Development of a Metallurgical Model for a Copper Concentrator

Michael Valenta and Barbara Mulcahy

Metallicon Process Consulting (Pty) Ltd, Ifafi, South Africa

Corresponding author: mvalenta@metallicon.co.za

The purpose of executing a project in a stage-wise fashion is to cost effectively evaluate all potential options, developing an understanding of the project drivers, thereby minimising overall project risk to the client. One of the more valuable tools in the exercise is the development of production and financial models that can be used to quantify the sensitivity of the project to the critical parameters.

The mine and concentrator used in this case study are located on the Kalahari Copper Belt that extends from Zambia through northern Botswana into Namibia. This area of Africa has seen very little industrial development and the project will create much needed employment and development in the region.

The success of this project thus far can be attributed to the close cooperation between the geological, mining and metallurgical disciplines. Regular updates of projected capital and operating cost estimates coupled with project outputs guided the teams in optimising their various discipline models, ultimately delivering a compelling argument for the execution of the project. Key to the success was the development of a metallurgical model for the prediction of concentrate tonnage, grade and recovery for the life of the mine.

This paper summarises the metallurgical findings and the outcome of the various project stages. The development of the model is discussed, as are the challenges faced in developing a suitable metallurgical process route.

INTRODUCTION

The Kalahari Copper Belt

The Kalahari Copper Belt is part of the Central African Copper Belt that extends from the Zambian Copper Belt across northwest Botswana into Namibia. The copper occurs predominately as low sulphur and low iron copper sulphides (chalcocite and bornite). However, at shallower depths (<25 m), chrysocolla and malachite exist in significant proportions within certain geological areas.

As illustrated in Figure 1, the project is found in the Ghanzi-Chobe Fold Belt that forms part of the Kalahari Copper Belt. The Ghanzi Group, that overlies the lowest stratigraphic unit (the Kwebe Formation) consists of three formations namely; the Ngwako Pan, the D'Kar formation and the Mamuno formation (Hall, 2012).

Mineralisation occurs in the first ten metres of the D'Kar formation and is characterised by low sulphur, low iron, copper sulphides (chalcocite and bornite). The overall genesis of the mineralisation resulted from hydrothermal convention cells generated by the high heat flow associated with the thinning, extended continental crust during basin formation. The fluids circulating in these hydrothermal convection cells leached base metals from the underlying Kgwebe formation and produced large volumes of metal enriched, calcareous and siliceous fluids. These fluids then percolated upwards into the highly permeable Ngwako Pan Formation sediments and through them into the base of the less permeable D'Kar Formation. This resulted in the base metals occurring in disseminated blebs or within thin parallel calcareous and siliceous veins.

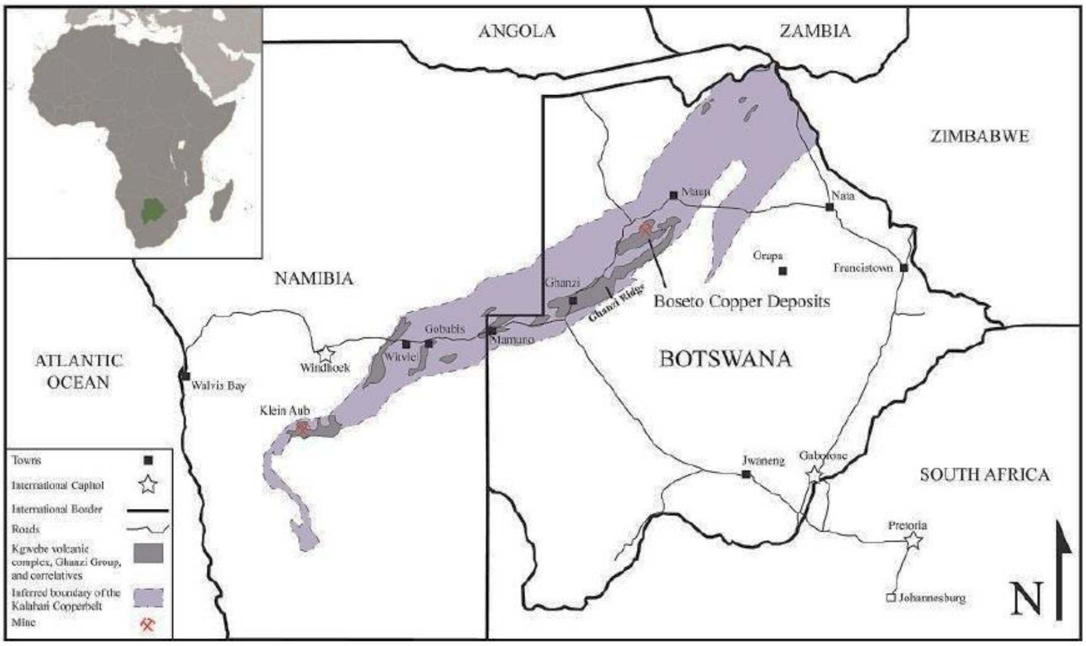

Figure 1. The Kalahari Copper Belt.

