KGWAKGWE HILL MANGANESE PROJECT PRELIMINARY ECONOMIC ASSESSMENT

Prepared For

Giyani Metals Corporation

Report Prepared by

srk consulting

KZ0528/UK30470

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KGWAKGWE HILL MANGANESE PROJECT PRELIMINARY ECONOMIC ASSESSMENT

1 INTRODUCTION

SRK Consulting (UK) Ltd was commissioned by Giyani Metals Corporation (hereafter referred to as Giyani, the Company, or the Client) to undertake a Preliminary Economic Assessment (hereafter referred to as the PEA) for the viability of the Kgwakgwe Hill Project (hereafter referred to as K-Hill, or the Project), which is located in the Republic of Botswana. SRK understands that the Company has an 88% interest in the Project at the time of writing.

This report presents the findings, conclusions, and recommendations that resulted from the PEA. SRK notes that this PEA is preliminary in nature, and that the findings and conclusions should be treated as preliminary as well.

Any schedule tonnes and grades in this document do not represent an estimate of Mineral Resources and are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorised as Mineral Reserves. There is no certainty that the preliminary economic assessment will be realised.

1.1 Scope of Work

The PEA encompassed the following tasks:

- Brief geological review and investigation of the block model (prepared by a third party)
- High level geotechnical recommendations based on the available data (known to be limited)
- Establishment of mining, processing and G&A costs to allow the construction of an optimal pit shell to a scoping level
- Scoping level design of a pit based on an optimal pit shell
- Development of a life of mine plan ("LoMp") to scoping level
- Review of the available metallurgical test work for the required processing plant
- A review of tailings storage options based on a desktop study
- Water management review
- Infrastructure review
- Environment and Social review including closure costs
- Financial model based on the LoM plan
- For all technical arenas the appropriate operating costs and capital costs were estimated to a scoping level of accuracy.

As there is no Indicated Resource within the block model at the time of writing, this report does not present Mineral Reserves, but instead presents a Resource converted to Run of Mine ("RoM") material where the mining factors have been applied to the Inferred Resource. A mine plan was made on this RoM material and a financial model was constructed on the basis of a scoping level mine plan.

The expected accuracy of costs will be within a -35% to +45% window, as is appropriate for the level of study and accuracy of the input data provided.

1.2 Project Location

The Project is located next to the town of Kanye, a 1-hour drive on good roads from the capital of Botswana, Gaborone, and a half hour drive from the town of Lobatse, on the border with South Africa. The major A2 arterial highway is only a few kilometres from the Project.

Giyani owns 100% of Menzi Battery Minerals, which holds the prospecting rights of 14 licenses covering 8,283 km^2. Menzi owns an 88% interest in Prospecting Licence No. 322/2016, which will expire on 31st December 2019. The license is renewable for another 2 years after the expiration date as an exploration license. It is also possible to apply for a 5-year retention on the license. The K-Hill project occupies 878 km^2 of that total licence area.

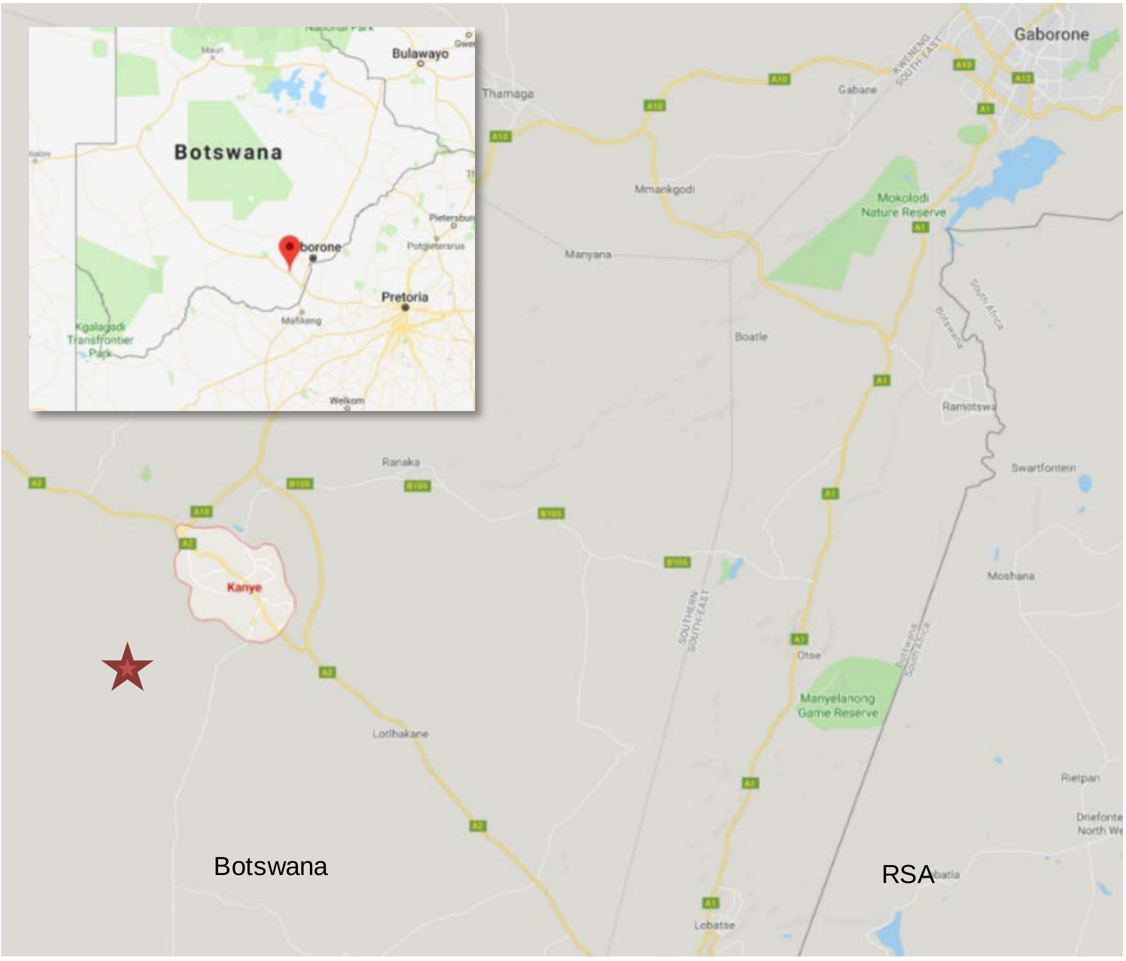


Figure 1-1: Project Location

1.2.1 Accessibility

The K-Hill Project is only a few hundred metres from the paved roads of the town of Kanye, and only a few kilometres from the large A2 highway, which is a major part of the Trans-Kalahari Corridor, which links the East Coast of the Southern African continent to the West Coast by running from Maputo, Mozambique, via Pretoria (now known as Tshwane), to Windhoek in Namibia.

1.2.2 Climate and Physical Geography

Southern Botswana is mostly covered in savanna. It is a high lying undulating plain, with rocky hills and outcrops. The town of Kanye lies at an elevation of approximately 1300 mamsl, with the K-Hill Project reaching nearly 1500 mamsl.

The climate is considered warm and arid with a summer rainy season. There is no hindrance to operations due to climatic reasons.

Local meteorological station records were obtained by SRK from publicly available National Oceanic and Atmospheric Administration (NOAA) databases. Accordingly, SRK obtained the precipitation data for Kanye Meteorological Station, which is located 4 km from the city of Kanye (Lat= -24.96675, Long= 25.33273). Precipitation records from this station cover the period from 1921 to 1989 however data of only 30 years (where there are no gaps) have been used to analyse the rainfall pattern.

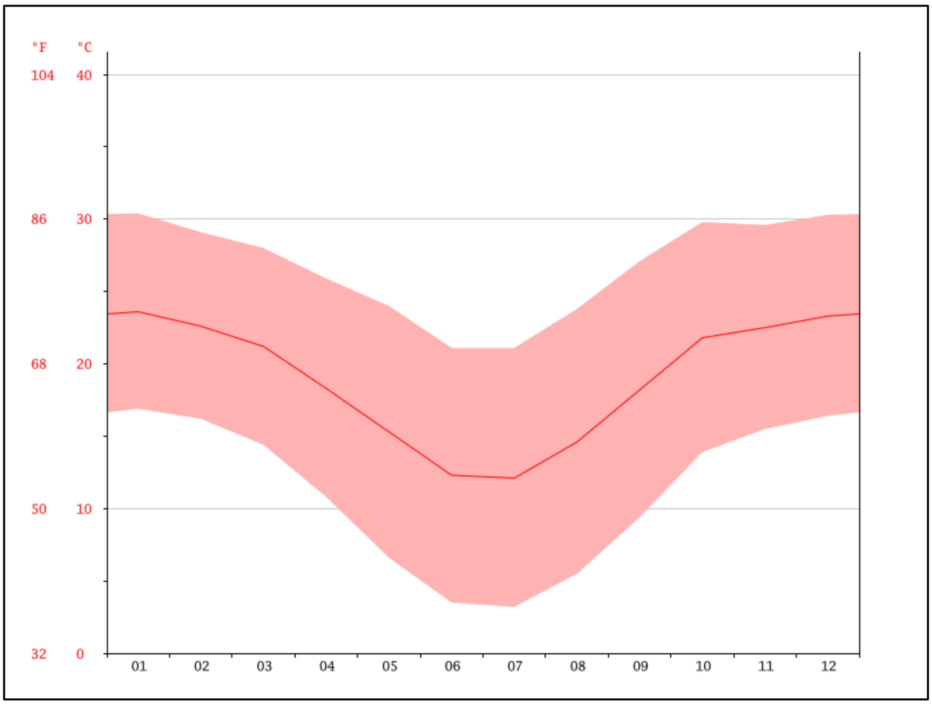


Figure 1-2: Average Monthly temperatures for Kanye town

Source: en-climate-data.org

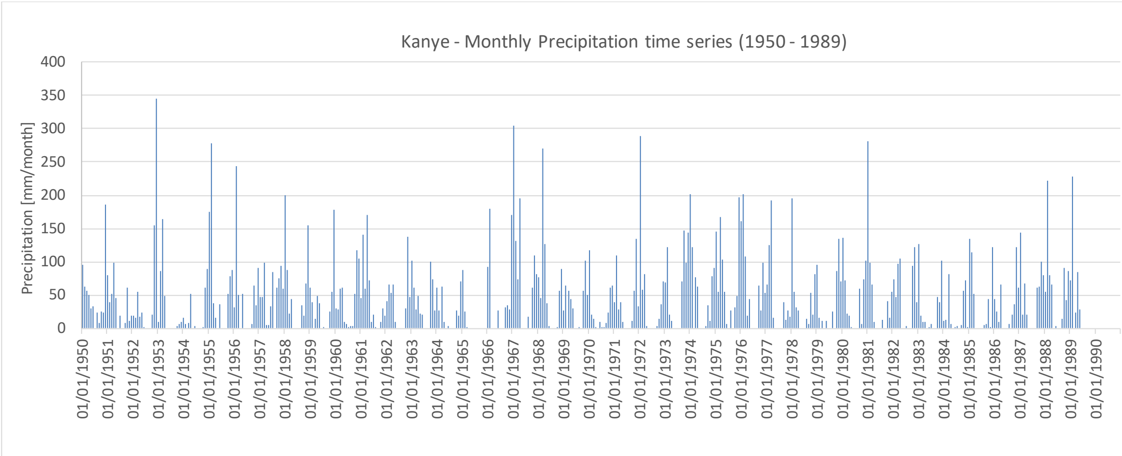


Figure 1-3: Monthly precipitation record at Kanye

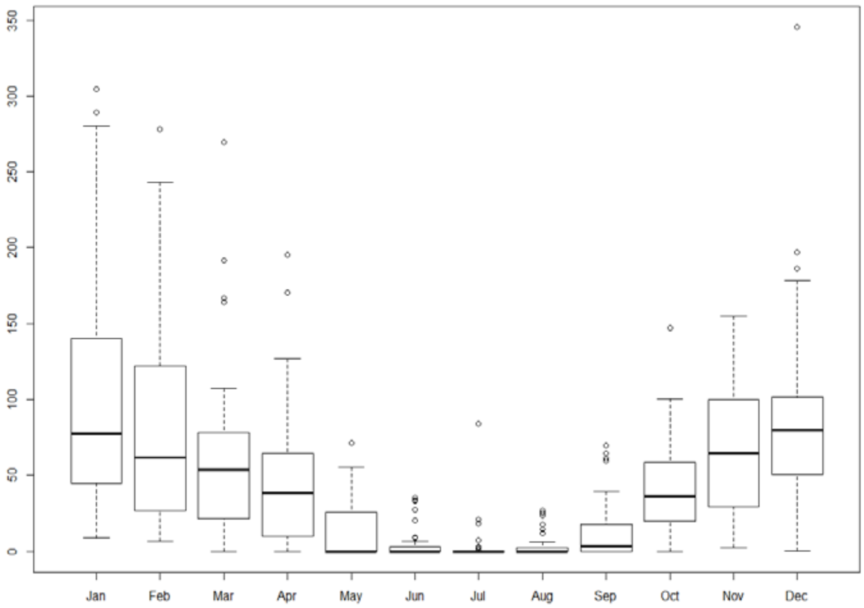


Figure 1-4: Variability of monthly precipitation (mm/month) based on monthly records at Kanye station over the period 1950-1989 (the whisker box shows the average and 1st and 3rd quartiles)

1.2.3 K-Hill Local Features

During the site visit, SRK observed several features of the Project area that are worth noting and have an impact on the Project.

Several tailings dumps as well as stockpiles were observed; remnants of both commercial and artisanal mining activities. The stockpiles are very small, in terms of tonnage, and will not be considered for this PEA as RoM feed, however Giyani is considering to sell these stockpiles into the steel market. They were, therefore, charted and measured in Giyani documentation delivered to SRK. SRK was informed that tailings were sampled but not found to yield any recoverable material.

Source: SRK 2019

Figure 1-5: Typical Historical tailings dump at K-Hill

The town of Kanye is very close, and should there be any blasting required, there may well be a risk of dust pollution. It will be advisable to plan any blasting with favourable wind conditions. However, it is anticipated that blasting will be limited to some of the chert breccia cap only. This will make it easy to plan blasting without affecting production.

Source: SRK 2019

Figure 1-6: Looking North from K-Hill over the town of Kanye

There are few concrete reservoir structures within the K-Hill Project area, one of which was only put to active use very recently for the Kanye town fresh water supply. Development of the K-Hill will require this reservoir to be moved. The reservoir has a diameter of approximate 20 m. The other reservoir structures seem older and not being used.

On the hill to the South, there is also the local cell phone tower for the town. It will also need to be moved to enable mining operations. There is some uncertainty on the cost involved in moving the structures on site. This will be covered by the contingency applied on the Project.

Figure 1-7 shows the general layout of the infrastructure at site, and the location of the exploration boreholes.

Source: Google Maps

Figure 1-7: K-Hill Project – Infrastructure

1.2.4 Historical Mining

Between 1957 and 1971, historical mining and exploration took place by Marble Lime and JCI companies. Manganiferous shale was sold after beneficiation to steel smelters and some stockpiles are still in place on site as evidence of this activity. Rand Mines estimated there to be around 1.6 Mt of manganese shale, of which around 60% could be extracted. In 1981, some exploration was done, but the licence was relinquished after one year.

For more information, this report refers to the NI 43-101 (and its recorded sources) that was completed by the MSA Group (Pty) Ltd in November 2018.

Source: SRK 2019

Figure 1-8: Historical mining excavation (height of stope, ca. 1m).

1.3 Economic Potential of the RoM material

The manganese market has changed in recent years, with the advent of more advanced battery technology. In previous eras, manganese was virtually only used to increase the melting point of steel, and also as an alloying element for hardening steel. For that purpose, only high-grade manganese deposits (over 45% MnO) were considered economical.

The RoM material at K-Hill is of low quality for the purpose of steel making, with an average grade of around 30%. However, it is considered viable for production of EMM based on the work carried out for this study. Electrolytic Manganese Metal ("EMM"), usually at 99.7% purity, is increasingly used as a starting material for the production of $MnSO_4$ (manganese sulphate) production for battery cathodes.

The increased demand for Li-Ion batteries has seen a sharp increase for battery metals such as cobalt and lithium. Whilst manganese is lagging behind somewhat, it is starting to become more important. Roskill, a UK-based research company, has predicted that manganese demand will grow at a compounded annual rate of 23% from 2019 till 2029. This is largely underpinned by demand for Nickel-Manganese-Cobalt ("NMC") cathodes.

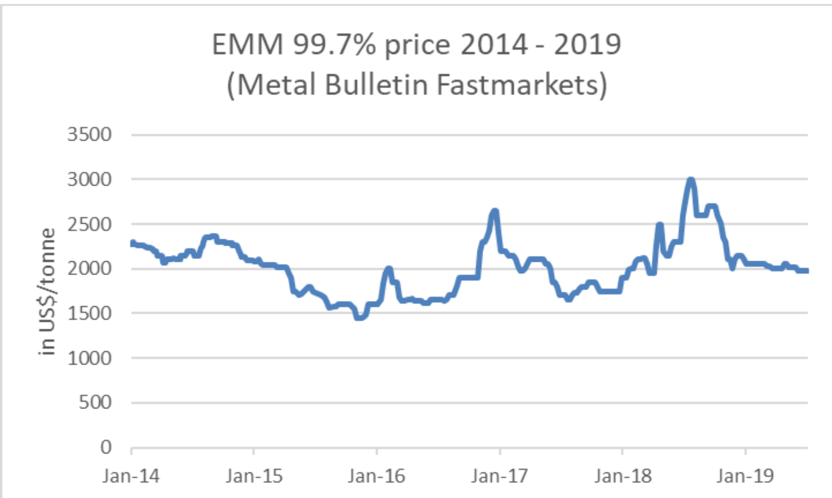
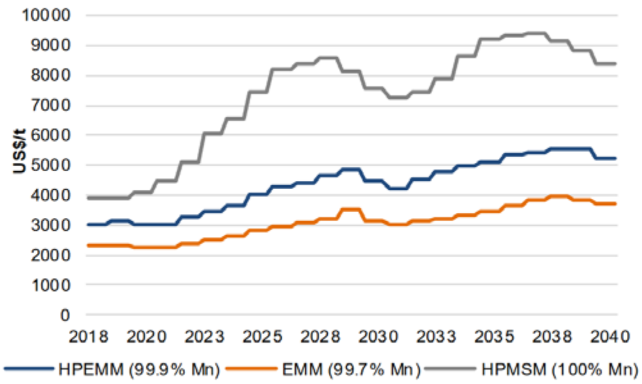


Figure 1-9: 99.7% EMM price trend in USD/t

It should be noted that the intention for the K-Hill plant is to produce 99.9% EMM, which sells for a significantly higher price. The pricing assumed for 99.7% EMM will be USD2500/t in this PEA, whilst the price assumed for 99.9% EMM will be USD4700/t. 99.9% EMM is also known as HPEMM, which stands for High Purity EMM. Electrolytic Manganese Metal ("EMM"), usually at 99.9% purity, is increasingly used as a starting material for the production of high purity $MnSO_4$ production for battery cathodes. A comparative pricing forecast, prepared by the CMP Group, can be found in Figure 1-10.



