final saleable product. The key improvements recommended can be summarised as:

- Introduce a new ore classification system that considers geological domains, mineral oxidation and mineral economics;
- Carry out regular programmes of detailed qualitative and quantitative mineralogical test work;
- Develop systems throughout the mine value chain that promote integration and knowledge sharing between key technical disciplines; and,
- Complete regular, periodic and systematic reviews of processes to ensure that they remain valid given the fast pace of change that the modern mining industry now experiences.

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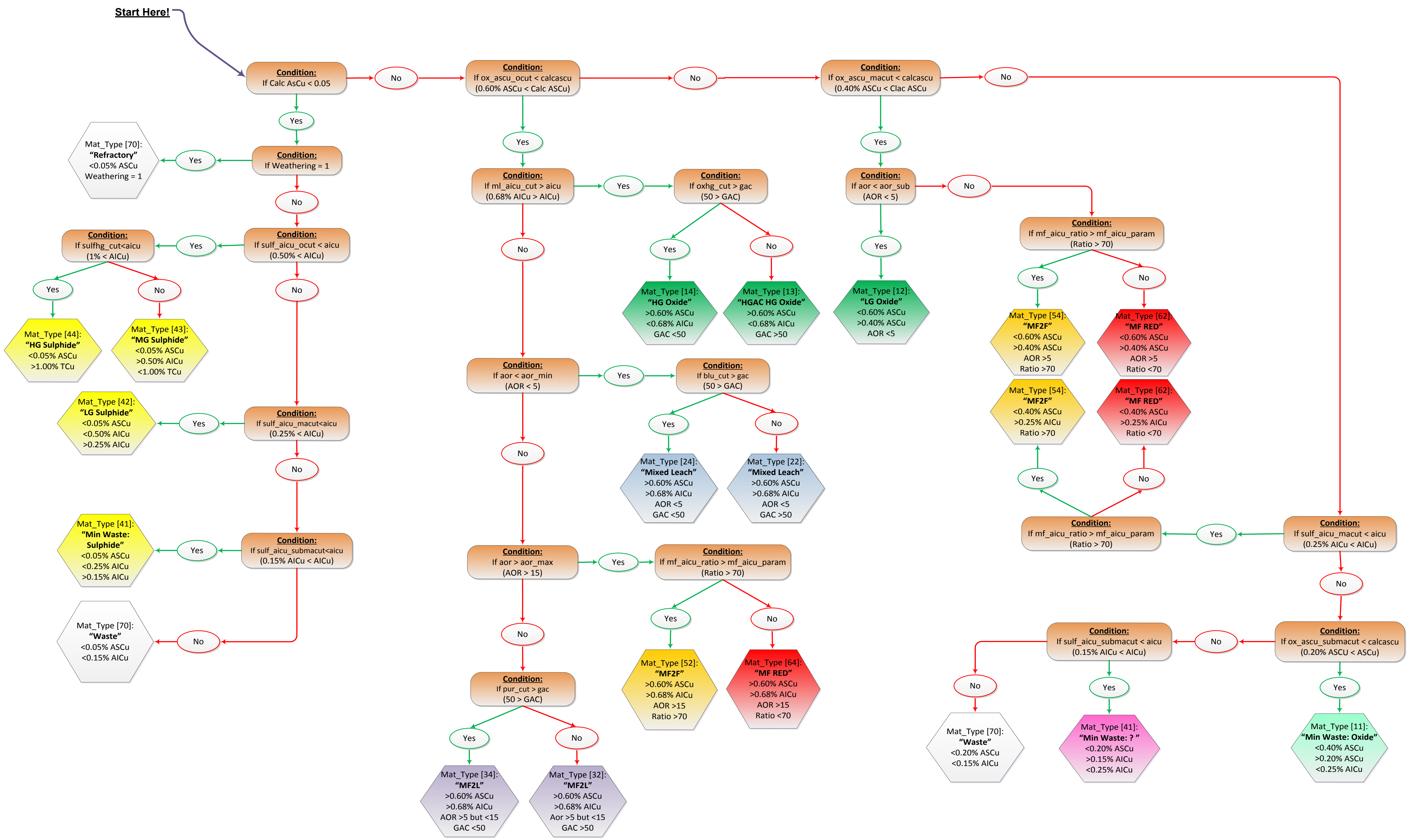
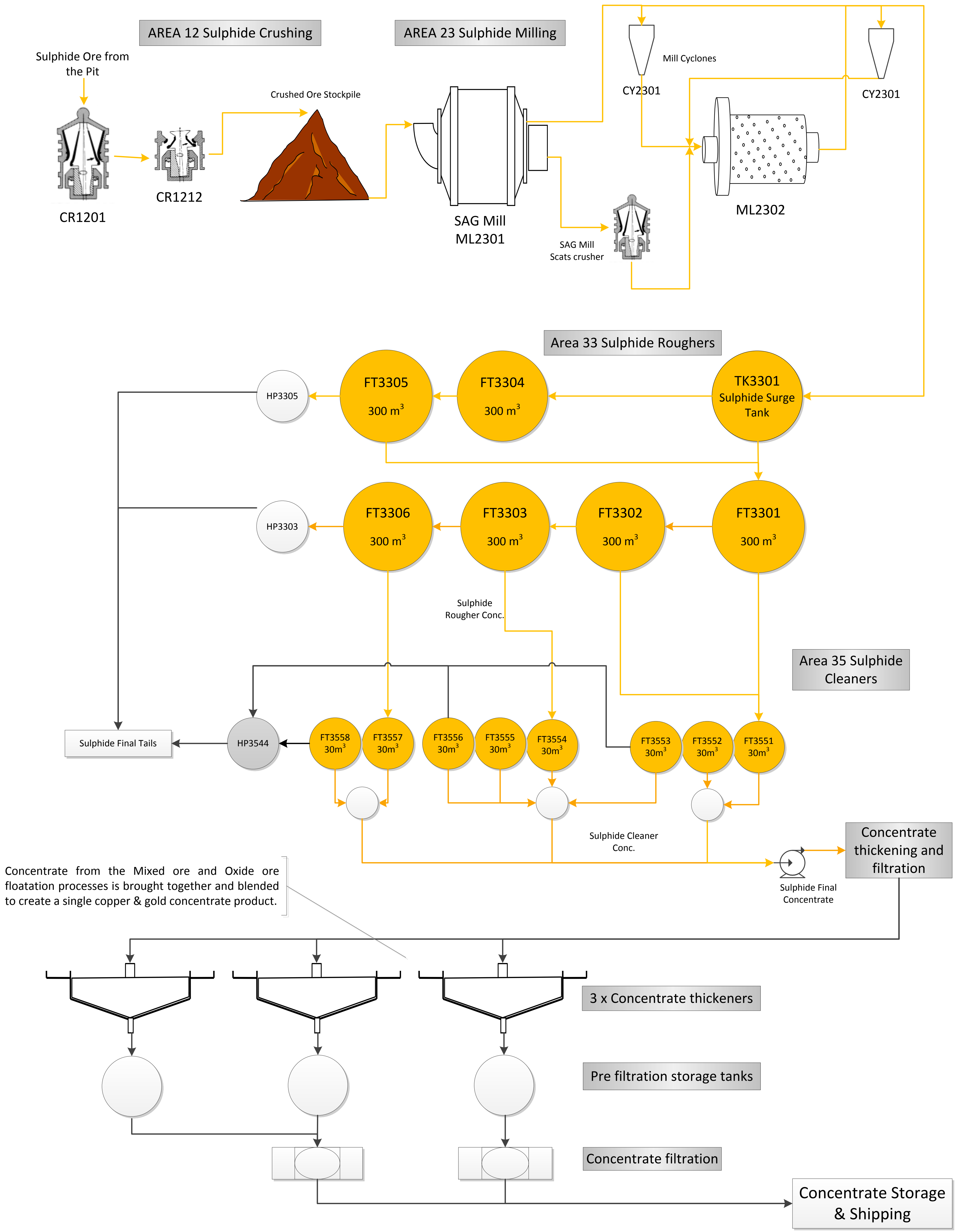
APPENDIX 1: DESCRIPTION OF PARAMETERS USED IN THE MAT_TYPE ORE CLASSIFICATION SYSTEM.