Two major areas (Area 1 and Area 2) have been identified in the project that are parallel in a south-west—north-east orientation and are approximately 1 km apart. The degree of mineral oxidation in Area 1 is significantly higher than Area 2 and this is attributed to an observed difference in bedding orientation within the deposits (Hall, 2012).

For the sake of definition the geologists initially defined three zones in the ore body based on the occurrence of various copper bearing minerals namely the upper zone (0–25 m), middle zone (25–35 m) and the lower zone (>35 m). This definition served its purpose in defining the test work that was required and the relative copper grades and recoveries that could be expected from each zone. On completion of the variability test work the definitions became less relevant when the predictive models were developed based on the mineral content of the ore.

Country Facts

The project is within 200 km of the Central Kalahari nature reserve and it is not uncommon to have game such as lion and elephants reported in the region. The project is also in close proximity to Maun that is regarded as the gateway to the Okavango Swamps and is of significant value to Botswana as an ecological gem and a tourist destination.

Furthermore, the project finds itself in an arid region where water is more precious to the local population than the diamonds from the nearby Orapa diamond mine. Throughout the development of the project the team considered all avenues to minimise the consumption of water. Botswana is also dependent on other countries, particularly South Africa, for the supply of electricity. The development in the resource market has put strain on the supply of electricity throughout the South African Development Community (SADC) and for this reason the team had to consider technology that would minimise the consumption of electricity.

Project Development
From the outset the project team planned to use a toll-gating approach and develop the project in stages as is pictorially illustrated in Figure 2.

The objective of such a strategy is to establish formal reporting deadlines thus allowing the three major project disciplines i.e. geology, mining and metallurgical to share information and update the project database. In other words the metallurgical engineers required updates from the geological and mining team on the latest geological models and mining models to refine their metallurgical test work. The mining engineers could then optimise their mining plans to focus on high yielding areas based on the head grades and the associated metallurgical response for each defined geological area.

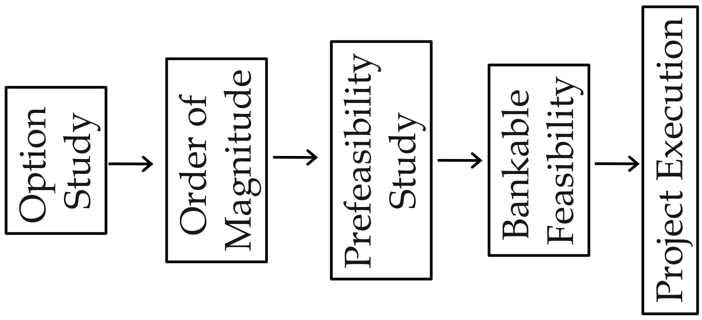

Figure 2. Stages in the development of the project.

Furthermore, as illustrated in Table I, the early stages of the project are done to a low degree of accuracy and do not normally involve all the engineering disciplines, i.e., mechanical, electrical, civil and structural. This significantly reduces the cost of these stages and the time taken to complete the work.

This allowed the metallurgical engineers the freedom to consider a variety of options early in the project with little or no input from the other engineering disciplines. The Capital and Operating cost estimates are however accurate enough to allow for a comparison of the different process options. Options could be carried forward to the next stage if further resolution was required.

Table I. Level of engineering and degree of accuracy.

Stage	Option Study	Order of Magnitude	Pre-Feasibility	Bankable Feasibility
Duration	Workshop	2 weeks	6 weeks	6 months
Expected Accuracy	none	30-40%	25-30%	5-10%
Typically Quoted Accuracy	none	-15%+30%	-10%+25%	-5%+10%
Process Engineering	Block Flow Diagrams	Preliminary Process Flow Diagrams (PFD's)	Decided PFD's	Final PFD's
Civil and Structural Engineering	none	Factored cost	General Arrangement Drawings (GA's)	GA's and Detail
Mechanical Enginering	none	Factored cost	General Arrangement	GA's, Detail and Tender process
Electrical and Control Engineering	none	Factored cost	Factored cost	Detailed take-off

METALLURGICAL TEST WORK

Throughout the test programme qualitative and quantitative mineralogical analysis was conducted on all of the samples. The findings of the analysis was related to the flotation results and provided valuable information in understanding the metallurgical response.