Source: CMP GROUP

Figure 1-10: EMM and HPEMM Pricing forecast

1.4 Site Visit

The SRK Project Manager for this PEA, Onno ten Brinke, visited the site on 19th March 2019. The following tasks were accomplished whilst on site, and at the local office in Gaborone, Botswana on the 20th of March, 2019:

- A site walk on the hill, identifying the old stockpiles, and the various shales facies, where they were outcropping. The deposit was identified to be outcropping on all sides of the hill
- Identification of the civil constructions on the hill, namely:
    - Water reservoir for the town of Kanye
    - Cell phone tower on the Southern part of the hill
    - Concrete road to the cell phone tower
- Identification of the borehole locations
- Inspection of long-standing walls of the pit for stability
- Short inspection of various water sources, including two surface water dams, a water pumping station (in use) for the town of Kanye, and several boreholes (not in use)
- Identification of the location of a 33kV electricity substation 11.2 km from the site
- Short visit to the manganese deposit of Otse, some 30km from site, which could have an impact on the location of the plant
- Inspection of the drill core in the core shed at the local Giyani office in Gaborone

1.5 References

1) Mineral Resource Estimate for the K-Hill Manganese Project, NI 43-101 Technical Report, MSA Group (Pty) Ltd; November 2018.

2 GEOLOGY

SRK Consulting was not asked to review the MSA NI 43-101 Report (the "Report") that was prepared by MSA Group (Pty) Ltd in November 2018. For a full understanding of the geology of the deposit, the reader is referred to that report. However, for context SRK presents a short summary of that report here.

SRK did perform a short geological review of the block model to ensure that it was fit for purpose for the compilation of a mine plan but carried out no checks on the underlying data or QAQC.

2.1 Geological Overview

The text in this section is taken from the Report.

2.1.1 Regional Geology

The stratigraphy in the K-Hill area consists predominantly of late Archean to early and middle Proterozoic rocks from the Ventersdorp (meta-volcanics) and Transvaal (meta-sedimentary) Supergroups, as well as the early Precambrian Gaborone Granite (intrusives) and later Waterberg (sedimentary) Groups (Key and Ayres, 2000). The prospect occurs within the mapped Transvaal Supergroup sediments consisting of shales, quartzites, limestones and conglomerates. In the Kanye area, only the lower parts of the Transvaal succession, the Black Reef Quartzite Formation and the Taupone Dolomite Group, are present.

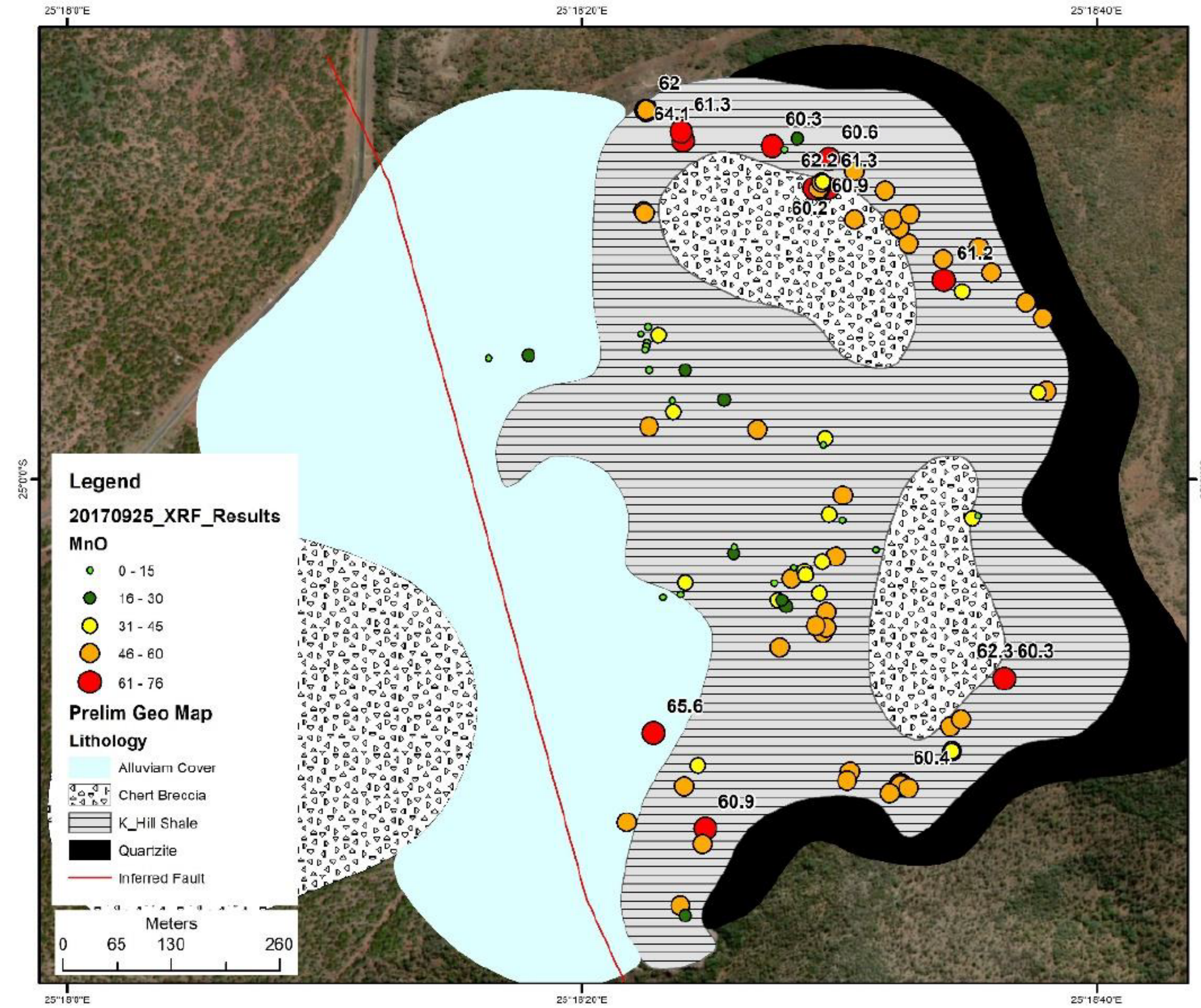
2.1.2 Local Stratigraphy

The mineralization at Kgwakgwe Hill is primarily associated with the upper shale horizon of the Black Reef Quartzite Formation. The quartzite package underlying the shales, rests unconformably on Archaean felsites of the Kanye Volcanic Group. The shales in turn are overlain by the chert breccias of the Paupone Dolomite Group (Figure 7-2), which suggests non-deposition of the intervening dolomites or a massive unconformity.

2.1.3 Property Geology

Mapping delineated two broad areas of enriched mineralization within the shale separated by an area of low manganese values. The northerly portion is elongated northwest over an area of approximately 400 m by 300 m and the average thickness of the mineralization in this area is 3.5 m.

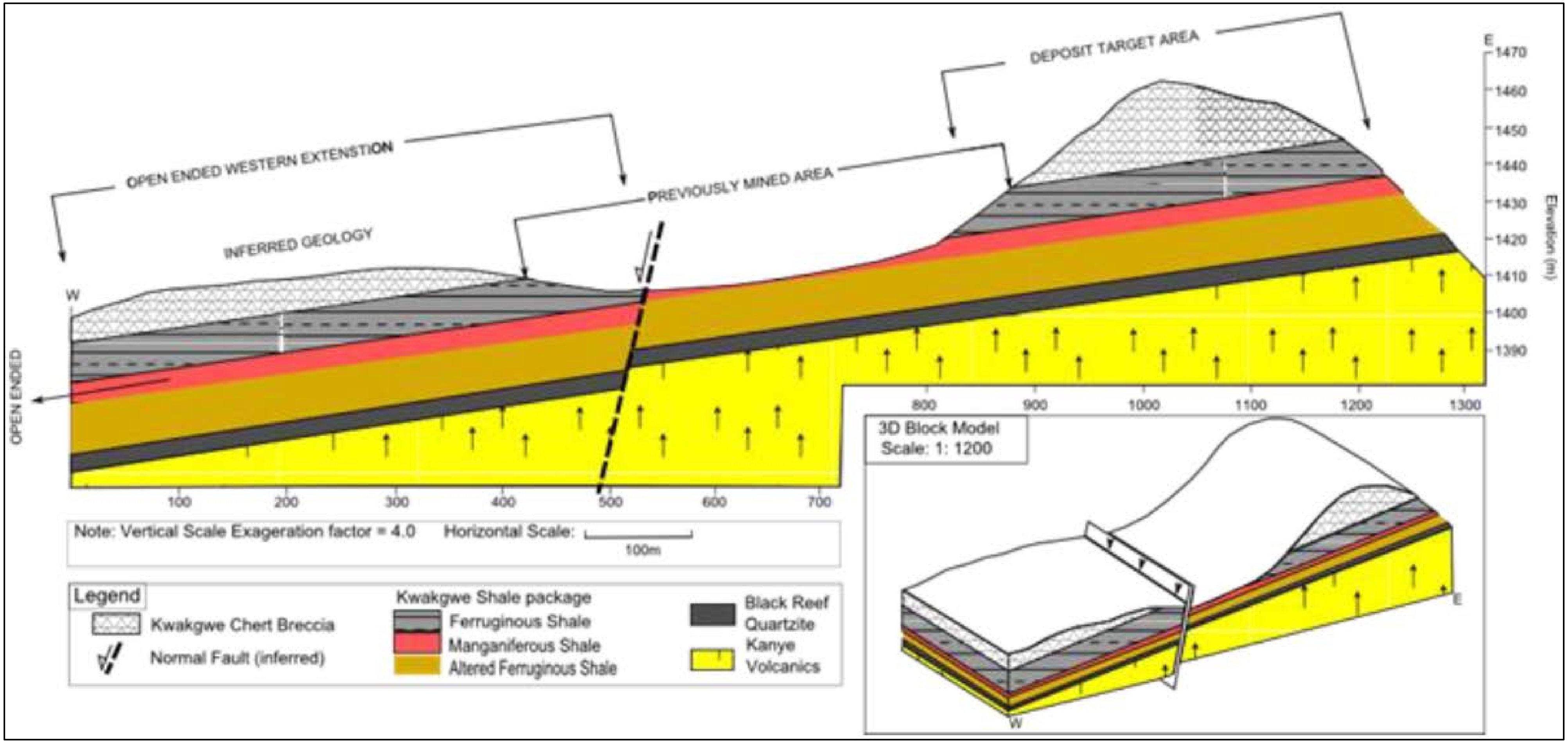
The southerly area of mineralization is elongated northeast over an area of approximately 570 m by 200 m, and has an average thickness of approximately 2.0 m. The manganese shale also outcrops along the easterly scarp slope of Kgwakgwe Hill.



Source: Theron, 2017

Figure 2-1: Simplified geological surface map for K-Hill

A schematic section, looking North, illustrating an interpretation of the geology at K-Hill is shown in the figure below. The mapped and sampled mineralisation is shown on the Eastern side, with potential mineralisation to the West.



Source: Theron, 2017

Figure 2-2: Conceptual cross-section through K-Hill

2.2 Geological Model review

2.2.1 Model Description

The block model supplied, *khill_class3.dm*, was supplied in Datamine format. The parent blocks are 50x50x5 m, with sub-blocking down to 12.5x12.5x1 m. A total of 22,469 blocks were stored in the file.

Although many types of metals were recorded in the model, only MnO was considered for economic value. Table 2-1 shows the most important block model attributes for this Study.

Table 2-1: Block Model Attributes

Attribute Name	Description
al2o3	Aluminium Oxide
cao	Calcium Oxide
class	Mineral Classification (0 = unclassified, 3 = inferred)
cr2o3	Chromium (III) Oxide
domain	Geological Domain
estdom	Estimation Domain
fe2o3	Iron Oxide
k2o	Kalium Oxide
mgo	Magnesium Oxide
mno	Manganese Oxide
n2o	Nitrous Oxide
na2o	Sodium Oxide
nsam_mno	Number of samples for estimating MnO
nsamp	Total number of samples
p2o5	Phosphorus Pentoxide
sg	Density
sio2	Silicon Dioxide
tio2	Titanium Dioxide
v2o5	Vanadium Pentoxide

2.2.2 Topography Data

The topography data used for the block modelling, dating back to August 2018, was rather low density; a 50 x 50 m grid which for the steep sides of the hill gives a lower accuracy. A new topographical survey was flown using drone technology, but it has proven to be incompatible with the existing block model data in local areas. SRK has recommended that the new topographical data should be used when updating the resource model, but has used the block model as it was created by MSA.

2.2.3 Drill hole Database

The complete drill hole database was received from the Company. No major flaws were found.

2.2.4 Wireframe data

The surfaces used to model the various horizons were received from the Company. No major flaws were found.

2.2.5 Model Review

SRK reviewed the data supplied and performed checks on resource tonnage and grade. The Report was reviewed. The Resource is classified as Inferred which is considered to be appropriate for the current level of data according to SRK. The current classification is due to low confidence in the following data:

1. The lack of accuracy in the surface topography.
2. Presence of the artisanal workings. The location of these workings can be mapped, but due to the age, small size, and stability issues volume determination of these workings is not practically possible. SRK agrees with the Client that the impact on the resource is small and an expensive investigation is not warranted.
3. Relatively poor core recovery, averaging 50% - this is mitigated by the tendency for higher MnO grade to be associated with higher core recovery so low core recovery is expected to understate grade.
4. Poor geostatistical resolution and use of IDW and assumed search parameters for grade estimation.
5. Irrespective of the estimation domain, identical search and estimation parameters were applied to all variables. In other words, sample grade was not shared across estimation domains.
6. Density appears to be variable but averages 2.42 t/m^3 overall for the model. It is important to note the comment made by MSA on the Resource Table in the NI 43-101 document. Tonnages are reported as wet, due to density measurements not completed on oven-dried core.

It is noted that the MSA block model is orientated North – South, which is oblique to the drilling pattern. A reasonably coarse sub-block size was used, and as a result the blocks for thin units do not always maintain continuity inside the wireframes. This is considered unlikely to give any material issues.

SRK considers the model fit for purpose with its current classification of Inferred.

3 GEOTECHNICAL

3.1 Data received

There are no holes with geotechnical logging at K-Hill at this stage.

3.1.1 Site Visit Observations

Onno ten Brinke, Principal Consultant (Mining Engineering) visited the site on 20th of March, 2019. Observed wall stability for the long-exposed walls at K-Hill has been very good. The exposed shale wall that must have been mined around 1971, still stands without any visible problems. The hanging wall, although fractured, also stands up stable.

The pit designs generated in this study have shown that the steepest walls, mined into the hill, require a ramp for access and as such the overall slope angle for the highest walls is reduced.

Source: SRK 2019

Figure 3-1: Long-standing wall of Manganiferous shale (i.e. potential ore), ca. 12m high

Source: SRK 2019

Figure 3-2: Typical fractures on hanging wall at K-Hill

3.2 Recommended Slope Configuration

Based on the on-site observations, the low annual rainfall, and the relative low height of the permanent walls that are expected, an inter-ramp angle of 45^0 can be assumed to be easily achievable, despite the lack of concrete data. In future pre-feasibility or feasibility studies, a steeper wall is likely to be possible using more accurate data to support this.

Considering the low mining rate and the accompanying small(ish) equipment, a bench height of 4 m is recommended, to be mined in two flitches, each 2 m in height. The onsite observations support a 70^0 batter angle. Using an overall inter-ramp slope angle of 45^0, and a bench height of 4 m, this results in a berm width of 3.2 m.

Table 3-1: PEA Level Slope Configuration

Description	Value
Inter-ramp angle	45^0
Bench height	4 m (mined in 2m flitches)
Batter Angle	70^0
Berm width	3.18 m

4 HYDROLOGY AND HYDROGEOLOGY

4.1 Data received

There are no groundwater monitoring boreholes at K-Hill at this stage and SRK is not aware of any hydrogeological investigation being carried out in the past. No hydrogeological information is available at this stage of the project.

4.2 Hydrogeology and groundwater

The hydrogeology of a mine can exert certain constraint on the project development, related to the following aspects:

- **Water inflow into the mine**: which can have impact on mine production. considering the geology of the site and limited catchment, it is likely that water inflows into the pit will be manageable. Field data should be collected from hydrogeological boreholes in the next stage of the Project to predict the potential inflows and specify dewatering system for the mine
- Pore pressure in the pit walls and impact on pit slope stability. considering the rock strength and arid climate, as reported in the geotechnical section of this report, it is likely that pore water pressure would have limited (or negligible) control on the pit stability. Such hypothesis should be confirmed in the next stage of the Project with field data and due analysis and modelling
- Impact on water resources: depending on the mineralogy and groundwater quality in the area, the development of the mine may lead to impact on water resources in the area, and downstream if contaminants are present and not controlled during mine operation and closure. Therefore, detailed studies of such aspect will likely be required to obtain relevant environmental and water use permits for the project development
- Water supply for the Project. The Project water balance depends on the availability of water within the site (inflows of groundwater and surface water within the Project site). If enough water is available within the site, limited or no water supply will be required. Considering the climate of the region however, water supply will likely be needed. The availability of groundwater in the mine area will need to be assessed in detail and based on field data in the next phase of the Project.

4.3 Surface Water

As shown in Figure 4-1 and Figure 4-2, K-Hill site is located along a hydrological catchment divide. The Project catchment surface water drains into streams that flow into the Taung river, at a close distance downstream of the Mogobane reservoir.