Condition Name	Full Name	Description	Formula
sulf_aicu_ocut	Sulphide ore AICu overall cut-off grade	Sulphide ore AICu overall cut-off grade. Creates a boundary between low grade sulphide ore [42] and medium grade sulphide ore [43].	
sulfhg_cut	Sulphide ore AICu high grade cut-off grade	Sulphide ore AICu grade boundary between high grade sulphide ore [44] and medium grade sulphide [43].	Manual input fixed at 1% AICu
sulf_aicu_macut	Sulphide ore AICu marginal cut-off grade	Sulphide ore, grade boundary between low grade sulphide [42] and sulphide mineralised waste [41].	
sulf_aicu_submacut	Sulphide ore AICu sub-marginal cut-off grade	Sulphide ore, grade boundary between sulphide mineralised waste [41] and waste [70]	
ox_ascu_ocut	Oxide ore ASCu overall cut-off grade	Oxide ore ASCu overall cut-off grade. Creates a boundary between low grade oxide ore [12] and high grade oxide ore [14].	
ml_aicu_cut	Mixed leach ore AICu overall cut-off grade	Separates leachable ore with higher AICu content from leachable ore with lower AICu content to assist with blending.	
oxhg_cut	Oxide ore high grade cut-off value	A cut-off value based on GAC content to assist with blending by splitting high quality low GAC ore [14] away from poor quality high GAC ore [13].	Fixed manual input at 50kg/t GAC
ox_ascu_macut	Oxide ore ASCu marginal cut-off grade	Oxide ore, grade boundary between low grade oxide [12] and oxide mineralised waste [11].	
ox_ascu_submacut	Oxide ore ASCu sub-marginal cut-off grade	Oxide ore, grade boundary between oxide mineralised waste [11] and waste [70]	
aor_min	Maximum acid to ore ratio value for leachable ore	The threshold value of AOR for mixed leach ore categories [22] and [24].	Fixed manual input set at 5
aor_max	Maximum acid to ore ratio value for non-leachable ore	The threshold value of AOR for mixed float ore categories [52] and [54].	Fixed manual input set at 15
aor_sub	Intermediate acid to ore ratio value to subdivide leachable and non-leachable ore	The threshold value of AOR for mixed leach ore categories [32] and [34].	Fixed manual input set at 5
blu_cut	"Blue ore" GAC cut-off value	A cut-off value based on GAC content to assist with blending by splitting high quality low GAC mixed leach ore [24] away from poor quality high GAC mixed leach ore [22].	Fixed manual input set at 50kg/t GAC
pur_cut	"Purple ore" GAC cut-off value	A cut-off value based on GAC content to assist with blending by splitting high quality low GAC mixed leach ore [34] away from poor quality high GAC mixed leach ore [32].	Fixed manual input set at 50kg/t GAC
mf_aicu_ratio	Mixed float ore AICu ratio	Determines the ratio between AICu and TCu to reduce the quantity of ASCu reporting to the mixed float circuit.	= (100 - ((AsCu / Tcu) * 100)
mf_aicu_param	Mixed float ore AICU ratio parameter	The threshold value for mf_aicu_ratio. A ratio not less than this parameter could be processed through the mixed float circuit.	Set to 70% through work by Paquot (2009).

APPENDIX 2: THE “MAT_TYPE” SYSTEM IF STATEMENT LOGIC TREE.