Initially two composite samples were produced from drill cores supplied by the geologists. These samples were used to conduct the preliminary tests to develop the process design criteria for the Order of Magnitude Study. The tests conducted included milling, gravity concentration, flotation and leach tests.

The preliminary results provided valuable information for the geologists and defined the criteria for characterising the samples. The subsequent reports from the geologists confirmed the varying oxidation highlighted in the literature (Hall, 2012) and three composites were generated for each of the two geological areas relative to sample depth as discussed above. Milling and flotation tests were conducted on the composites and the findings of the quantitative mineralogical analysis related to the results of the flotation tests. This provided adequate information for the development of the fundamental processing flow sheet.

Variability flotation tests were done using the proposed flow sheet on individual core samples to determine the variability in the flotation response and concentrate grades produced.

RESULTS AND DISCUSSION

Mineralogy

Qualitative Mineralogy

Qualitative mineralogical analysis on various drill cores from the two deposits confirmed the geological observations. The predominant copper mineral observed in samples from depth (>35 m) was chalcocite with small amounts of bornite. Trace amounts of malachite were also observed in these samples. Liberation of the copper-bearing minerals did not appear to be a challenge.

Shallower samples contained increasing amounts of malachite with samples shallower than 25 m containing varying concentrations of chrysocolla. The chrysocolla appeared to be associated with the malachite and liberation of the copper-bearing minerals would not pose a challenge.

Quartz was identified as the predominant gangue mineral, whilst plagioclase, mica, chlorite and calcite were also identified. Calcite seemed to be relatively more abundant in the upper zones than in the lower zones. Clay mineral occurs in minor to trace amounts in some of the upper and middle samples. Gangue mineralogy of calcite and chlorite, both high acid consumers, indicated that acid leaching may not be a process option.

Quantitative Mineralogy

Following on the initial qualitative mineralogy done on the earlier drill cores, more detailed quantitative mineralogy was done on the composite-by-depth samples and the variability samples.

The copper deportment in the two ore deposits are illustrated in Figure 3. It is clear that a large amount of chrysocolla is present in shallower samples that were submitted for test work. The middle zone appears to show a mixture of sulphide and oxide copper minerals, while negligible malachite and chrysocolla are evident in the lower zone.

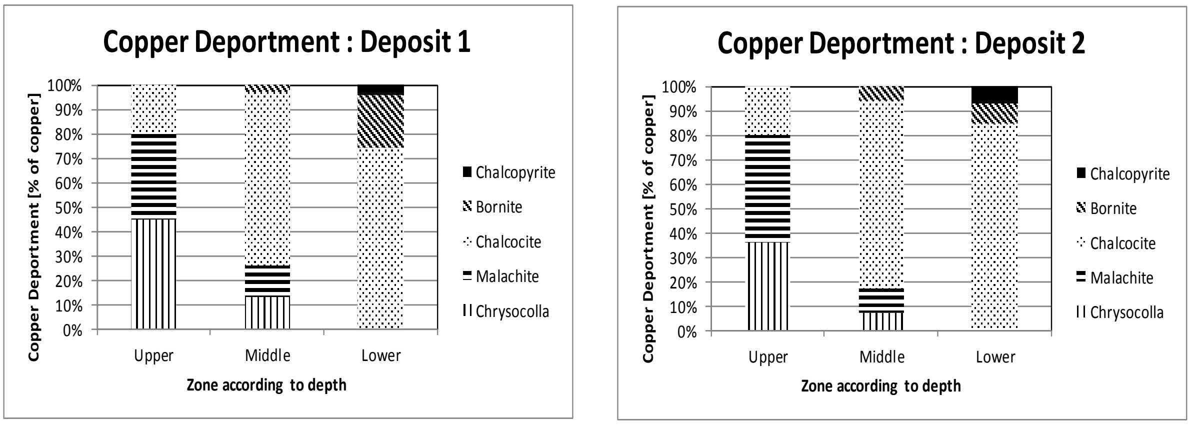

Figure 3. Copper deportment in the two deposits.