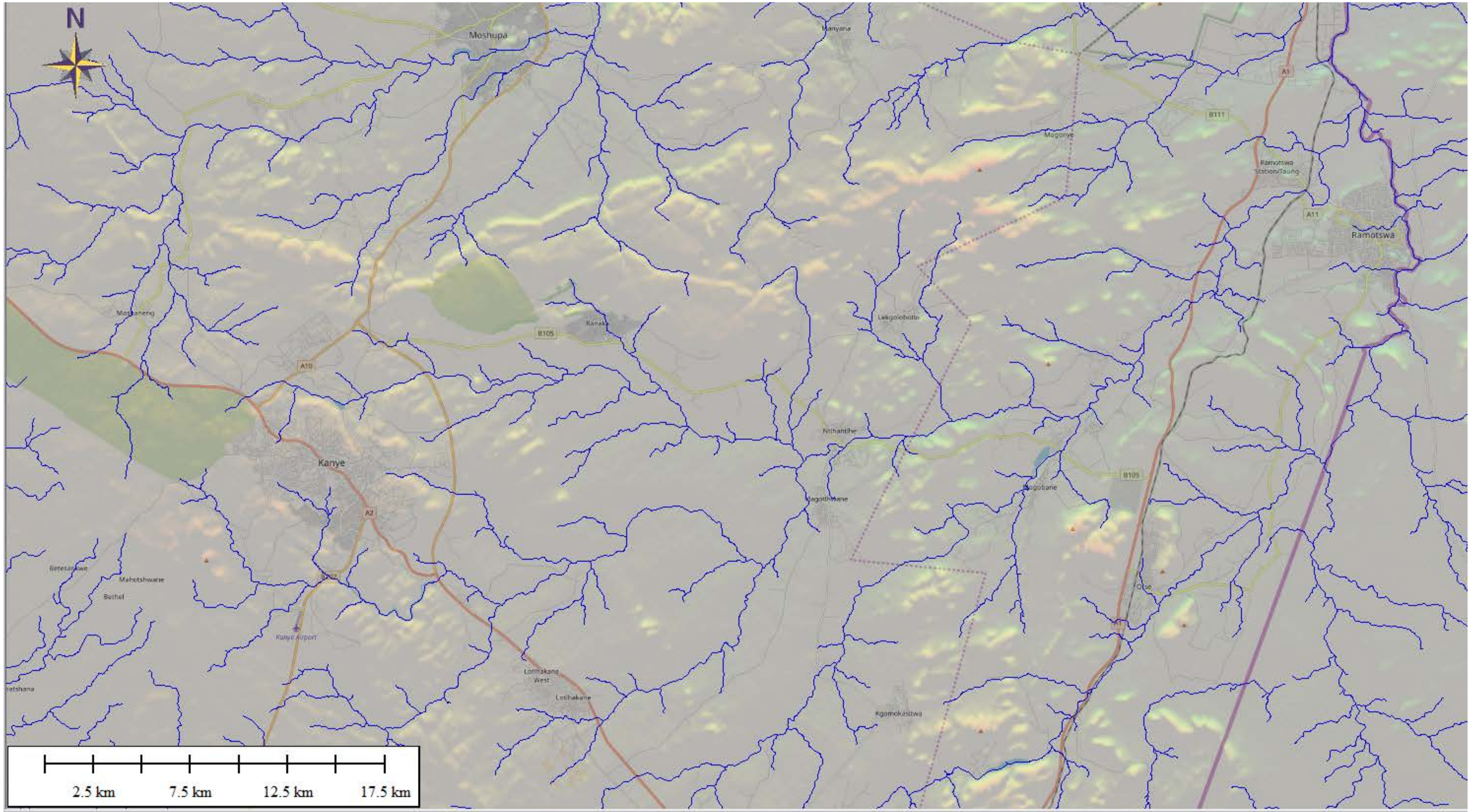


Figure 4-1: Regional Hydrological Catchments and Streams

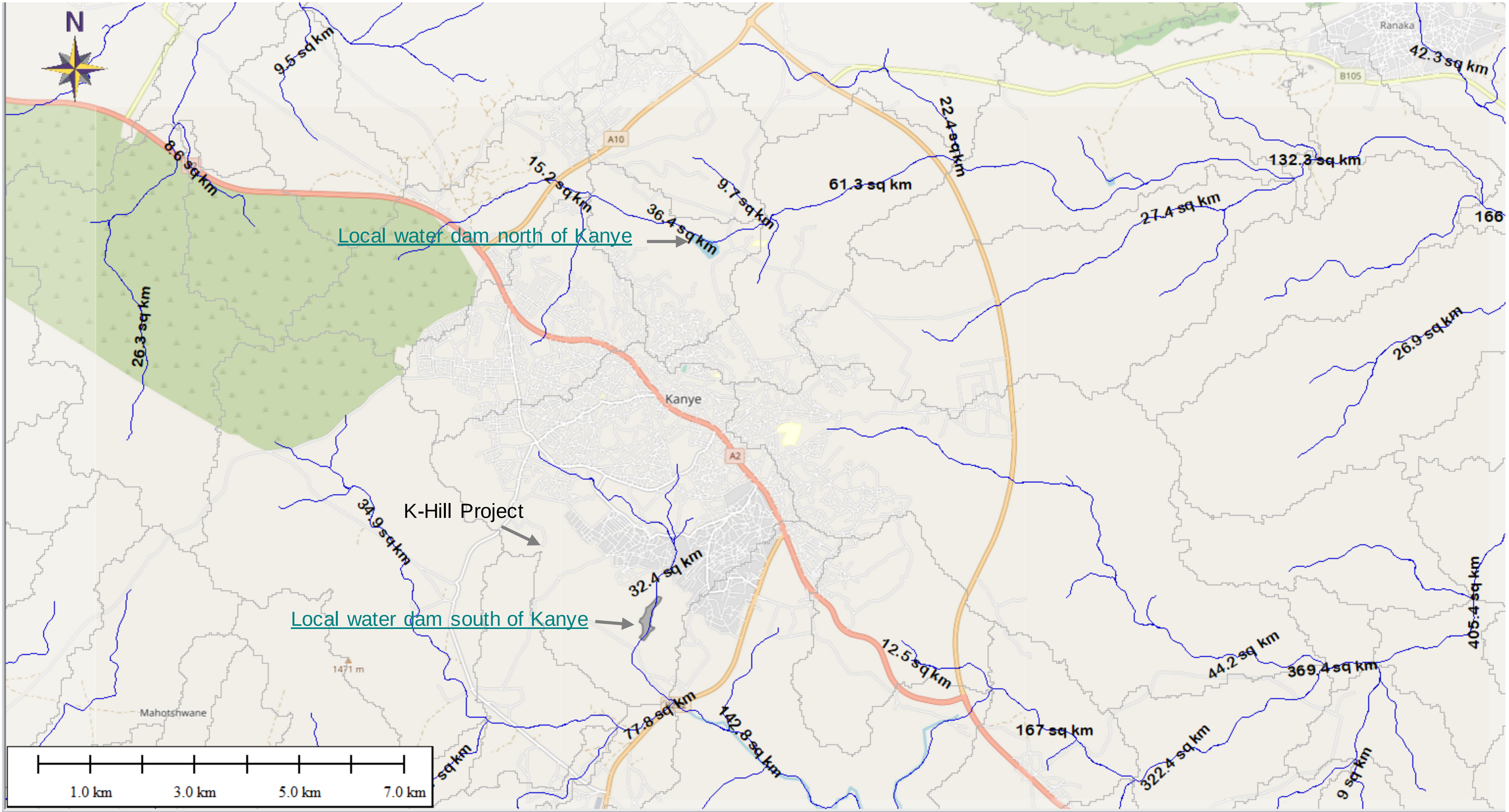


Figure 4-2: Local Streams and Hydrological Catchments

The Taung river flows northerly to join the Ngotwane river which feeds and flows through the Gaborone reservoir, just south of the Capital city (Gaborone) of Botswana. The Ngotwane river originates in South Africa and flows along the border between the two countries before reaching the Capital of Botswana. Downstream (north) of the latter it flows northerly within Botswana before joining the Limpopo river that flows along the border between the two countries for more than 50 km before reaching Zimbabwe. The Limpopo River continues its journey along the border between Zimbabwe and South Africa then through Mozambique toward the sea (Mozambique channel).

Therefore, water drainage from the propose K-Hill Project site is part of a complex trans-boundary surface water drainage system, which feeds several reservoirs and eco systems. Due to such hydrological setting, it is likely that detailed environmental and social impact assessment will be required in the next stage of the project development, especially if international funding is needed for the Project. The Project should therefore target zero water discharge policy by making maximum use and recycling of water within the Project area.

Source: SRK 2019

Figure 4-3: View of the local surface water reservoir south of Kanye, 2.5 km from K-Hill

4.4 Project Water supply

It is understood that only limited amount of water is needed for processing at K-hill site. Based on information from other projects, it appears that water supply for the Project will vary between 0.2 and ~0.6 m^3/t. At the proposed production rate of 150 ktpa, this equates to about 4 -11 m^3/hr water demand for processing.

The option of water supply from local reservoirs has been envisaged in project team discussions, however, it appears that water from the reservoir is limited, and this is confirmed from the limited hydrological catchments of the streams as illustrated in Figure 4-2, which show that the area of the catchment for both reservoirs ranges between 32 and 36 km^2.

Considering the limited amount of water required for the Project, it is envisaged to use a local pumping well to supply water. Depending on the size and hydraulic properties of the target aquifer a single borehole maybe able to provide the required amount of water.

If we consider that only one pumping well is needed and the latter will be located in close proximity to the plant, the cost of drilling and equipment of such well may vary between ~USD60,000-USD150,000 depending on the depth and diameter of the borehole.

5 MINING

5.1 Introduction

The envisaged mining method for the K-Hill Project is traditional truck and shovel. Due to the low processing throughput, and reasonable strip ratio, the volume of total material moved (TMM) is easily manageable. For the mining part of this PEA, the key tasks that were undertaken are listed below.

5.1.1 Key Tasks

- Analysis of the geological model and adaptation for mine planning purposes
- Definition of key operating cost components, revenue and applicable royalties
- Open pit optimisation to generate a pit shell
- Practical design of the pit including ramp access
- Practical waste dump design
- Layout of haul roads
- Generation of a mining schedule, including pushbacks
- Equipment calculations to determine fleet requirements
- Determination of mining fleet based on similar operations worldwide

5.2 Mining Model Description

The block model provided by the Company was in Datamine format and called "*khill_class3.dm*". The block model and its attributes are described in Table 5-1. Only the MnO grade was used for generating a pit shell, other metals were ignored.

Table 5-1: Block Model Variables and Header

Block Model header				
Name	**khill_class3.dm**			
		X	Y	Z
Minimum Coordinates		7233400	328900	1275
Maximum Coordinates		7234400	330400	1525
User Block Size		50	50	5
Minimum Block Size		50	50	5
Rotation		0	0	0
Total Blocks		22469		
Attributes	Description			
al2O3				
cao				
class	Resource Class (1=Measured, 2=Indicated, 3=Inferred)			
cr2o3				
domain				
estdom	Estimation Domain			
fe2o3				
k2o				
loi				
mgo				
minloss				
mno	Percentage MnO			
n2o				
na2o				
nsam_mno	Number of samples used for estimating MnO			
nsamp				
p2o5				
perfil				
select				
sg	Density			
sio2				
svol				
tio2				
topo				
trdip				
trdipdir				
v2o5				
whittle	Rock code for exporting to Whittle			
zone				

5.2.1 Grade-Tonnage Curve

Generating a Grade-Tonnage curve for the resource tonnage from the MSA block model, SRK notes that 87% of the resource has a grade above 20% MnO, and the lowest grade in the resource is 11% MnO.

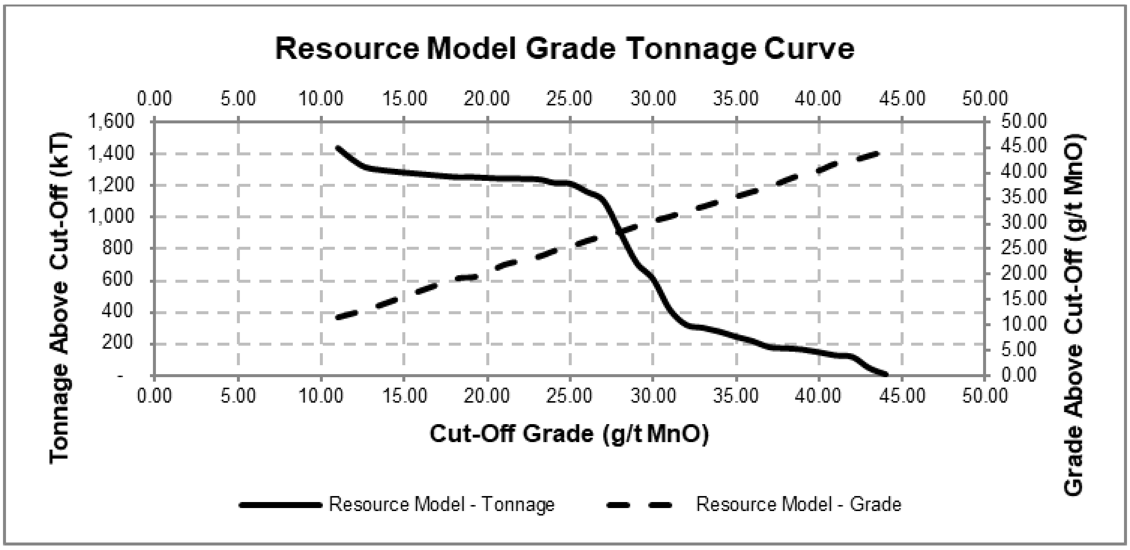


Figure 5-1: Resource Model Grade-Tonnage Curve

5.3 Open Pit Optimisation

Pit optimisation was done using the industry-standard Whittle software package, which uses a Lerch-Grossman type optimisation algorithm.

5.3.1 Pit Optimisation Parameters

The pit optimisation parameters are shown in Table 5-2 and are discussed below, as far as they have not been discussed previously in this document.

The mining cost was calculated based on the S&P database for published 2018 mining costs for similar small-scale mining operations around the world, as well as a similar-sized SRK client operation in Mozambique. A total of 6 operations was considered and a weighting was applied to the relevance of the operation.

Dilution and recovery were estimated based on similar results achieved on relatively small scale equipment and taking into account the well-defined shale horizon. It is anticipated that the given figures might be improved upon in a well-run operation.

The processing recovery used in the pit optimisation was based on initial results from the metallurgical test work on leaching. This figure has dropped after results for electrowinning and refining have been introduced. Due to time constraints the pit optimisation work was not redone, however a test run with the lower recovery of 87.5% revealed that the pit shell stayed the same. This can be attributed to the high margin that is achieved on this Project. From about USD1500 /t of MnO (this is equivalent to USD1950 /t EMM), the entire resource is mined.

Processing cost is high due to the electrowinning. Separation of the Mn requires a lot of electricity, estimated at 6,800 kWh/tonne processed. This is a similar electrical requirement as published for the Euromanganese Chvaletice Project.

The sale price of U SD4700/t EMM has been assumed by the Client for a 99.9% pure EMM product. SRK notes that the lower grade product of 99.7% EMM carries a price of USD2500/t EMM. It should be noted that the entire resource was mined in the pit optimisation, regardless of the price assumed.

General and Administration cost is high, expressed per tonne, for a project with relatively small throughput. The Client estimates that the mine should be able to operate comfortably at

USD3 M/yr. Royalty in Botswana for Manganese is 3%. The Cut-off Grade ("CoG") calculated for a selling price of USD4700 /t EMM is 8.81% mill feed, which means 9.26% in-situ grade. Referring to the Grade-Tonnage curve in Figure 5-1, we find that the CoG is below the minimum grade in the resource. The CoG for USD2500 /t EMM (99.7) is 17.40% in-situ. This means only 8% of the resource falls below the CoG for USD2500 /t EMM.

Table 5-2: Pit Optimisation Parameters

Parameters	Units	Base Case	Basis
Production			
Production Rate - RoM	(tpa)	150,000	SRK Assumption
Geotechnical			
Overall Slope Angle Oxide	(Deg)	45	SRK Assumption
Overall Slope Angle Fresh	(Deg)	45	SRK Assumption
Mining Factors			
Dilution	(%)	5.0	SRK Calculation
Recovery	(%)	95.0	SRK Calculation
Processing			
Recovery MnO	(%)	94.0	Client provided
Operating Costs			
Mining Cost	(USD/t_{rock})	3.14	SRK Calculation
Incremental Mining Cost	(USD/ 1m bench)	0.01	SRK Assumption
Reference Level	(Z Elevation)	1385	
Replacement Capital	(USD/t_{RoM})		Not used
Rehabilitation Cost	(USD/t_{RoM})		Not used
Processing	(USD/t_{RoM})	250.00	SRK Calculation
Selling Cost Mn	(%)	3.0	Botswana
	(USD/t)	141	
G&A	(USDm/Year)	3,000,000	Client assumption
	(USD/t_{RoM})	20.00	
Metal Price			
EMM	(USD/t)	4,700	Client provided
Other			
Discount Rate	(%)	10	
Cut-Off Grade			
Marginal	(% MnO)	8.81	SRK Calculation

5.3.2 Pit Optimisation Footprint Constraints

No specific footprint constraints have been applied in the pit optimisation process.

5.3.3 Pit Optimisation Results and Shell Selection

Figure 5-1 shows the metal price sensitivity results from the pit optimisation. Table 5-3 shows the pit optimisation results.

Whittle has not generated any shells between the EMM prices of USD4042 /t and USD5734 /t as all these prices allow Whittle to mine almost everything in the resource. The USD5734 /t threshold allows an additional 5 kt of manganese RoM to be mined, but in principal we can see in the graph that from a price of USD3000 /t EMM there is very little difference in the generated pits.