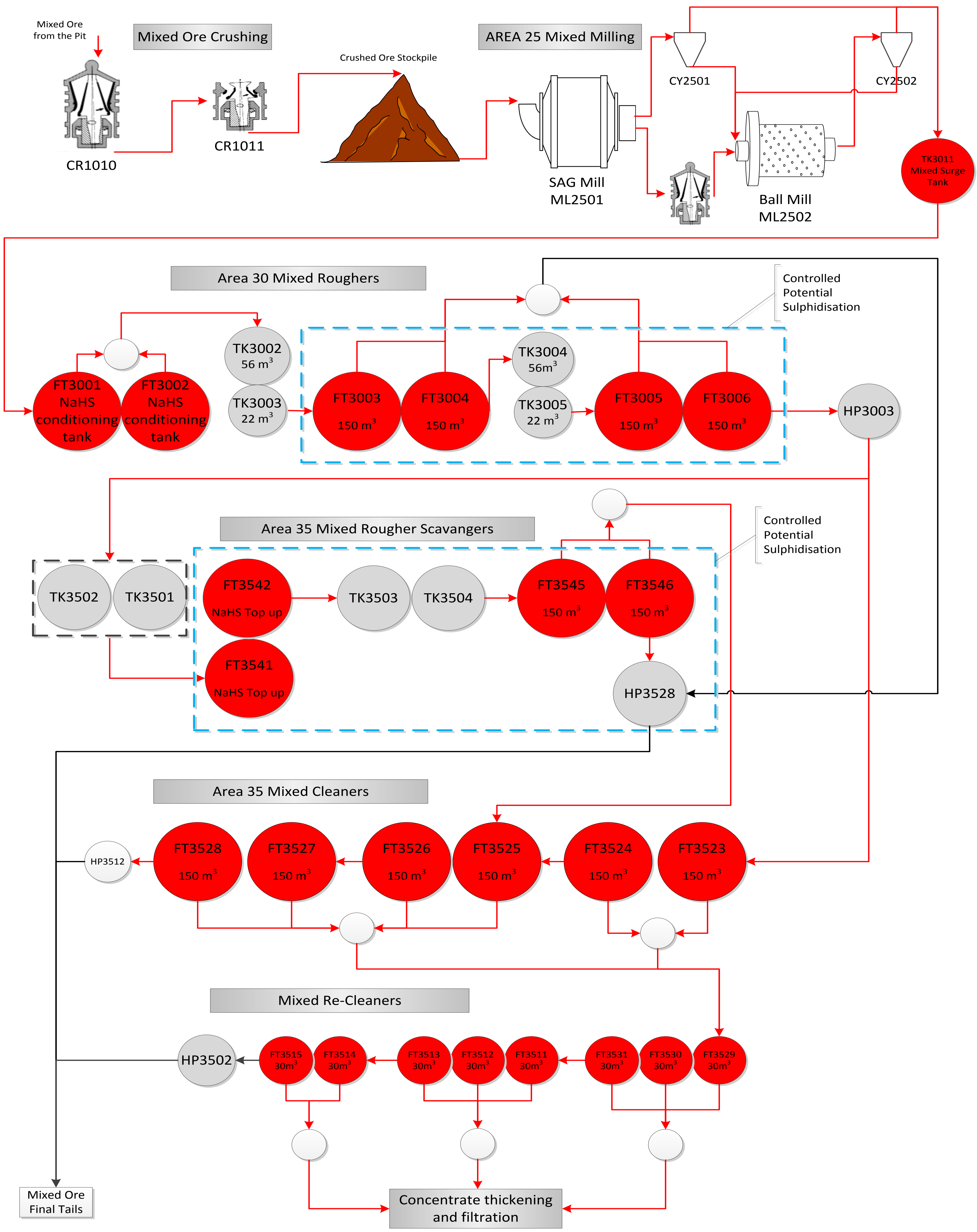


APPENDIX 3: METALLURGICAL PROCESS FLOW DIAGRAM FOR THE SULPHIDE ORE FLOTATION CIRCUIT.