Pre-Concentration

To allow for an informed choice with regards to the inclusion of a Dense Media Separation (DMS) plant into the main process flow sheet, some heavy liquid separation (HLS) test work was carried out on the –6 mm to +1 mm fraction of the reef material.

Results showed that a DMS plant operating at a cut density of 2.7 t/m^3 would theoretically reject 30 percent of the mass, lose 9% of the copper to waste at a grade of 0.4% copper.

The DMS simulation indicated that due to the large amount of near density material, unstable operation of the DMS plant could be expected. The inclusion of a DMS plant in the flow sheet was therefore not recommended.

Comminution

Flotation tests were done at various grinds and it was found that the optimal grind was 55% passing 75 µm. Comminution test work included the characterisation of the ore through the determination of the Crusher Work Index; Rod and Ball Mill Bond Work Indices, and drop weight tests to determine the fully/semi-autogenous (FAG/SAG) mill parameters.

Regrinding the sulphide concentrate to a P80 of 38 µm did not yield an improvement in recovery but a significant improvement in copper grade. It was however noted that finer grinding of ores containing oxides resulted in a decrease in recovery and this was attributed to the liberation and subsequent loss of chrysocolla that was originally associated with floatable species. This was confirmed in the qualitative mineralogical analysis.

Flotation and Leaching

The initial test work done on the composites of the two drill cores indicated that the recovery of the predominant copper bearing mineral (chalcocite) did not pose a challenge and that recoveries in excess of 90% could be achieved at relatively high copper grades of up to 45%. The results were achieved using a classical sulphide flotation reagent suite of Potassium Amyl Xanthate (PAX) and a common, water soluble, poly-glycol ether as a frother (DOW 200).

Flotation tests on the shallower samples yielded a lower recovery. The mineralogical analysis revealed the presence of other copper minerals, particularly malachite and chrysocolla, in the flotation tailing.

The gangue acid consumption of the oxide ore was found to be excessively high, making it uneconomical to recover the oxide copper by acid leach. It was decided to utilise a sulphidisation step in the flotation circuit with the addition of NaHS to make the specific oxide minerals more susceptible to sulphide flotation (Kongolo et al., 2003, Newell et al., 2006, Raghavan et al., 1984).

In principle, the sulphidisation flotation method is attractive, but the major disadvantages include potential depression of the sulphide minerals when used in excess, difficulty in controlling optimum dosage and the fact that different oxide minerals respond differently to the addition of the sulphidiser.

Due to the fact that the sulphidiser may result in depression of the sulphide minerals, and the fact that the sulphide and oxide minerals reacted differently to fine grinding of the rougher concentrate, it was decided to process the ore in two rougher-cleaner circuits in series i.e. a sulphide circuit and a oxide circuit.

The chrysocolla was found to respond poorly to flotation, even with the addition of a sulphidiser. The mineralogical analysis revealed that the majority of the chrysocolla is recovered to a flotation concentrate through association with floatable species, e.g., malachite, that is recoverable under sulphidising conditions.

Solid–Liquid Separation

Test work was done to determine the thickening and filtration characteristics of the tailings and concentrate.

It was found that the tailing was not difficult to filter and that high thickener underflow densities of up to 69% solids by mass could be achieved. Paste thickening tests showed that paste could be generated at 72% solids by mass.

PROCESS DESIGN

Option Study

The option study took the form of a one day workshop and included a number of industry experts in the processing of copper. Presentations were done by geologists, mining engineers, minerals processing engineers and hydrometallurgists on the status of the project and the findings of other studies on similar ores. Various options were considered and the most viable options compiled in an option study report for future reference. It was the opinion of the group that the most viable option given the available data would be a classical flotation plant utilising a conventional crusher-grinding circuit.

Order of Magnitude Estimate

The process flow sheet recommended in the option study was considered and a capital and operating cost estimate generated.

The procedure followed required the development of the process engineering deliverables i.e. the process flow diagrams, mass balance and equipment sizing. Prices for major equipment were sourced from selected vendors and the team's database of prices. The concentrator direct capital cost was generated through a process of factorisation by assuming that the cost of the mechanical equipment would make up 40% of the overall direct capital cost. At the time of compiling the estimate this assumption was proven valid for six similar flotation plants. The operating cost was compiled from first principles utilising the team's experience and data base, and where necessary input from the client project team.

This information was used by the mining engineers as an input to optimise the mine design that is turn provided a revised input to the metallurgical team for the next stage.