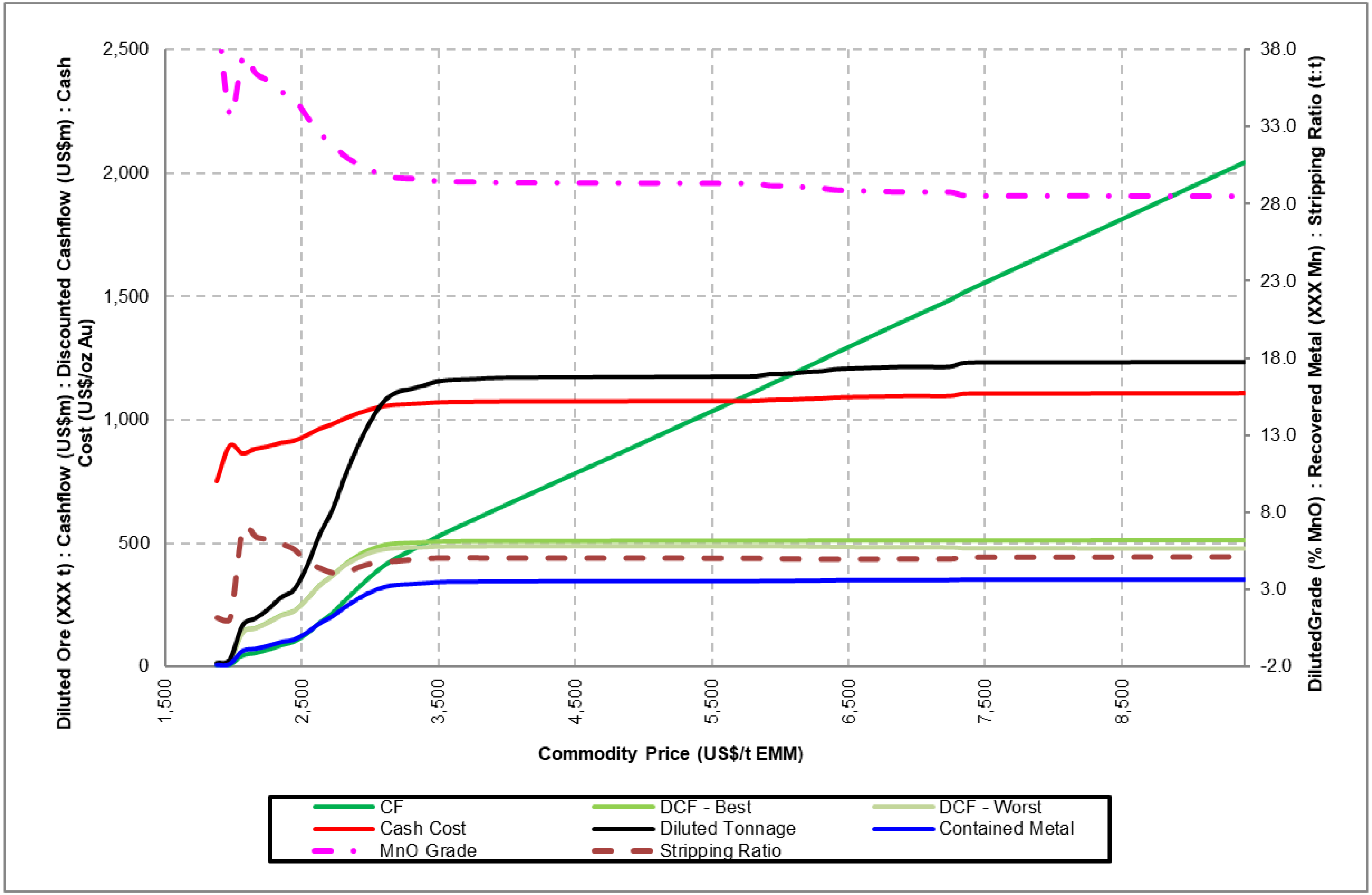


Figure 5-2: Metal Price Sensitivity

Table 5-3: PIt Optimisation Results

Optimisation Results	EMM Rev Factor Units	2068 USD/t 0.44	2538 USD/t 0.54	3572 USD/t 0.76	4042 USD/t 0.86 4700 USD/t equivalent	5734 USD/t 1.22
Insitu						
Inventory	(kt)	157.9	384.8	1,106.6	1,115.5	1,120.0
	(% MnO)	41.49	37.18	32.57	32.49	32.44
	(t MnO)	65,512	143,082	360,410	362,417	363,331
Modifying Factors						
Mining Dilution	(%)	5.0	5.0	5.0	5.0	5.0
Dilutant	(% MnO)	0.0	0.0	0.0	0.0	0.0
Mining Recovery	(%)	95.0	95.0	95.0	95.0	95.0
Au Metallurgical Recovery	(%)	0.94	0.94	0.94	0.94	0.94
Diluted						
Inventory	(kt)	165.8	404.0	1,161.9	1,171.3	1,176.0
	(% MnO)	37.53	33.64	29.47	29.39	29.35
	(t MnO)	62,237	135,928	342,389	344,296	345,164
	(t EMM)	48,202	105,276	265,180	266,657	267,330
Quantities						
Total Rock	(kt)	1,330.9	2,332.8	7,009.2	7,061.5	7,082.0
Mineral Inventory	(kt)	165.8	404.0	1,161.9	1,171.3	1,176.0
Waste + OM	(kt)	1,165.1	1,928.8	5,847.3	5,890.2	5,906.0
Waste	(kt)	1,165.1	1,928.8	5,847.3	5,890.2	5,906.0
Inventory (Below C/O)	(kt)	0.0	0.0	0.0	0.0	0.0
Stripping Ratio	(t:t)	7.0	4.8	5.0	5.0	5.0
Operating Expenditures						
Mining	(USD/t_{mined})	3.38	3.39	3.47	3.46	3.46
	(USD/t_{RoM})	27.16	19.60	20.90	20.88	20.85
Rehabilitation Cost	(USD/t_{RoM})	0.00	0.00	0.00	0.00	0.00
Processing + G&A	(USD/t_{RoM})	270.00	270.00	270.00	270.00	270.00
EMM Selling Cost	(USD/t_{metal})	20	20	20	20	20
Total Cash Cost	(USD/t_{metal})	317	310	311	311	311
Product						
Recovered Metal (EMM)	(t EMM)	45,310	98,960	249,270	250,658	251,290
Economic Summary						
Metal Price	(USD/t)	2,068	2,538	3,572	4,042	5,734
Revenue	(USDm)	94	251	890	1,013	1,441
Mining Costs	(USDm)	5	8	24	24	25
Processing Costs	(USDm)	45	109	314	316	318
Selling Costs	(USDm)	1	3	6	6	6
Other Costs	(USDm)					
Cashflow	(USDm)	43	132	546	666	1,092
Discount Rate	(%)	10.0	10.0	10.0	10.0	10.0
Mill Rate	(ktpa)	150.0	150.0	150.0	150.0	150.0
DCF - Best Case	(USDm)	139	272	507	508	509
DCF - Worst Case	(USDm)	139	273	489	490	491
Project Life	(years)	1.1	2.7	7.7	7.8	7.8

The shell selected for the basis of pit design is the shell with a revenue factor of 0.86, which is the same shell as one generated for a revenue factor of 1.0, therefore corresponding to the optimal shell for an EMM price of USD4,700 /t.

5.4 Pit Design

5.4.1 Pit Design Parameters

The overall inter-ramp slope angle used was 41^0, as explained in section 3.2. To accommodate small equipment and reduce dilution, mining is done on a 4-metre bench height, mined in 2 flitches of 2 metres. The face angle is 70^0 based on observations in the field. This gives a standard berm width of 3.2 m. No catch berms are considered necessary due to the relatively low height of the permanent walls. In most walls, the slope angle is considerably lower than the recommended inter-ramp angle, due to the placement of the ramps on the footwall, going into the hillside.

The pit optimisation shells indicate that the pits will have relatively a large amount of room on the pit floor with good access. Ramps are relatively short and in most instances two ramps can be accommodated without adding too much waste, thereby allowing for a drive-in ramp and a drive-out ramp. Considering the size of the equipment, such as the Caterpillar 725C2 articulated trucks (or equivalent), a ramp width of 10 m was chosen.

The clearing radius of the Caterpillar 725C2 is 8.05 m, and this has been taken into account in the pit designs. The final pit design can be seen in Figure 5-5.

5.5 Waste Dump Design

The Waste Dump design has been done on industry standard parameters, using 37^0 face angles, and an overall slope of 23^0. Bench heights used were 20 m, and ramps were created at 30 m wide to allow easy access and reduce risk of collision.

The Waste Rock Dump ("WRD") was designed on the required waste dumping volumes, assuming a 30% swell factor. Figure 5-6 shows the WRD position and layout.

5.6 Strategic Assessment

Initial strategic schedules in Whittle using the Milawa Balanced option on an annual basis indicated that a mining limit of 1.2 Mtpa should be enough to keep the plant full. This is shown in the Whittle schedule graph. However, closer analysis of mining on a quarterly basis, as well as the increased waste of the practical pit designs, revealed that accessing RoM does require some opening-up of the pit floor and for this reason the final practical schedule mines more waste tonnes in the early years. Year 1 uses 3 months of pre-stripping and Year 4 has a peak total material movement of 1.95 Mt.