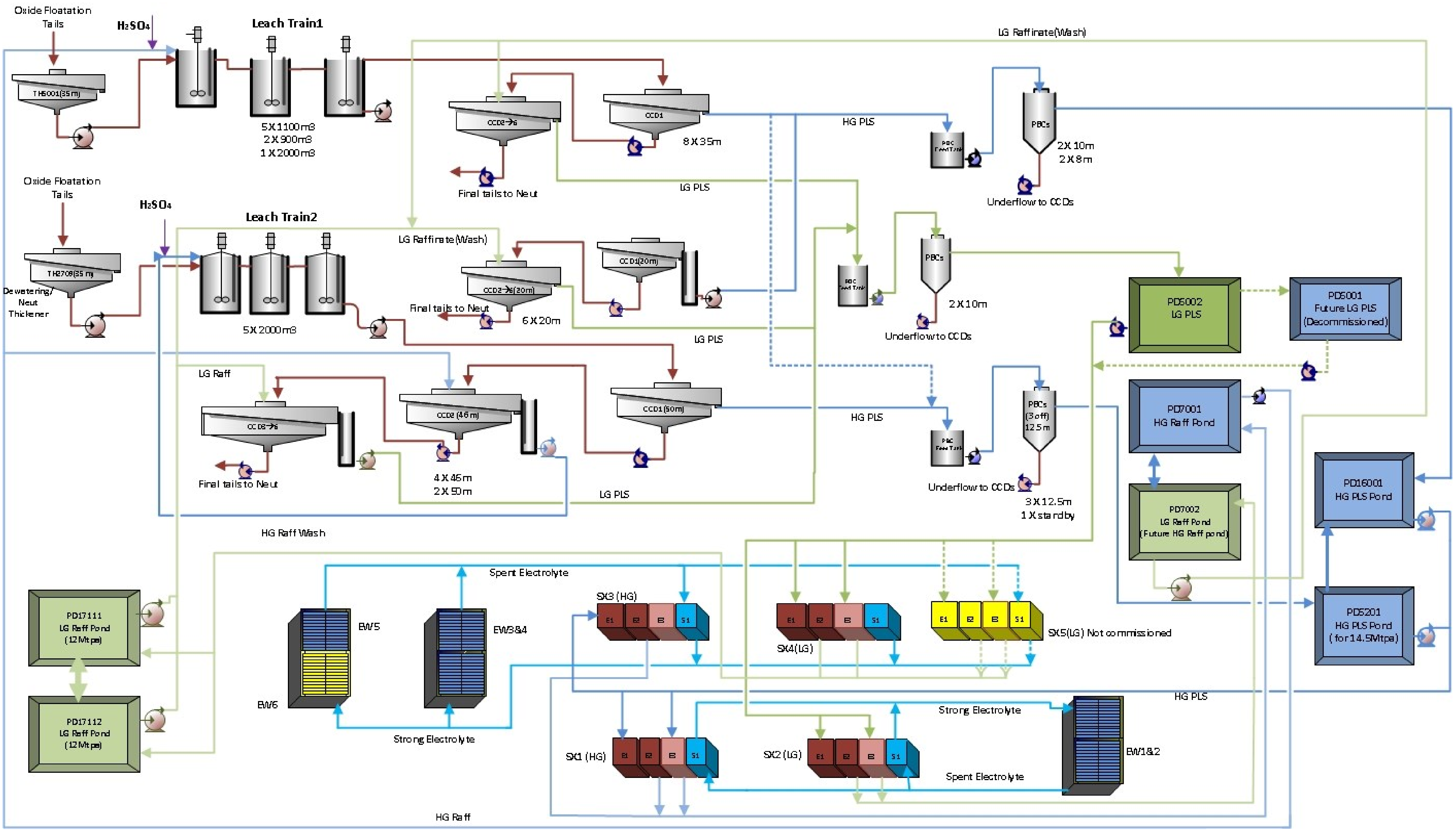
(Adapted from Lumb 2015)

APPENDIX 4: METALLURGICAL PROCESS FLOW DIAGRAM FOR MIXED ORE FLOTATION CIRCUIT.



(Adapted from Lumb 2015)

APPENDIX 5: METALLURGICAL PROCESS FLOW DIAGRAM FOR OXIDE LEACH, SOLVENT EXTRACTION AND ELECTRO-WINNING CIRCUIT.



(Adapted from Lumb 2015)