Pre-Feasibility Study

The first set of test work confirmed earlier assumptions and it was decided that a flotation plant would be the most suitable process for the processing of the ore. The geological and mineralogical data available at the time of the study indicated the presence of oxidised copper minerals, although the relative amount was negligible. A conventional sulphide copper circuit in a rougher, cleaner and recleaner configuration was therefore selected for the flotation circuit.

Test work did however; indicate a benefit for a sulphide rougher concentrate regrind to a p80 of 38 micron and was therefore included in the design.

Various plant circuit configurations were considered with the focus on the comminution and tailing dewatering circuits. A large relative difference in the Rod Mill Bond Work Index (21 kWh/t) and the Ball Mill Bond Work Index (15 kWh/t) at a limiting screen size of 150 micron raised a flag indicating the potential for a critical size build-up in a conventional SAG mill. Three scenarios were considered for the milling circuit, namely:

- Conventional three-stage crushing circuit followed by two-stage ball milling;
- A conventional SABC circuit with a pebble crusher;
- A top-size crusher and a high pressure grinding roll (HPGR) followed by a conventional ball mill.

A cost analysis was conducted and although the first two options were very close in terms of cost the first option was selected as the option of least risk. This would be reconsidered during the next phase of the project.

Given the shortage of water in the region three options were considered for the processing of the flotation tailings. In each case the mode of deposition was also considered in the generation of the capital and operating cost.

- Belt filtration;
- High density thickened tailing (non-paste);
- Paste thickening.

The cost of the belt filtration and the associated material handling requirements for the filter cake eliminated this option. The negligible improvement in the water consumption using paste thickening did not justify the high capital and operating cost associated with the pumping of paste. Therefore, high density thickened tailing with a conventional tailing dam was selected.

The capital and operating costs were generated using a factorised approach as discussed in the section for the Order of Magnitude Estimate.

Bankable Feasibility Study

A large proportion of the metallurgical test work conducted during the Bankable Feasibility Study concentrated on conducting variability test work and locked cycle test work on the various ore types to confirm and support the assumptions made throughout the previous study. Extensive mineralogical analyses were included, which revealed the presence of varying amounts of oxidized copper minerals, i.e., malachite and chrysocolla.

Various alternatives were considered and an oxide flotation circuit employing a sulphidising agent was then included in the design.

Based on the findings of the test work the comminution circuit in Figure 4 was proposed. The run-of-mine ore is processed through a primary crusher followed by a coarse ore stockpile. Coarse ore from the coarse ore stockpile is processed through a secondary and tertiary crusher to a fine ore stockpile ahead of the primary ball mill.

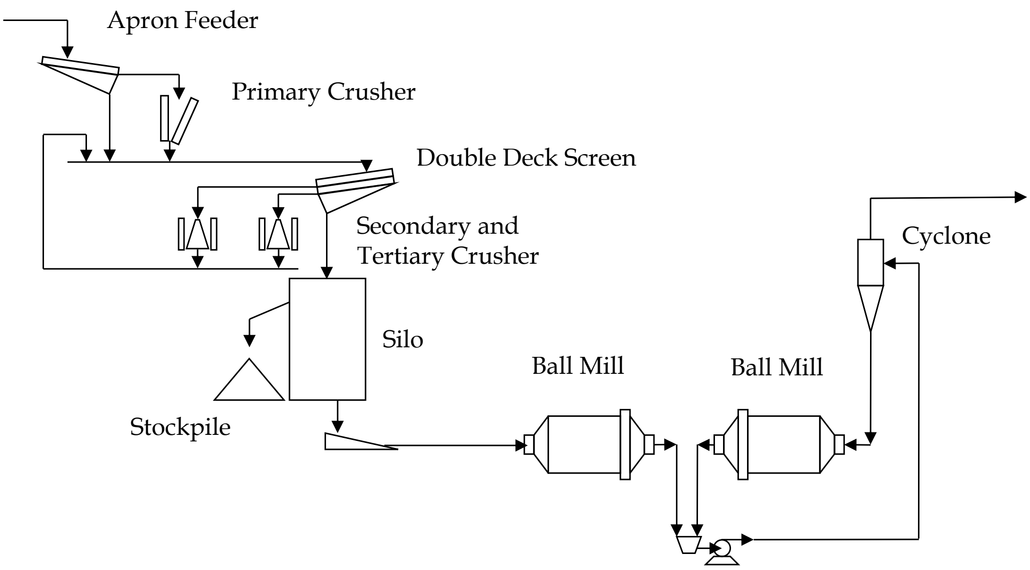

Figure 4. Proposed milling circuit.