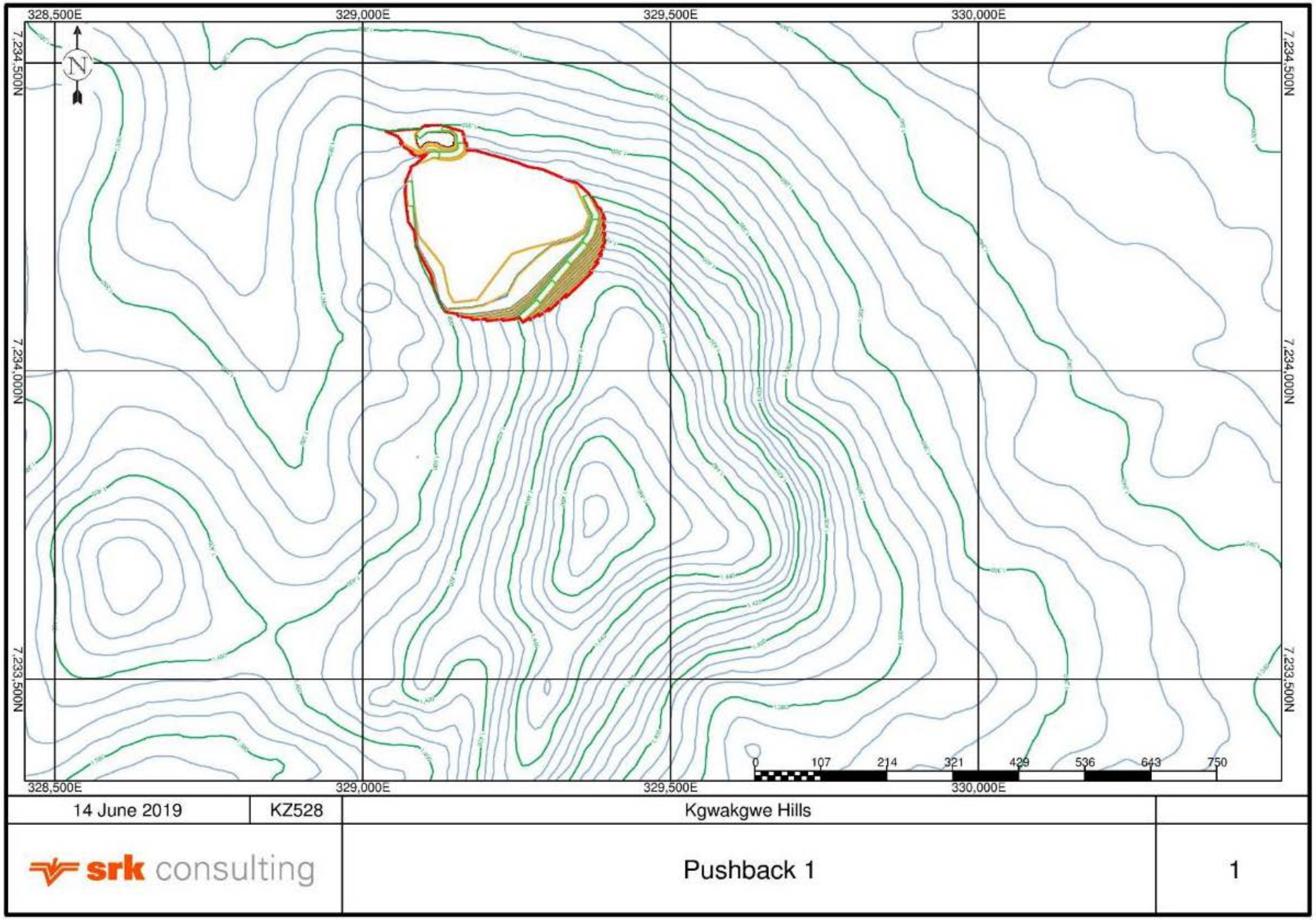


Figure 5-3: Pit design of Pushback 1

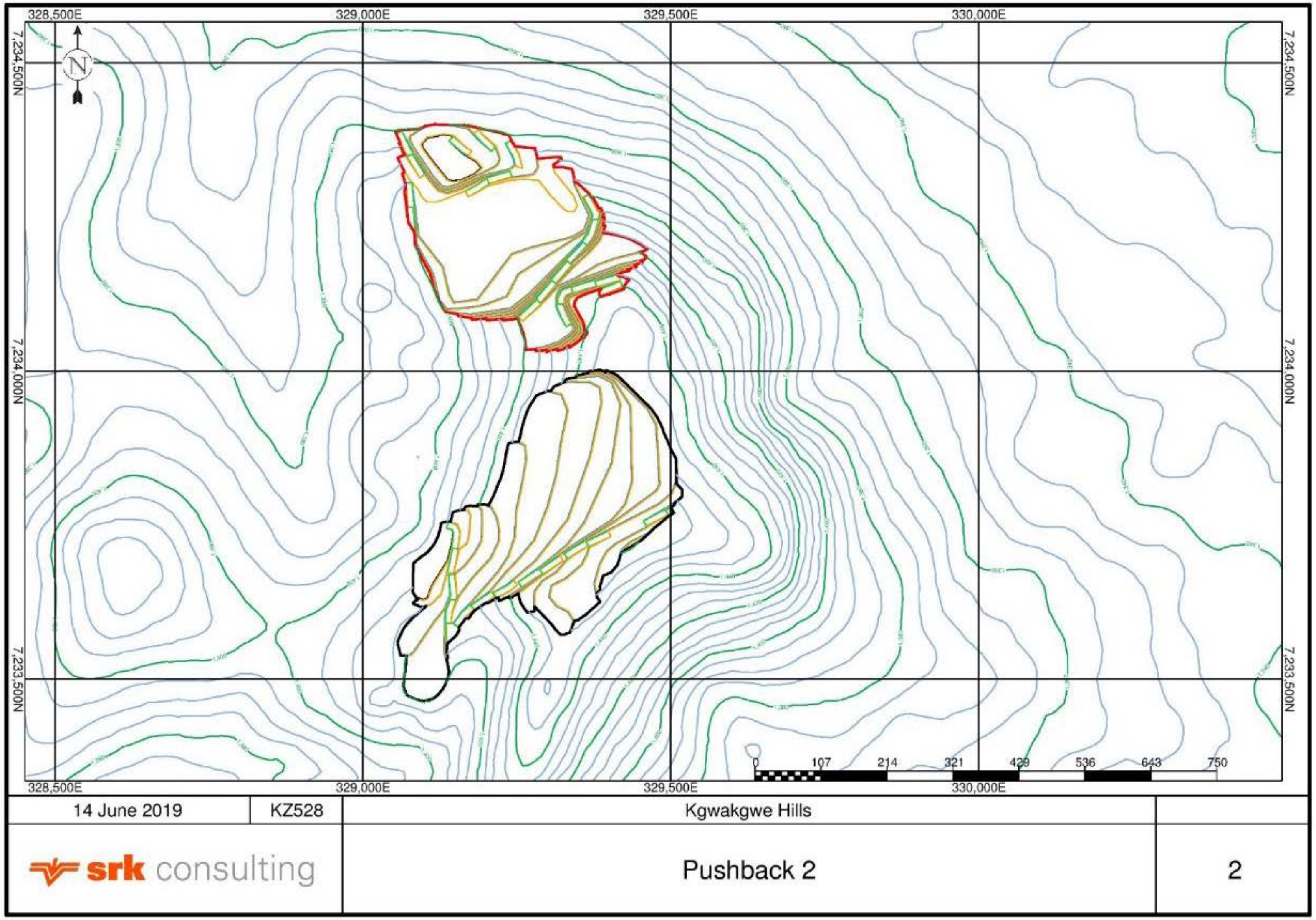


Figure 5-4: Pit design of Pushback 2

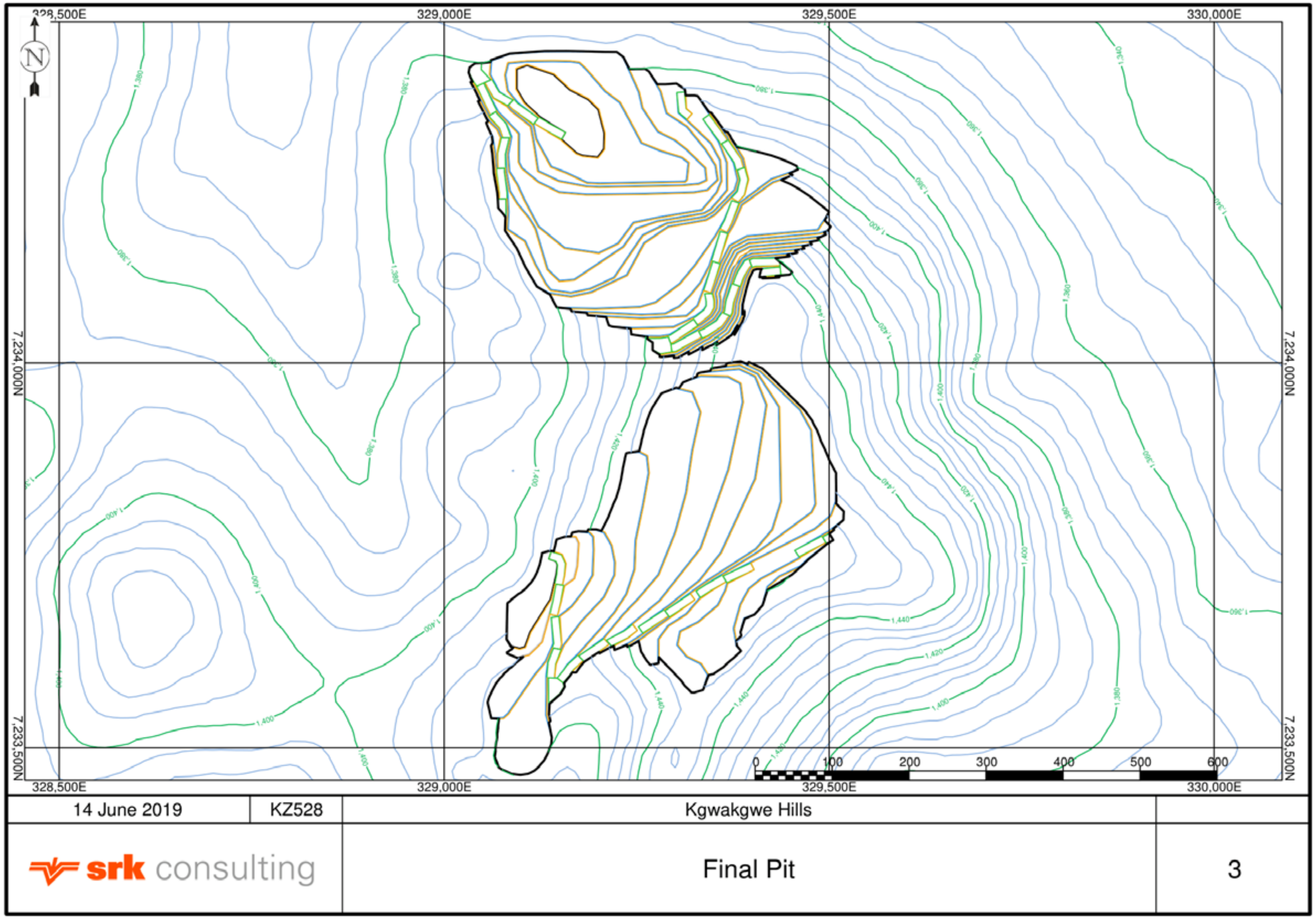


Figure 5-5: Final Pit Design

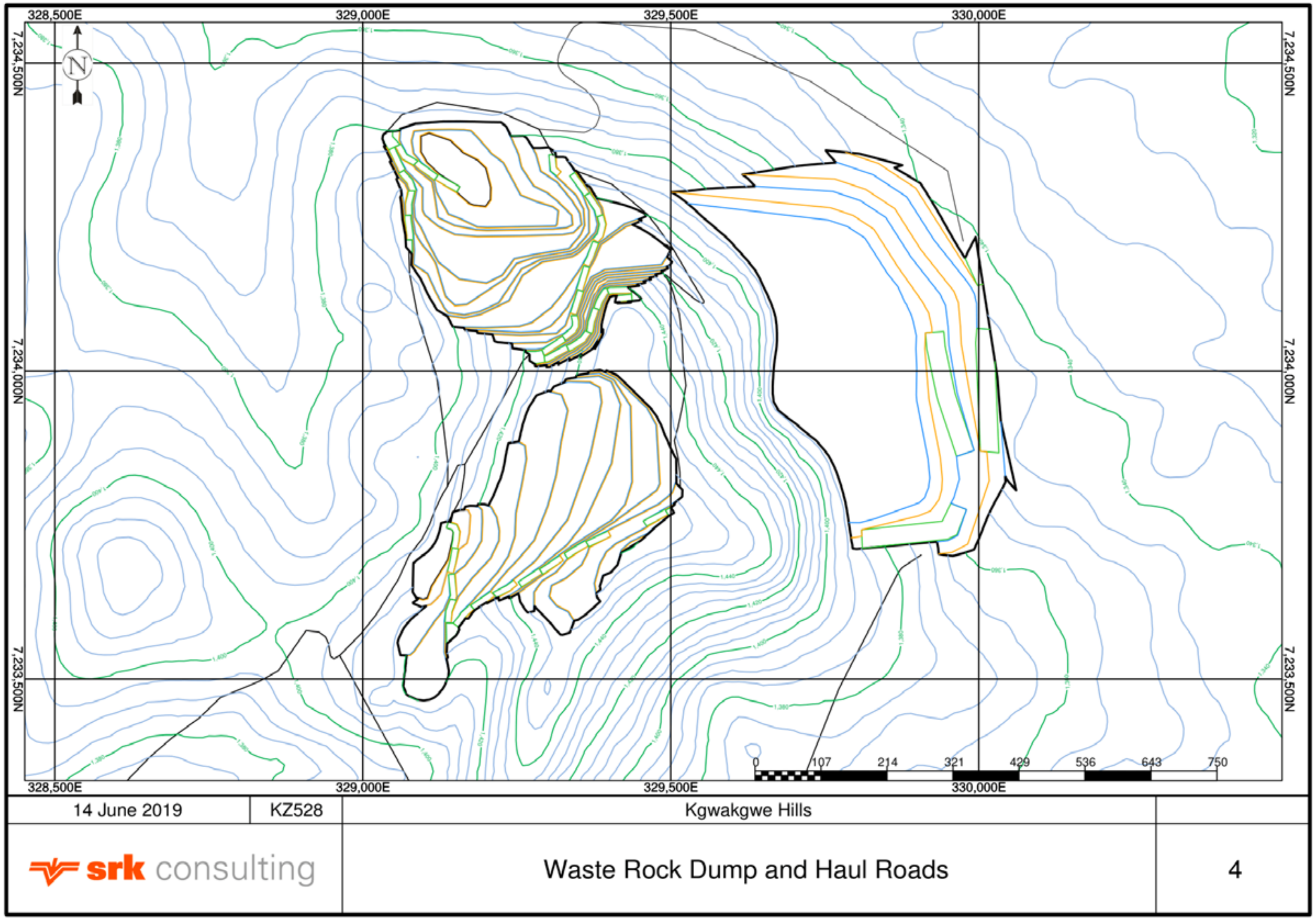


Figure 5-6: Waste Rock Dump

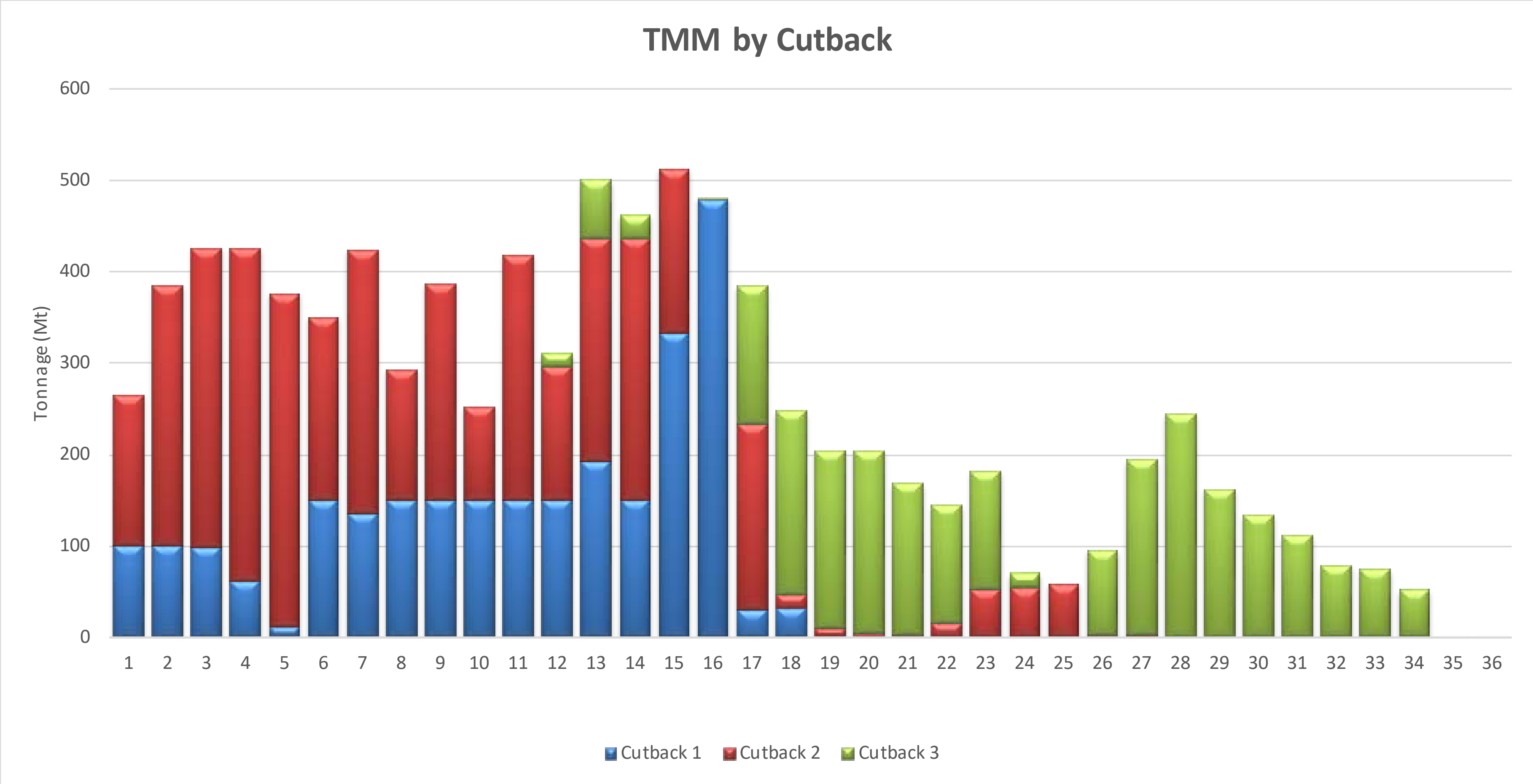


Figure 5-7: Total Material Movement by Cutback on a quarterly basis

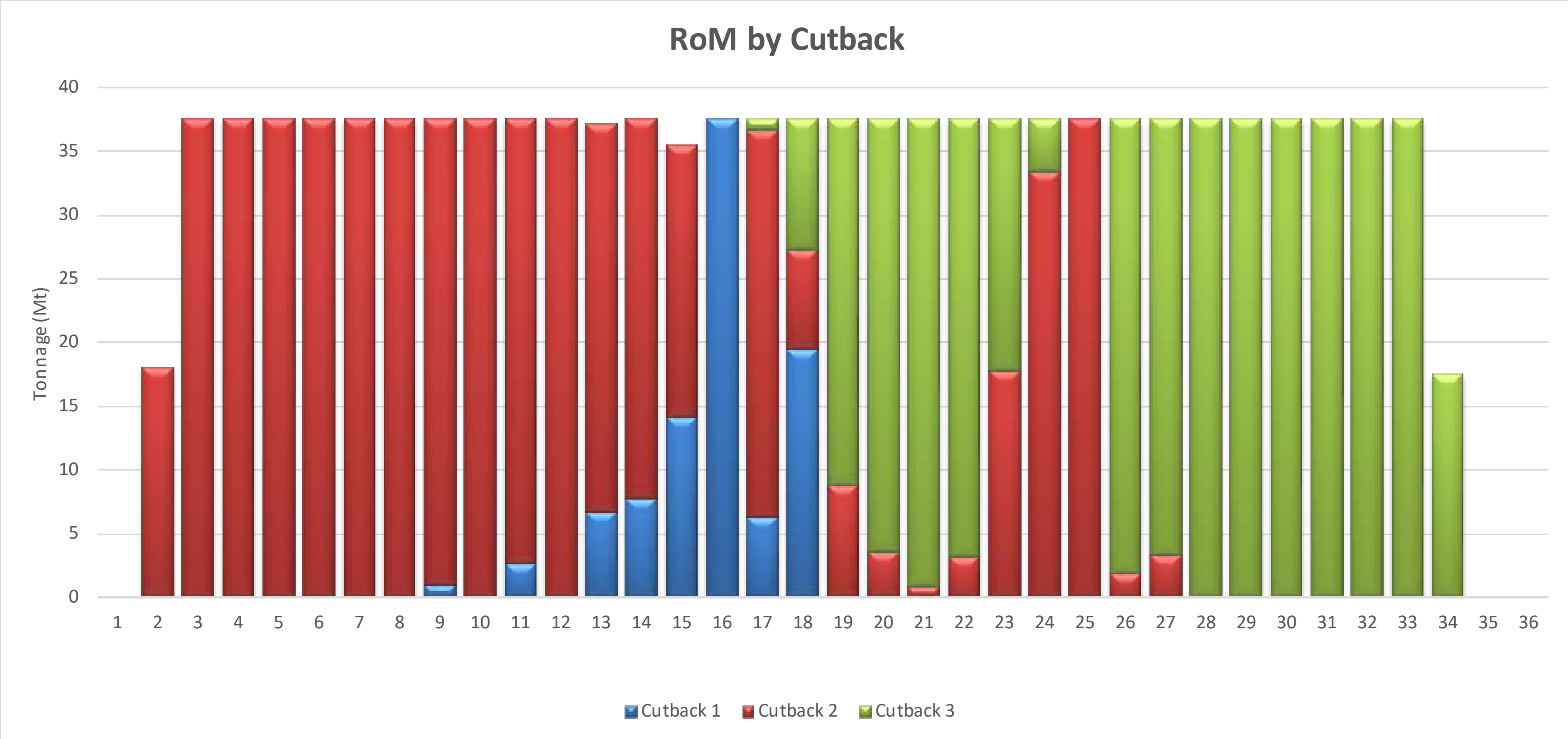


Figure 5-8: Run-of-Mine RoM Tonnage to plant, split by Cutback, by quarter

5.7 Pushback Sequence

The Whittle shells were used to determine a strategic schedule with pushbacks (also known as cutbacks) to balance the waste and RoM stripping. The chosen Whittle shells on which the pushback designs were based are 7, 11, and 22, which contain respectively 2.0, 3.9 and 7.0 million tonnes.

Figure 5-7 and Figure 5-8 show the total material moved from each cutback on a quarterly basis, and how much RoM is delivered to the plant from each cutback.

5.8 Mining Schedule

The mining schedule for total material movement is shown in Figure 5-9 below.

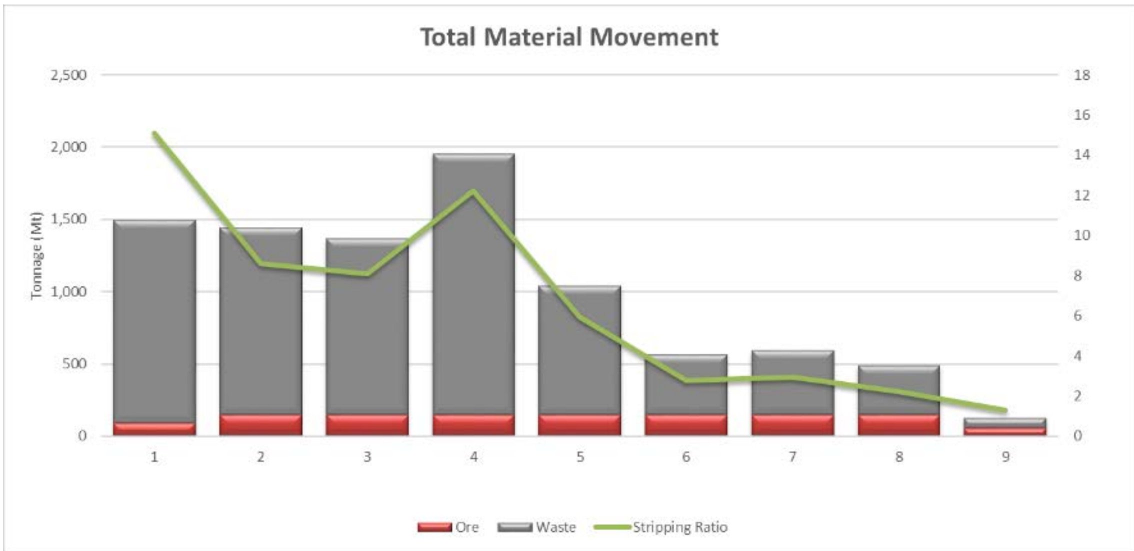


Figure 5-9: Total Material Movement for K-Hill

The schedule shows a stripping hurdle in Year 4, but this can easily be overcome by renting 1 additional truck for 1 year. An effort was made to smooth the stripping as much as possible, but mining into a hill made this somewhat difficult. It is anticipated that the next update to the resource model, which should see the two separate pits merge into 1 larger pit, will make it easier to plan the stripping.

5.9 Delivered Mill Feed

The mine schedule assumes that no RoM will be processed in the first quarter of Year 1, allowing the mining sequence to open up the pit floor and allow access to RoM. A start-up quarter for the plant has been envisaged of 18 kt, after which full production of 37.5 kt per quarter is reached in Quarter 3 of Year 1.

The delivery of RoM to the plant is consistent and the grade delivered is quite evenly distributed with some spikes. Notably the grades delivered in Year 4 are higher than the LoM average, as can be observed in Figure 5-10.

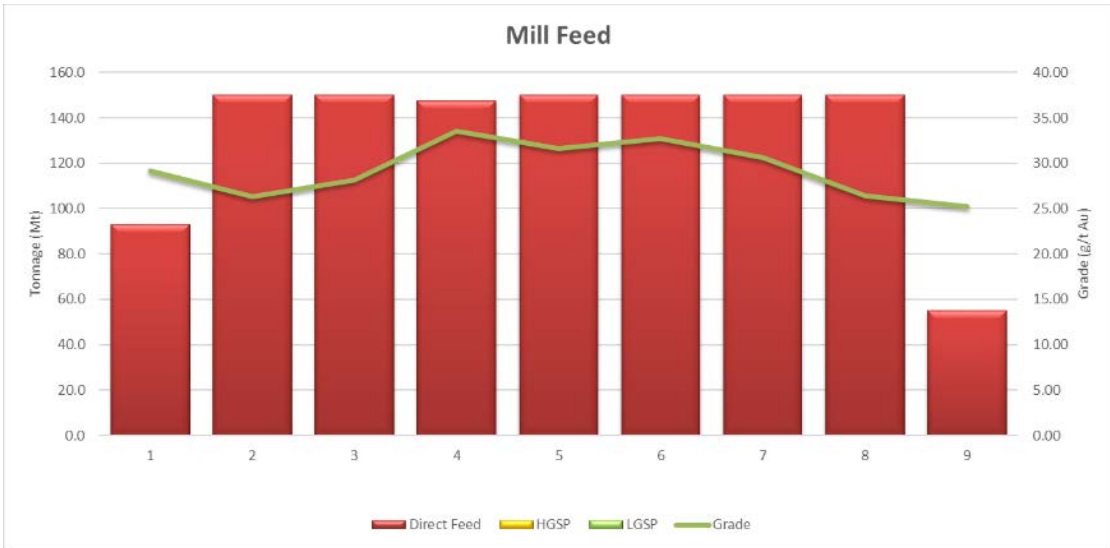


Figure 5-10: Mill Feed

5.10 Cut-Off Grade and Stockpiling Strategy

At this stage of the Project, no work has been done on a stockpiling strategy. The deposit delivers fairly even grade and supplying the plant is not limited by the mining fleet.