The primary and secondary ball mill discharge are combined in a communal cyclone feed sump. The secondary ball mill is in closed circuit with a cyclone producing a cyclone overflow of 55% passing 75 µm.

The cyclone overflow is gravitated to the surge tank/conditioner ahead of the sulphide rougher flotation bank as illustrated in Figure 5. Sulphide rougher concentrate is re-milled to 80% passing 38 µm prior to cleaner flotation. Sulphide recleaner concentrate is thickened prior to filtration. Sulphide cleaner tailing is pumped to the head of the oxide cleaner bank.

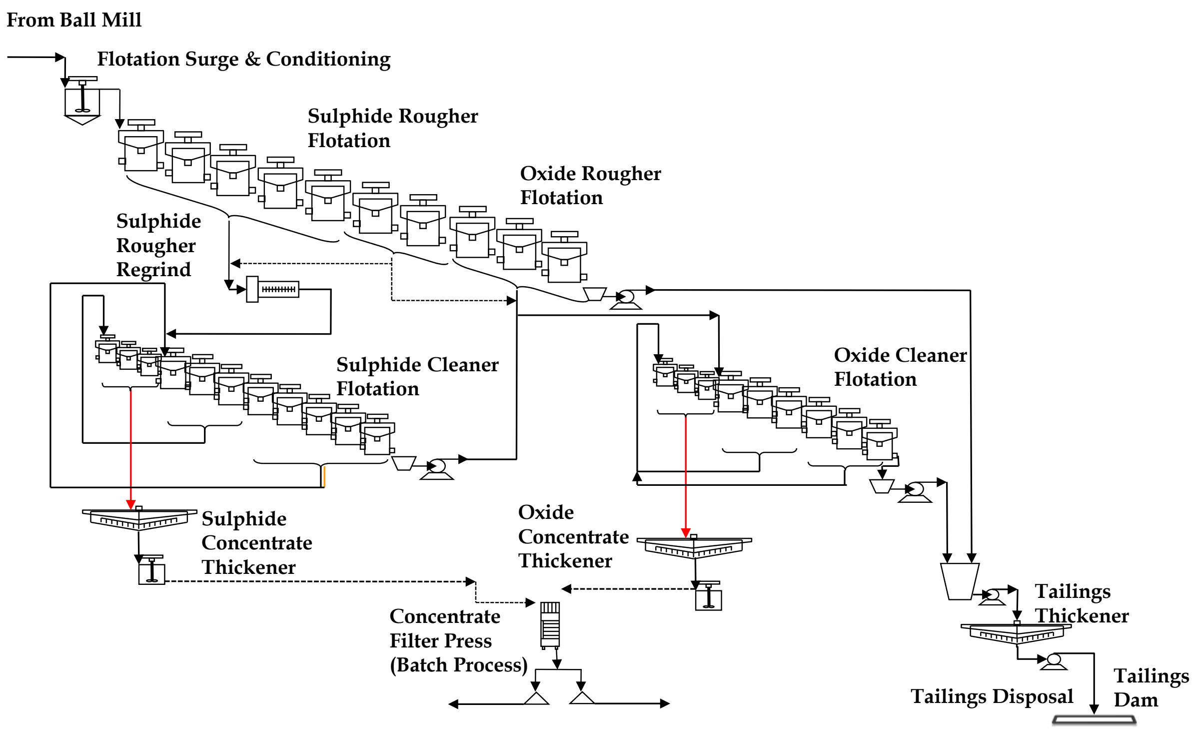

Figure 5. Proposed flotation circuit.

The sulphising agent (NaHS) is added to the sulphide rougher tailing ahead of the oxide rougher bank. The oxide rougher concentrate is not milled and processed in the oxide cleaner and recleaner bank. The oxide recleaner concentrate is thickened prior to filtration.

The oxide rougher tailing is thickened in a high density thickener to 68% solids by mass before being pumped to the tailing dam.

METALLURGICAL MODELS

The metallurgical inputs are critical in determining the viability of a new mining venture. The value of this data is important from two aspects:

- Determining the amount of valuable mineral that will be produced as a direct input into the financial model;
- Providing inputs in terms of metal recovery and associated costs into the mining plan and financial model in order to optimise the mining plan.

In the initial phases of the project, discrete recovery predictions for each mining area, based on the initial batch test work, were used to estimate overall recovery for the mining plan and financial model. As more flotation and mineralogical data became available, a more detailed model forecasting the recovery, concentrate grade and concentrate tonnage, was developed.

Development of the Models

A relationship between the sulphur:copper ratio and copper recovery became evident when testing samples from the different zones. It became necessary to further investigate this relationship conducting tests on individual samples within the range of sulphur:copper ratios for these ore bodies (typically between 0 and 30%).

The relationship illustrated in Figure 6 was observed and linear regression of the data generated. It is interesting to note that for the lower sulphur:copper ratios the recoveries for Area 2 are significantly higher than the corresponding recoveries for Area 1. This supports the observation made in the geological findings that the copper minerals in Area 1 tend to exhibit a higher degree of oxidation compared to Area 2.

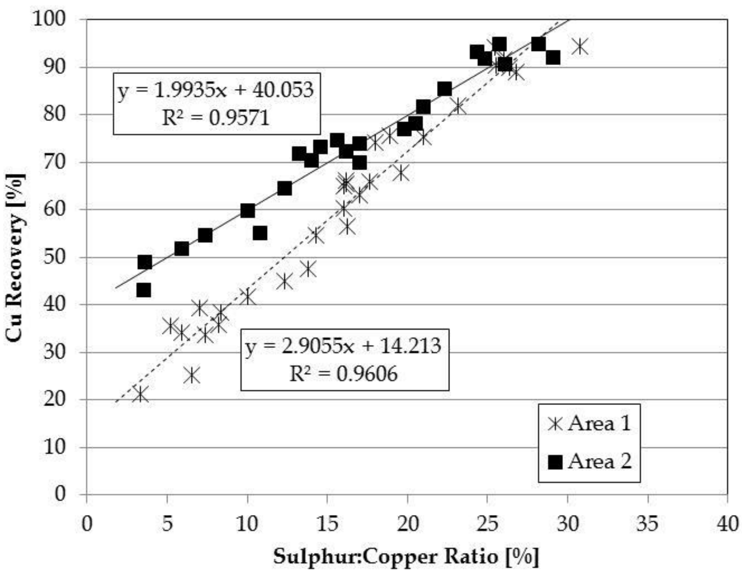

Figure 6. Relationship between Cu recovery and sulphur:copper ratio.

A similar relationship was observed between the mass ratio reporting to the sulphide concentrate and the sulphur:copper ratio. The relationship was important in determining the overall grade of the concentrate produced.

A comprehensive analysis was done of the sulphide and oxide concentrates for the upper, middle and lower zones identified by the geologists for each mining area. Based on the zone that was being considered in the mining plan a sulphide concentrate grade and oxide concentrate grade could be derived. This overall concentrate grade could then be calculated from the mass ratio to sulphide or oxide concentrate discussed above.

Concentrate Production Model
The mining plan compiled by the geologists and mining engineers was used to generate a forecast of the metal production with its associated cost. The algorithm derived from the above relationships was included in the mining plan and the financial model together with the capital and operating costs.

The algorithm can be summarised as follows:

- Calculate the mass of copper recovered to concentrate using the mining head grade and tonnage, and the recovery from the recovery model;
- Calculate the fraction of concentrate reporting to the sulphide concentrate using the linear relationship with the sulphur:copper ratio;
- Apply the copper concentrate grade based on the sulphur:copper ratio to determine the grade of the sulphide and oxide concentrate;
- Calculate the mass of sulphide and oxide concentrate produced.

This algorithm was used to predict the copper production and the concentrate grade for the life of mine. The copper production and copper grade for the first 24 months of operation are illustrated in Figure 7.

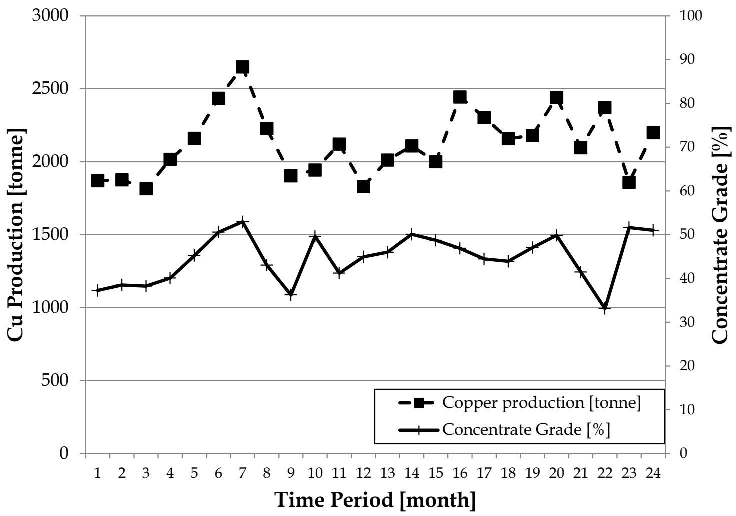

Figure 7. Forecast concentrate grade and production.