A cut-off grade strategy is not worth investigating at this stage, as so little of the deposit has a grade close to the cut-off. There is very good margin on all the RoM material, and since the grade distribution is also quite even, based on current data SRK considers that there is little chance of paying back any additional stockpile rehandling cost through feeding higher grade material earlier in the mine life.

5.11 Equipment

The location of the Project, close to the South African border and the Botswana diamond mines ensures good access to mining equipment, spares and services. There is also a large market of second-hand mining equipment that can be utilised to reduce initial capital expenditure for the mining fleet, and in fact quotes have been sourced by the Company which have been used in the capital estimate.

The PEA assumes that the Project will be owner-operated and that all required equipment will be bought.

Table 5-4 below shows the equipment number and capital estimate for the mining fleet for the K-Hill Project.

The quote for the yellow equipment is from the South African manufacturer BELL, which has a full sale and services branch in Gaborone.

Table 5-4: Mining Fleet Capital estimate

Equipment	QTY	Cost (BWP)	Cost (USD)	Total (USD)	Country Scaling	Total (USD)
Excavator 2,1BCM - Kobelco SK380LC	1	2,558,991	237,163	237,163	1.1	260,880
Backhoe TLB - BELL 315SL	1	891,520	82,625	82,625	1.1	90,887
Articulated Dump Truck - Bell B30E	4	3,641,109	337,452	1,349,809	1.1	1,484,790
Track Dozer - Bell 850J	2	3,510,517	325,349	650,698	1.1	715,768
FEL 2,3 BCM - Bell L1506E	1	1,785,026	165,433	165,433	1.1	181,977
Water Bowser - 18kl	1	2,794,508	258,991	258,991	1.1	284,890
Light Vehicles	4		50,000	200,000	1.0	200,000
Motor Grader - Bell 770G	1	3,446,141	319,383	319,383	1.0	319,383
Lighting plants, radios, misc	1		80,000	80,000	1.0	80,000
						3,618,574
Exchange rate BWP:USD	10.79					

5.12 Haulage Fleet Requirements

The assumption for the mining operation is that work will be done on 340 days a year, using three 8-hour shifts per day.

PEA-level calculations for the load and haul requirements have revealed that one single shovel with a 2.2 m^3 bucket can deal with the total material movement for the LoM. A smaller backhoe will be used for back up to the main excavator, as well as for small jobs and mining RoM whilst the main excavator is mining waste. It is assumed that the excavator will be able to break the ground at site. For the breccia, this will be tough to achieve and a ram should be bought to install on the excavator to break the rocky cap before proper mining begins.

The assumption for haulage calculations assumed an average of 2 km haulage and an on-site speed of 20 km/h for loaded trucks. This means that the operation requires two 30 t articulated dump trucks, with a spare to ensure continuity of operation during maintenance. The additional tonnage required in Year 4 will require one additional truck, bringing the total to 4. It is assumed for the financial model in this study that this truck will be bought, but it could be rented instead.

Due to the low truck requirements after Year 4, it is considered reasonable to not budget for a rebuild. Working capital in the financial model covers the spares cost.

5.13 Ancillary Requirements

One Front-End Loader (FEL) will be used to feed the plant. One track dozer will be used to construct mine roads, and level the dumps. Considering the distance between the dumps and the pit, it is considered unpractical to use only 1 track dozer, and therefore one will be stationed at the dumps and one at the pits.

One motor grader will be used to grade and maintain the mine roads. A 18,000 litre water bowser will be used for dust suppression, which is important considering the proximity of the town. Four light vehicles have been budgeted for, as well as lighting plants, radios and other light equipment.

5.14 Mine Labour

A relatively light compliment of labour suffices for this straight-forward operation. The table below shows the labour compliment.

Table 5-5: Mining Labour Requirements

K-Hill Mining Department Labour requirements			
	Per shift	Total	
Chief Surveyor	1	1	
Surveyor Assistant	1	2	
Production Super	1	1	
Foreman	1	3	
Geologist	1	1	
Articulated Truck Driver	4	16	
Excavator Operator	1	4	
Backhoe Operator	1	4	
Dozer Operator	2	8	
Grader Operator	1	2	*Not at night*
Bowser driver	1	2	*Not at night*
FEL Operator	1	4	
Total	**16**	**47**	

The surveying and geology requirements are relatively light and can be dealt with by setting out in advance and sharing responsibility between the chief surveyor and the assistants, as well as the Production Superintendent and the Geologist.

It has been assumed that cool conditions at night allow the bowser to be run only 12 hours of 24.

5.15 Reserve Statement

Considering that all the RoM material is classified as inferred, no reserve statement can be given.

However, the indicative mining schedule for this PEA Study is given in Table 5-7 at the end of this chapter.

5.16 Capital Costs

Almost the entire capital expenditure needs to take place in the first year. One articulated dump truck purchase can be postponed until Year 4. The capital schedule is shown in Table 5-6 below.

Table 5-6: Capital Schedule

Equipment	QTY	Year 1	Year 2	Year 3	Year 4
Excavator 2,1BCM - Kobelco SK380LC	1	260,880			
Backhoe TLB - BELL 315SL	1	90,887			
Articulated Dump Truck - Bell B30E	4	1,113,592			371,197
Track Dozer - Bell 850J	2	715,768			
FEL 2,3 BCM - Bell L1506E	1	181,977			
Water Bowser - 18kl	1	284,890			
Light Vehicles	4	200,000			
Motor Grader - Bell 770G	1	319,383			
Lighting plants, radios, misc	1	80,000			
	Total	**3,247,376**	**-**	**-**	**371,197**

5.17 Operating Costs

Mining operating costs have been estimated using various operations around the world with similar operating procedures and similar annual production size. The sources for these were the S&P global database for mining operations, as well as actual costs from other operations known to SRK. A total of 6 operations were used to calculate an average mining cost to be expected, utilising a weighting system that gave more weight to costs of operations in Africa.

The mining costs were escalated by bench height to simulate longer haulage distances using USD0.01 /m. This higher incremental cost is considered appropriate considering the small scale of the mining operation. The total mining operating cost is a small part of the total cash cost, as the graph in Figure 5-11 demonstrates.

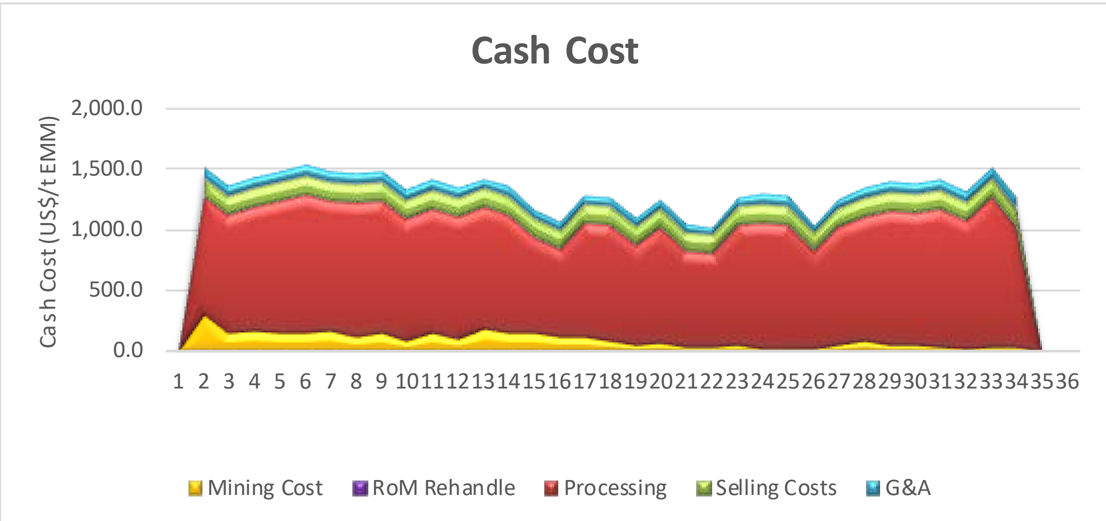


Figure 5-11: Cash Cost in USD /tonne processed, on a quarterly basis

5.18 Conclusions

Based on the work undertaken to date, the following key conclusions are made:

- The mining of the K-Hill deposit is a relatively straightforward affair, with no known or expected issues
- Mining equipment, maintenance and support services to the mining operation are easily obtained in the local region and are easily brought to site
- The site has good access to quality infrastructure
- No blasting is anticipated to be required, but a ram will need to be used to break the breccia cap
- A relatively small mining fleet will be able to deal with the required volumes that need to be moved
- A relatively light labour crew is required

5.19 Recommendations

SRK recommends the following actions to be taken on the mining operational aspect to bring this project into a pre-feasibility level of accuracy:

- A geotechnical study is required to investigate the validity of the assume pit wall slopes
- A load and haul study should be undertaken to prove assumed equipment requirements
- An updated pit optimisation should use the recoveries that have come out of recent test work
- Further analysis on the load and haul could shorten the haulage times and reduce cash costs

Table 5-7: Mining Schedule for K-Hill

Base Case	Units	Project Year Total	1	2	3	4	5	6	7	8	9
Physicals											
Rock	(t)	9,071,577	1,498,282	1,442,298	1,365,842	1,953,310	1,040,698	565,666	592,601	487,033	125,847
Waste	(t)	7,875,994	1,405,282	1,292,298	1,215,842	1,805,699	890,698	415,666	442,601	337,033	70,874
Stripping Ratio	(t waste : t RoM)	6.6	15.1	8.6	8.1	12.2	5.9	2.8	3.0	2.2	1.3
RoM	(t)	1,195,583	93,000	150,000	150,000	147,610	150,000	150,000	150,000	150,000	54,973
Rehandle	(t)	0	0	0	0	0	0	0	0	0	0
Expit RoM											
Total	(t)	1,195,583	93,000	150,000	150,000	147,610	150,000	150,000	150,000	150,000	54,973
	(% MnO)	9.22	29.2	26.3	28.2	33.5	31.6	32.8	30.6	26.5	25.3
	(t MnO)	354,360	27,114	39,418	42,246	49,522	47,386	49,138	45,969	39,682	13,885
Processing											
Mill Feed	(t)	1,195,583	93,000	150,000	150,000	147,610	150,000	150,000	150,000	150,000	54,973
Total	(% MnO)	29.64	29.15	26.28	28.16	33.55	31.59	32.76	30.65	26.45	25.26
	(t MnO)	354,360	27,114	39,418	42,246	49,522	47,386	49,138	45,969	39,682	13,885
	(t MnO)	333,099	25,487	37,053	39,711	46,551	44,543	46,190	43,211	37,301	13,052

6 METALLURGY AND MINERAL PROCESSING

6.1 Introduction

Dr Ian Flint of Lab 4 Inc. (Canada) was engaged by Giyani to provide metallurgical consulting services. Testing was carried out by Lab 4, the Department of Geology of Dalhousie University and the Minerals Engineering Centre of Dalhousie University, all of Halifax, Nova Scotia, Canada. Details are as follows:

- Stage 1 was performed by Lab 4 and the Department of Geology of Dalhousie University. This included optical and electron probe work that identified the valuable and waste minerals, the particle sizes, and the approximate grind sizes required for liberation to exposure
- Stage 2 was performed by Lab 4 and the Minerals Engineering Centre of Dalhousie University. This included tests on the leaching of various samples to determine the chemistry, and residence times required to dissolve the manganese as the first stage of a hydrometallurgical process. The grind size was based on the results of Stage 1

This work was further extended to cover solution purification (by Solvent Extraction or SX) and meta recovery (by electrowinning or EW).

6.2 Mineralogy

A mineralogical analysis was performed on a selection of manganese oxide bearing rocks taken as grab samples from K-Hill. Four samples were tested to determine the mineralogical composition of the manganese minerals. These samples and the resulting determinations are found in Table 6-1. Hematite, other iron oxides and some pyrite along with silica, and in one horizon of the shales, kaolin, are found throughout.

Table 6-1: Identified and possible manganese minerals in the K-Hill samples

Sample	Description	Mineral	Formula
KH17MT01	Dump Material	Cryptomelane Hausmannite Hollandite Psilomelane Pyrolusite	$K(Mn_7^{4+}Mn^{3+})O_{16}$ $Mn^{2+}Mn_2^{3+}O_4$ $Ba(Mn^{4+}{}_6Mn^{3+}{}_2)O_{16}$ $BaMn^{2+}Mn_8^{4+}O_{16}(OH)_4$ MnO_2
KH17MT02	Altered Shale	Jacobsite	$Mn^{2+}Fe^{3+}{}_2O_4$
KH17MT03 KH17MT05	Shale with Kaolin	Coronadite Cryptomelane Hausmannite Hollandite Psilomelane	$Pb(Mn^{4+}{}_6Mn^{3+}{}_2)O_{16}$ $K(Mn_7^{4+}Mn^{3+})O_{16}$ $Mn^{2+}Mn_2^{3+}O_4$ $Ba(Mn^{4+}{}_6Mn^{3+}{}_2)O_{16}$ $BaMn^{2+}Mn_8^{4+}O_{16}(OH)_4$
KH17MT04	Silicified Shale	Hausmannite Psilomelane group	$Mn^{2+}Mn_2^{3+}O_4$ $BaMn^{2+}Mn_8^{4+}O_{16}(OH)_4$

In terms of presentation, the manganese minerals occur in three forms:

- As staining on the silicates, iron oxides and themselves
- As small veins where manganese oxides have been deposited on each other particularly within the well fissured portions

- As nodules where the manganese has built-up into botryoidal masses that may contain other minerals within them. This study did not investigate the nodules as this would involve a significant sample to determine their mass percent and size distribution at various locations

This supergene mineralogy is complex as there are many oxidation states of manganese and a variety of other metals are present.

The results of the assays from these samples, converted to the standard oxide form, are shown in Table 6-2. These assays were taken on gram sized samples and not on individual crystals thus represent a number of different minerals as found within the entire rock sampled.

Table 6-2: Mineralogy Samples Head Assays

Sample	Description	Weight Percent	
		FeO	MnO
KH17MT01	Dump material	39.7	48.7
KH17MT02	Altered shale	27.3	48.8
KH17MT04	Silicified shale	4.5	4.3
KH17MT03, KH17MT05	Shale with kaolin	23.2	60.8

6.3 Leaching

Core samples from drill hole DDKH18_0018 were subjected to leaching tests. A description of these samples is provided in Table 6-3. Given the that the samples were sourced from a single location they are not completely representative of the various types and styles of mineralization and the mineral deposit as a whole.

The required grind size for leaching was estimated from the thin sections to be approximately 200 µm based on breakage occurring along existing fractures, manganese deposited preferentially on surfaces and within the fractions, and liberation to leaching liquid exposure. Leaching tests was conducted on a P_{80} of this target size. No grind size optimisation testwork was conducted.

Tests were conducted with and without the addition of sucrose as a reductant. Having ground the sample to the target P_{80}, ten sub-samples were drawn of 25 g each. One sub-sample was set aside and assayed later as the head grade. Four sub-samples where used for reductant tests and five for non-reductant tests. All tests were performed using 25 g of rock sample, 125 ml of 3.64M H_2SO_4 solution (approx. 260 g/l) at 90 °C. Four independent leaches were performed with the reductant with only residence time being varied: 30, 60, 120 and 180 minutes. Five independent leaches were performed without the reductant with only residence time being varied: 15, 30, 60 120 and 180 minutes. All ten sub-samples were assayed for Mn and Fe.

Table 6-3: Description of samples (DDKH18_0018) used in leach tests

Sample	Description	Photograph
KH18MT010	Mn-Shale High density Metallic Luster Interval 23.73-27 m Drilled thickness 3.27 m Recovery 1 m Sample type – core/chips/rubble Specific Gravity 3.05	
KH18MT011	Fe Shale Massive Mn Oxide mineralization within bands Interval 27-30.85 m Drilled thickness 3.85 m Recovery 3.7 m Sample type – core Specific Gravity 2.54	
KH18MT012	Mn-conglomerate Massive Mn Oxide mineralization Granular texture Finer grained conglomerate unit Interval 48-50.73 m Drilled thickness 2.73 m Recovery – 0.56 m Sample type – core Specific gravity 2.62	

The results of the leach tests are summarised in Table 6-4.

Table 6-4: Leach test summary

Sample	Reductant	Head Assay (%)		Leach Extraction (%)									
				15 min		30 min		60 min		120 min		180 min	
		Mn	Fe	Mn	Fe	Mn	Fe	Mn	Fe	Mn	Fe	Mn	Fe
MT10	Without	24.7	13.8	2.4	6.1	2.6	7.3	2.9	13.7	3.1	16.6	3.2	24.4
	With			-	-	77.5	18.6	91.7	29.1	91.4	29.4	87.7	22.4
MT11	Without	30.7	12.1	1.7	2.0	1.7	2.3	2.3	18.9	2.1	13.9	0.4	6.6
	With			-	-	12.2	34.4	83.1	58.3	93.9	78.8	90.8	74.2
MT12	Without	6.85	1.99	1.2	6.0	1.2	14.0	1.8	32.0	2.3	38.0	1.8	40.0
	With			-	-	90.3	63.0	100.5	87.0	94.6	83.3	98.9	87.0

The results show negligible Mn leaching in the absence of the sucrose reductant. Figure 6-1 shows the Mn leaching kinetics for the tests with reductant.

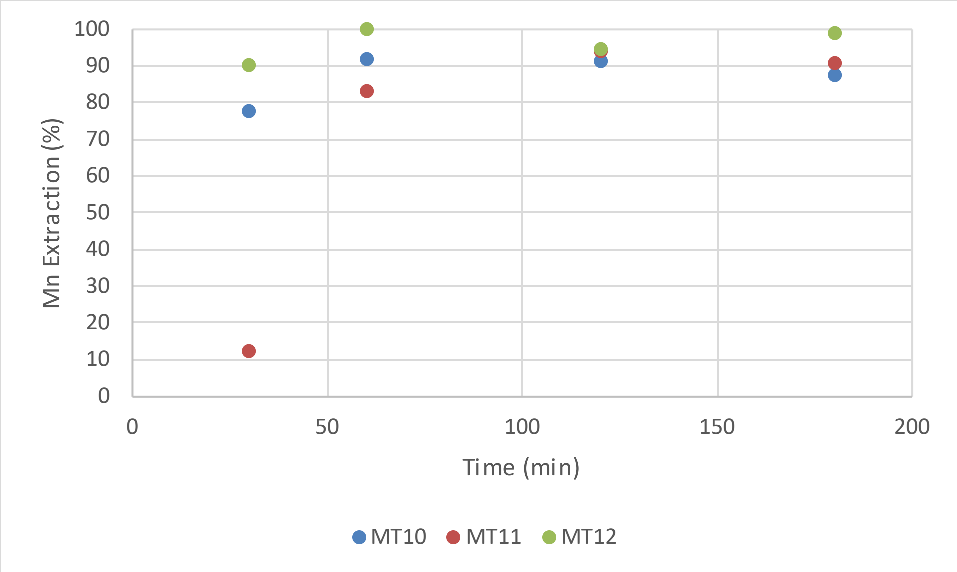


Figure 6-1: Mn Leaching Kinetics (with Reductant)

The results show that most of the recovery occurs within the first hour, with limited additional recovery occurring after hour two.