Use of the metallurgical model within the integrated mining plan yielded an additional 900 000 tonne of mineable ore from an overall 14.5 million tonne reserve. This is equivalent to an increase of 6% in the mineable reserves.

QUALITY OF THE METALLURGICAL TEST WORK

Comparison of Assays

The geological team had commenced with the assaying of the drill cores well before the metallurgical test work had commenced at Mintek in Randburg, South Africa. As the client was based in Australia it had been decided to submit the geological samples to the Genalysis laboratory in Australia.

A comparison was completed on the head assays for each of the drill core samples for the Mintek assays and the Genalysis assays. Applying the paired t-test analysis to the data set confirmed that the two data sets were not significantly different at a 95% confidence level. This finding was significant as it indicated that the model developed using the Mintek analysis could be applied to the mining model developed in Australia using results from Genalysis.

Quality of Flotation Test Work

During the flotation test work a number of routine checks are completed to determine the quality of the test work, e.g., the comparison of the re-constituted head grades and the assayed head grades of the samples for the various flotation tests as illustrated in Figure 8. The paired t-test analysis of the datasets found that there was no significant difference in the two sets of data at a 95% confidence level.

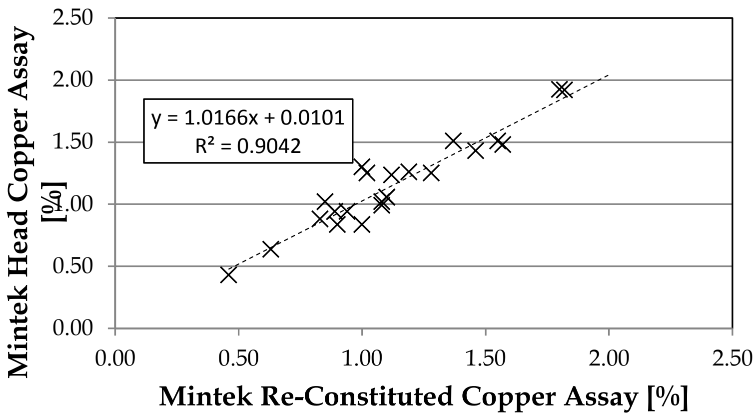

Figure 8. Comparison between Mintek and Genalysis assays.

CONCLUSIONS

The mineralogy of the Kalahari Copper Belt is interesting and presents challenges, particularly in the recovery of copper in the form of copper oxides, where recoveries as low as 30% can be encountered. Further fundamental test work needs to be completed in identifying more efficient means of recovering the copper, particularly in the presence of high acid consuming species.

Detailed mineralogical analysis prior and during the metallurgical test work added a significant amount of value in that it allowed the metallurgists to relate the results of the test work to the observations made under the microscope. It also added value in that the mode of occurrence of the copper lost to tailing could be identified and addressed in the process design.

Variability test work proved to be critical in accurately defining the metallurgical characteristics of the ore deposit; thereby ensuring that ore variability was recognised in the mining plan.

Cooperation and regular communication between the various disciplines on the project proved to be critical in ensuring the success of the project and minimising risk to client. As illustrated in this case study, the combination of the discipline models has resulted in ore previously classified as "uneconomical" being reclassified and deemed mineable, making the project a more viable proposition.

REFERENCES

Hall, W.S. (2012). Geology and paragenesis of the Boseto copper deposits, Kalahari Copper Belt, North West Botswana. Thesis. Colorado School of Mines.

Kongolo, K., Kipok,a M., Minanga, K. and Mpoyo, M. (2003). Improving efficiency of oxide-cobalt ores flotation by combination of sulphidisers. *Minerals Engineering*, 16, 1023–1026.

Newell, A.J., Skinner, W.M. and Bradshaw, D.J. (2006). Restoring the floatability of oxidized sulfides using sulfidisation. *International Journal of Mineral Processing*, 84, 98–107.

Raghavan S., Adamec E. and Lee L. (1984). Sulfidisation and flotation of chrysocolla and brochantite. *International Journal of Mineral Processing*, 12, 173–191.