Figure 6-2 shows the Fe leaching kinetics both with and without reductant. These results show a considerable variability in the Fe leach response. Without the addition of the reductant, Fe extractions were still up to 40% (MT12). With the reductant addition, Fe extractions were up to 80% (MT12), however the Fe extraction for MT10 was still relatively low (30%).

Based on these results, a Mn leach extraction of 94% has been assumed for the subsequent analysis.

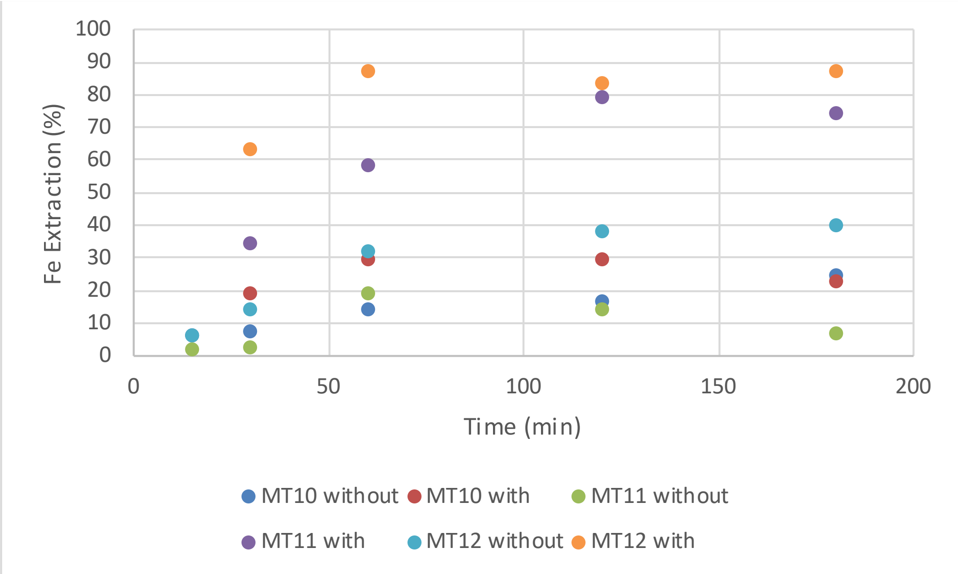


Figure 6-2: Fe Leaching Kinetics

6.4 Solvent Extraction

The concept of the SX circuit within the overall extraction circuit is shown in Figure 6-3.

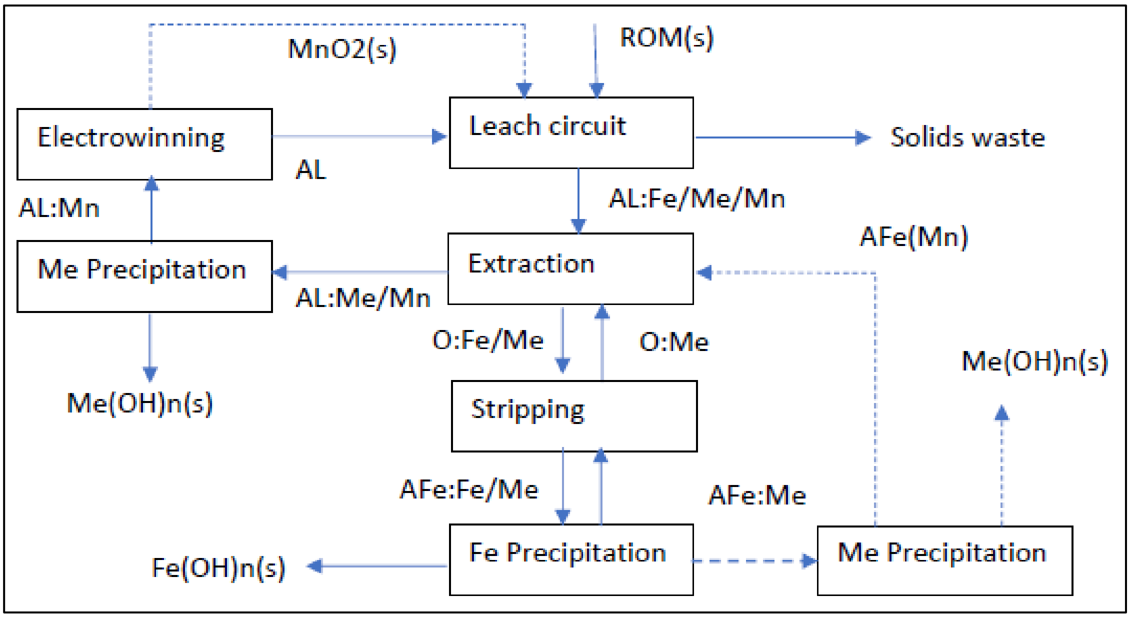


Figure 6-3: Extraction Circuit Schematic

The organic phase is used to extract Fe from the leach solution, with the Fe-free solution progressing to Electrowinning. The Fe (and other metals) are then stripped from the loaded organic, and subjected to two stages of precipitation, firstly to remove Fe for disposal, then to remove the other metals, either for disposal or for recycle, depending on the amount of Mn reporting to that stream.

Depending on the content of other metals in the Mn-rich solution leaving SX, a metal precipitation stage may be required on this stream ahead of the EW stage.

The use of an organic phase to separate Fe and Mn is based on the relative extraction response of these metals in the chosen extractant D2EHPA (di-(2-ethylhexyl)phosphoric acid),as shown in Figure 6-4, which shows a pH window to separate between Fe and Mn in the range 2.5-3.5, with a pH of less than 1 required to strip the Fe from the loaded organic..

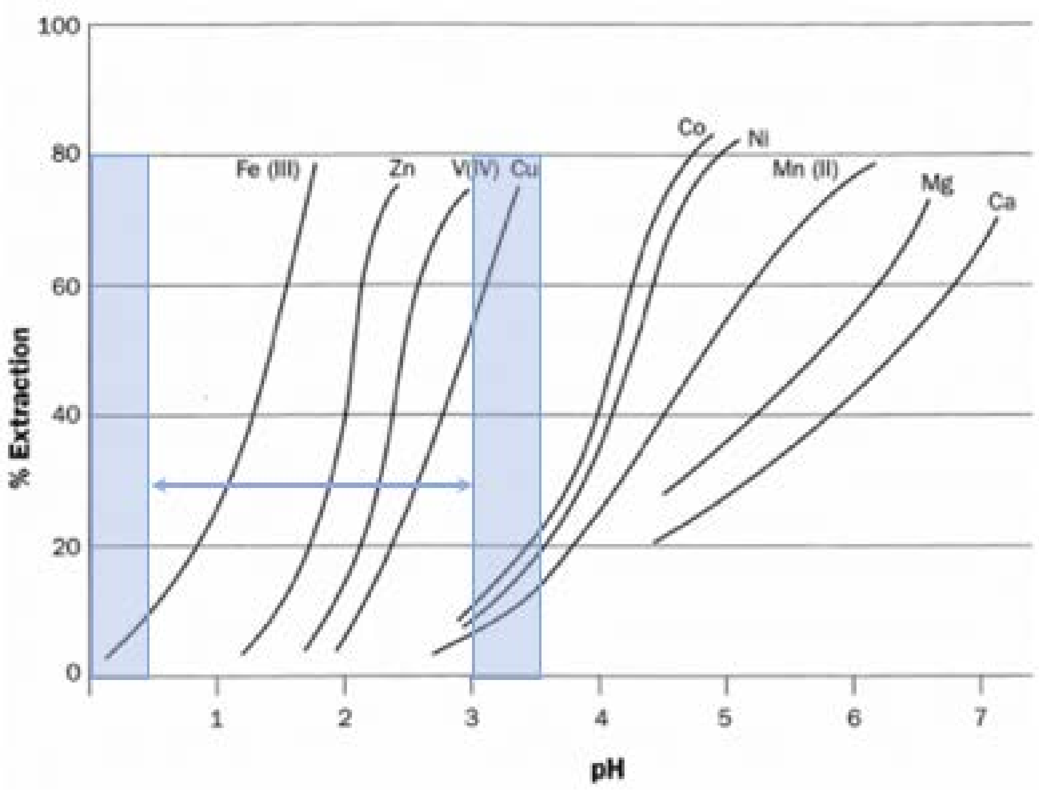


Figure 6-4: D2EHPA extraction response of metals with pH

Extraction tests were conducted using the same samples as used in the leach tests. The process was refined starting with MT12, then MT10 and finishing with MT11. The extraction test results are summarised in Table 6-5.

Table 6-5: Extraction test summary

Sample	Mn ppm	Mn Recovery	Fe ppm	Fe Recovery	pH
MT10	16908	95.8%	30.3	1.2%	3.0
MT11	12222	96.7%	0.2	<1.0%	3.65
MT12	7047	92.2%	18	1.2%	3.6

The MT11 results suggest that a stage recovery of in excess of 96% should be possible.

Metal precipitation uses the differing properties of the metals of interest with pH, as shown in Figure 6-5.

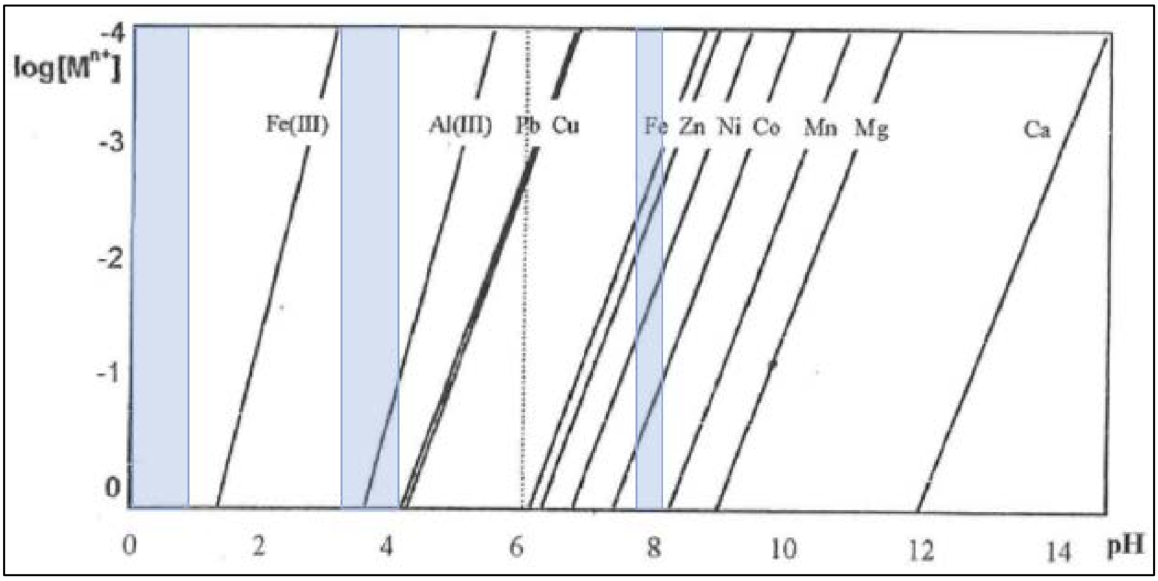


Figure 6-5: Precipitation response of metals with pH

The loaded solution from stripping will be at pH 1. Raising the pH to above 3.5 will remove Fe preferentially, following which a bleed solution would be removed as required to control the other metals, after which the pH of the remaining solution would be lowered to 1 for recycle to the stripping circuit.

Raising the pH of the bleed solution to ~8 will be used to precipitate most of the remaining metals, after which the solution can be recycled to the extraction circuit to recover the remaining Mn.

Fe precipitation test work used a pH of 5, which precipitated over 99% of the Fe and less than 1% of the Mn.

6.5 Recovery Methods

6.5.1 Process Description

A simplified block diagram for the proposed process for Mn metal production is shown in Figure 6-6.

6.5.2 Comminution

The comminution circuit will consist of several stages of crushing and grinding to achieve the target grind size, which is a P_{80} of 200 μm subject to further optimisation.

In order to preserve the water balance, a solid / liquid separation is likely to be required between the comminution and leaching stages. Given the relatively coarse target grind size, a vacuum belt filter is envisaged without prior thickening. Filtrate would be recycled internally within the grinding circuit.

6.5.3 Leaching

Leaching will be undertaken in a series of open topped tanks. The testwork indicates a total leach residence time of two hours. Filtered solids from the comminution circuit will be mixed with barren electrolyte returned from the EW stage, with reagent sulphuric acid added to meet the target acid strength (260 g/l based on the test work), as well as the reductant sucrose, which is consumed during the leach reaction.

An on-site boiler will be used to generate the steam required to maintain the target leach reaction temperature of 90 °C.

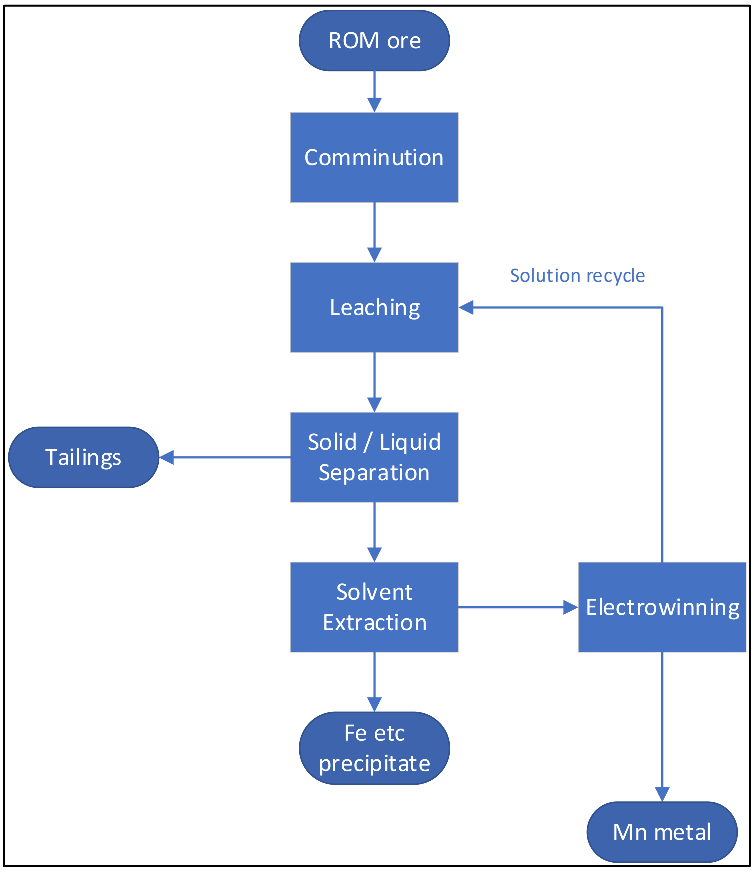


Figure 6-6: Process Block Diagram

6.5.4 Liquid/Solid Separation

As with the dewatering stage ahead of leaching, a vacuum belt filter is envisaged for the solid / liquid separation between leaching and SX. This stage will incorporate a cake washing stage, in order to both maximum soluble Mn recovery, and to minimise the residual acid content of the filter cake.

The resulting washed solids will be suitable for conveying and dry stacking.

6.5.5 Solvent Extraction

The filtrate from the leaching stage will be subjected to SX for impurity removal. The SX circuit will consist of one or more extraction stages, one or more stripping stages, plus washing / scrubbing stage/s as appropriate.

Purified electrolyte from the SX circuit will be advanced to the EW circuit. Raffinate will be recycled internally within the SX circuit. A bleed stream of raffinate will be removed for water balance and impurity (principally Ca and Mg) removal purposes. As part of the bleed stream treatment, a manganese-containing precipitate will be produced, which will be recycled to the leach or extraction circuits to minimise Mn losses.

6.5.6 Electrowinning

Manganese metal will be recovered from the purified solution from SX by electrowinning. Due to the particular electrochemical behaviour of manganese, the EW cells will be configured with a membrane to separate the anodic and cathodic reaction zones.

6.5.7 Electrorefining

In order to produce high purity (>99.7%) Electrolytic Manganese Metal (EMM), a second stage of electrorefining is required. The grades of the first stage EMM is typically suitable for electrorefining in halide-based solutions. The final product EMM will be produced in typical flake form. The scoping report on the Electro Refining (ER) circuit is attached as Appendix A to this report. Based on the test work conducted to date, a current efficiency of 44% has been assumed for the EW stage, a figure at the lower end of typical values for manganese EW. The ER stage typically has current efficiencies in the order of 80% or about half as much electricity consumption as the first stage. SRK understands that the EW and ER processes will be run sequentially so that the maximum power demand will be no more than 45 MW.

6.6 Process Design Criteria

The design plant production rate is 150 ktpa of RoM. The head grade, based on the annualised mine plan, is 29.6% Mn, which combined with the assumed Mn recovery through the process of 87.5% (combining leaching and SX recoveries), is expected to give an average annual production of 39.5 ktpa of >99.7% Mn EMM, with variances of as low as 35 kt for Year 2 and as high as 44.2 kt for Year 4.

6.7 Process Capital Estimate

6.7.1 Process Plant

A procurement firm based in the UK, Prolog Supply & Services Ltd (hereafter referred to as PS&S), provided a capital cost estimate for the main plant based on quotes received from a well-known mining EPC contractor, BGRIMM, which is given in Table 6-6. The quotes were based on access to a draft version of this report, which means it is based on the K-Hill RoM composition, flow diagrams and processing methodology as provided by Ian Flint. A capital cost estimate for the electro refining circuit was prepared by Ian Flint (Appendix A) which is necessary to produce a product of saleable quality. The figures in Table 6-6 include contingency. It is in the 'Others' capital and amounts to USD9.59 M.

Table 6-6: Capital Estimate for Processing Plant

No.	Items	Cost (thousand USD)
1	Civil and structure	21,000
2	Equipment	34,000
3	Installation	7,000
4	Others	17,910
	Electrorefining circuit	16,000
5	Total	95,910

The capital estimate is for a plant that will produce 99.9% EMM when combined with the electro refining process. SRK has therefore recommended that the contingency for the plant capital estimate should be set at 15%. PS&S noted in their report that the quotes should be viewed with caution, which indicates support for SRK's recommendation.

These estimates should be considered to be commensurate with a Class 5 estimate as defined by the AACE International, and so have an accuracy of no better than -20/+30%. SRK considers that the estimate of USD95.9 M, with an added 15% contingency, is a reasonable estimate for a PEA but requires more detailed laboratory test work at a pilot scale and detailed engineering to reach an acceptable level of accuracy for a Feasibility Study.

6.8 Process Operating Cost Estimate

6.8.1 Process Plant

As with the capital cost estimate, the operating cost estimate was based on published data from similar projects, although in this case a unit cost for power was known, and so that value (USD 0.06 /kWh) was incorporated into the estimate.

The comparison was based on a typical operating cost breakdown for such a facility, namely Labour, Reagents and Consumables, Maintenance and Power. The total estimated operating cost was USD276.45 /t RoM, broken down as follows:

- Labour: USD35 /t;
- Reagents and Consumables: USD55 /t;
- Maintenance: USD35 /t;
- Power: USD125 /t and a further USD8.45/t for repayment over the LoM to Botswana Power Corp for their initial capital outlay; and
- Electrorefining: USD18 /t (includes the power cost for electrorefining)

As with the capital cost estimate for the process plant, this estimate should be considered to be commensurate with an AACE International Class 5 estimate.

6.9 Conclusions

Based on the scoping level work completed for this assignment, SRK concludes the following:

- There is viable process route for the proposed EW product
- Further detailed test work is required to support the product specification, recoveries, operating costs, plant flow sheets and capital cost estimation

6.10 Recommendations

Based on the scoping level work completed for this assignment, SRK recommends the following:

- Further detailed testwork to be completed to support the product specification, recoveries, operating costs, plant flowsheet and capital costs
- A key focus needs to be the exact product specification that can be produced
- Pilot plant tests to be completed at a much larger scale

7 TAILINGS WASTE DISPOSAL

The tailings storage facility (TSF) for Kgwakgwe Hill will be constructed south of proposed plant location as shown in Figure 7-1 on page 53. The TSF is expected to be built using the downstream technique utilizing the north hill and south dam as containments.

The tailings slurry will be pumped downhill from elevation 1332 m to elevation 1315 m.

The general characteristics of tailings disposal are listed below.

Table 7-1: Tailings Details

Characteristics	Value	Comments
Total tonage of tailings	1.2 Mt	Assuming 75 % of mined RoM will be deposited in form of tailings
Number of years of production	8 years	
Yearly production	0.15 Mtpa	
S.G.	2.3	assumed
Unit density of placed tailings	1 to 1.1 t/m3	assumed
Required volume of tailings to store	1.2 Mm3	

Other assumptions used in the PEA study:

- The TSF is not lined because the tailings are assumed to be non acid and/or metal leaching
- Foundations are suitable for construction of TSF (no geotechnical site investigation was performed)
- All clean surface water will be diverted around TSF and only precipitation falling directly on the TSF is to be stored and pumped back to the Plant
- Water from TSF is suitable for recycling in the plant (no need for water treatment)

7.1 Construction of Containment

The containment dam was assumed with a crest width of 8 m and 2H:1V upstream and downstream slopes. For the final crest at 1315 m, the capacity of TSF was calculated to be 1,200,000 m^3 assuming 3 m of the freeboard. An additional volumetric capacity could be achieved in the future by raising the facility. For example, by raising the TSF additional 1 m, an extra capacity of 250,000 m^3 can be achieved.

The total volume of the containment dam to the final elevation 1315 m is 394,456 m3. The distance from TSF to the plant is 600 m as shown in Figure 7-1.

The TSF will be built in two stages:

- Starter facility
- Final facility

The starter TSF will be built to elevation 1310 m and it will involve a construction volume of 131,000 m^3 of fill material. This will allow for storage of 358,000 m^3 of tailings (roughly 2 yrs).

The initial and final TSF containment dam will be built using the mine waste materials whenever possible. Assuming that the water reclaim pool will be away from the dam containment, the mine waste material which will consists of clay shale could be used for construction. The dam will be homogeneous, without core and core filters because of low height and water pool situated away from the containment dam. The fill material must be conditioned for the suitable moisture content and compacted in 0.3 m lifts using a sheepsfoot roller or other similar compaction device.

7.2 Transport of Tailings and water return

The tailings slurry will be transported using the PVC pipeline (6-inch diameter). Because of downhill gradient (30 m) between the plant and TSF, the tailings slurry will be transported by gravity.

The return water will be pumped back to the plant from the TSF using a floating barge through a 4-inch diameter water line.

7.3 Capital Cost Estimate

The following in Table 7-2 are the main capital costs for the TSF which at this stage amount to USD2.738 M.

Table 7-2: Capital Cost Estimate for TSF

Item	Unit Cost (USD)	Quantities	Subtotal (USD)	Remarks
Site Clearing and grubbing	$1/m^2$	30,000	300,000	
Excavation of top soil TSF	$3/m^3$	150,000	450,000	Storage of top soil
Dam construction (starter dam)	$3/m^3$	131,0000	393,000	If Mine waste could be used
Dam construction (final dam)	$3/m^3$	265,000	795,000	If Mine waste could be used
Tailings slurry linen including utilities corridor	$100/m	1000	100,000	
Floating barge	$200k	1 barge	200,000	
Water return line	$100/m	1000	100,000	
Power line	$100k/km	1 km	100,000	
Clean water ditches	$100/m	3,000	300,000	
Total			**2,738,000**	

7.4 Operating Cost Estimate

The operating costs will include everyday operation of Tailings Storage Facility such as discharge point changes, line changes, power for water barge, maintenance, and other costs.

It is assumed that these costs will be at USD0.5 /t of tailings, totalling USD0.6 M.

7.5 Conclusions

The PEA level design for the TSF (not including the plant tailings costs) are assessed at USD2.738 M in capital costs, and USD0.6 /t for operating costs.

7.6 Recommendations

The design of the TSF assumes the downstream construction of the containment with the mine waste material and no lining of the TSF to prevent any fugitive seepage. The need for lining will be assessed in the next stage of the Project.

SRK notes that if these conditions change during the subsequent technical studies then the cost of the tailings storage could increase materially.

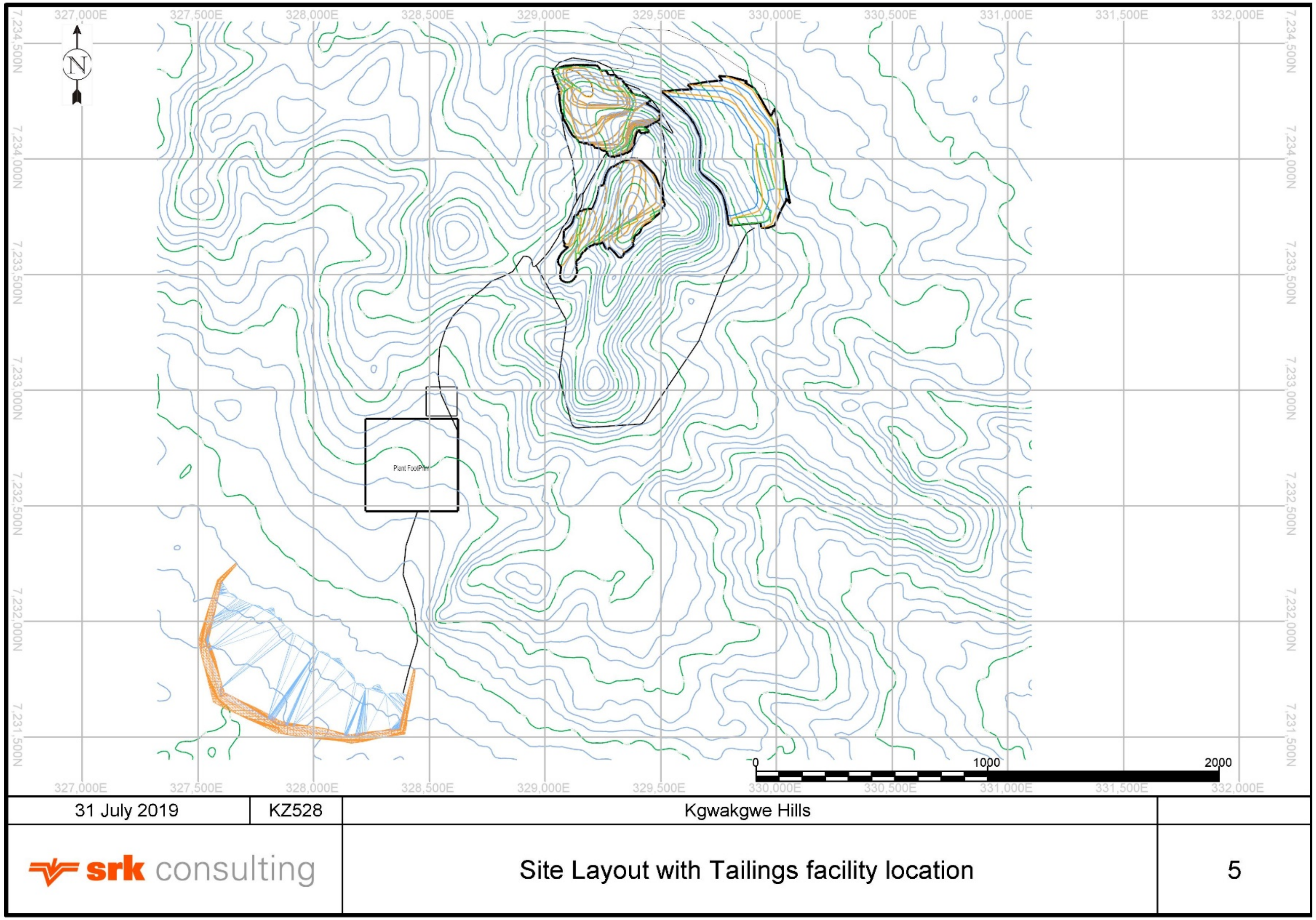


Figure 7-1: TSF location south of proposed plant

8 INFRASTRUCTURE

SRK was requested to provide input on mine support infrastructure and power supply on a desk top basis. The infrastructure required to support the mining and processing operations will be as follows:

- General project infrastructure (i.e. buildings, utilities, including mining support as the operation will be owner mining)
- Plant support infrastructure at the processing plant (full description and costs considered in the processing section)
- No accommodation block is needed. Accommodation can be found for the workers in the town of Kanye and visiting staff such as consultants or off-site management can be housed in the town's B&B and hotel facilities.
- Power supply
- Water supply

8.1 Project Infrastructure

8.1.1 Office and Change House Buildings

An administration building will be located near the mine site entrance gate. The building will have a reception area, offices, meeting rooms, medical clinic, kitchenette and washrooms. The offices are for the mine manager, engineers, geologists, and clerks. The building will be of pre-fabricated construction.

A welfare and ablutions building will include limited male and female showers, bathrooms, and a change room with lockers as it is envisaged that all workers will prepare for work at the accommodation block. A muster area is located at the reverse with personal protective equipment stores.

The sampling and assaying laboratory will be located at the processing plant and will also be utilised by the mining and geology teams for grade control and exploration.

Central communications consisting of access and control, intrusion and detection alarms will be located at the administration offices. General mine and plant communications will be delivered by Ultra High Frequency ("UHF") mobile radio, local and repeater channels. An existing cell phone tower needs to be relocated prior to construction and an allowance has been made in the cost estimate.

A Very Small Aperture Terminal ("VSAT") antenna will provide broadband data access for the project. The broadband data access will also be utilised to provide telephone communication, utilising over Internet Protocol ("IP") telephony.

8.1.2 Mobile Equipment Workshop, Tyre Change and Washing

The mining and ancillary vehicles workshop will service the mining mobile equipment fleet and ancillary equipment. The activities will include: maintenance, tyre change, and vehicle wash-down. After vehicle washing the water will be mechanically treated and recycled. The infrastructure maintenance facility will also include containers for spares storage. The workshop building will be constructed of containers with fabricated roof, with a tyre yard, change area and wash area located beside the workshop. Wash-bays for mining and ancillary vehicles will consist of concrete aprons with pollution control point and raised sides.

8.1.3 Fuel and Lubricant Storage, Dispensing

The primary method of refuelling for mining and ancillary vehicles is anticipated to be by "mobile bowser". The fuel storage and dispensing unit will have a self-bunded fuel tank with appropriate dispensing equipment. Lubricants will be stored in a secure warehouse adjacent to the vehicle maintenance workshop. Heating fuel oil will also be stored in the fuel storage area.

8.1.4 Warehouses & Facilities Maintenance

Lockable TEU / FEU containers will provide storage for general items. An integrated workshop warehouse area with concrete loading and unloading apron will house larger spares. The workshop building will be constructed of containers with fabricated roof.

8.1.5 Ancillary Vehicles

The Project will require ancillary vehicles such a 4x4 vehicles for operatives, crew buses, equipment for maintenance team (truck, excavator, roller, FEL etc.). An allowance for these for is incorporated into the mining fleet and owner's costs.

8.1.6 Water Treatment and Water Supply Facilities

The required water supply facilities are described in Section 4. The raw water will be pumped to central storage tanks. A proportion of the raw water will be sent to a vendor packaged treatment plant.

8.1.7 Waste Disposal

A septic system will be utilized for sewage disposal. Septic tanks ("ST") will be located at the main infrastructure facility, accommodation camps and near the open pit for mining operations. The septic tank sludge will be removed by vacuum truck at regular intervals. Exact location of septic tanks and sludge dumping and disposal options will be defined at the detailed design stage.

A management area for waste streams from the workshops is positioned next to the workshop. Materials (such as tyres, used oil, batteries) shall be stored, sorted and periodically emptied to go to a local landfill and approved safe disposal areas off-site in accordance with local regulation and / or international standards.

Used oils will be temporarily stored in a temporary storage area within the industrial site for subsequent handing over to the local certified companies, dealing with the treatment of these industrial waste categories in accordance with the regulations of Botswana. In case there are no local certified companies, it is recommended to consider options of installation of an incinerator at site for burning of wastes.

Processing waste streams are discussed in separate chapters.

8.1.8 Earthworks

The main infrastructure facilities will be located on a levelled earthwork pads and located adjacent to access roads and within easy access to the processing plant and accommodation but screened from the main highway. An allowance of USD0.5 M is included for this item. The earthworks and civils for the processing plant and any earthworks required for pre-stripping, waste dump and tailings development are assumed to be incorporated into these cost items respectively.

8.1.9 Internal Roads

Based on the preliminary site layout, around 5 km of site roads are required excluding plant internal roads. The following site roads are envisaged:

- Main access and plant to mining area access for 1,500 m
- Other site roads such as plant perimeter road and access tailings facility, camp and support infrastructure for 3,500 m
- Access and site roads comprise assumed as unbound pavement, minimal drainage and no requirement for bridges or significant culverts

At this stage it is assumed that there is no requirement to modify or upgrade any surrounding national roads.

8.1.10 Fire Response / Emergency Industrial Medical

Given the location and complexity of the processing, a dedicated fire depot will provide fire-response services to the Project and consists of a building for fire engine vehicles with training facilities. The main clinic will be equipment with medical equipment to manage industrial accidents and emergencies.

8.2 Plant Infrastructure

Miscellaneous buildings are also required for the processing plant such as:

- Gatehouse and security
- Plant change-house and ablutions
- Laboratory
- Plant workshop and reagent store
- Plant administration and training building
- Plant weighbridge

This costs for main plant facilities; plant control room, are assumed to be covered within the overall processing plant capital cost with operating costs captured in either plant operating costs or general and administration.

8.3 Accommodation

There is a well-developed local hotel sector in the adjacent town. The Company expects this to be sufficient for construction and operations. Any additional accommodation required for construction workers would be provided by the Contractors.

8.4 Power Supply

8.4.1 Strategy

The installed capacity is estimated to be 45 MW for production of EMM. SRK understands that the EW and ER processes will be run sequentially so that the maximum power demand will be no more than 45 MW. The Company’s preference is to connect to the national grid operated by the Botswana Power Corporation ("BPC"). The nearest point of access is a 132-kV substation around 15 km from the Project site.

8.4.2 Botswana National Grid

The national grid is concentrated in the southeast of the Botswana with transmission voltages of 132kV, 220 kV and 400 kV. A literature review confirms that energy demand has grown in Botswana and demand is met by imports from Electricity Supply Commission (Eskom) in South Africa, Namibia Power Corporation (Nam-power), Zambia Electricity Supply Corporation (ZESCO) and Southern African Power Pool (SAPP). The country is liable to experience load shedding (power black-outs) as a result of power supply shocks in exporting countries although is currently rehabilitating current generation capacity and extending this capacity. SRK understands that the BPC has informed the Company that the national grid can supply the requested power although point of access hasn't been confirmed.

8.4.3 Required Power Supply Infrastructure

On the basis that the nearest point of access is a 132-kV substation around 15 km from the project site and demand is 45 MW, which the existing substation and connecting line can provide, the following infrastructure will be required:

- Connection works at the existing substation
- 132 kV double circuit overhead line for 15 km
- Project Substation 110 kV step down to the plant distribution voltage
- Site electrical distribution (assumed to primarily be within the fence-line of the plant / refinery area) with some eternal distribution for support infrastructure
- Back-up power generators for camp, security infrastructure and critical loads in the plant. 1.5 MW has been assumed

8.4.4 Power Cost

SRK understands that the BPC has informed the Company that the cost of power via the national grid, and which has been used in the operating cost is USD0.06 /kWh (BWP0.634).

8.5 Estimated Capital Costs

An indicative cost estimate has been completed using benchmark, historical or in-house database rates and costs for the infrastructure required to support the mining and processing operations. The cost estimates are based on the PEA level definitions and resulting costs are to an accuracy of +50/-30%. The cost estimates are considered to include direct element only. Indirect costs and contingency have been added at project level.

The power supply capital cost is provided for by the Botswana Power Corporation. The capital is estimated at USD 7.1 million. This capital will need to be repaid over the life of mine. SRK has calculated that the kWh price, normally BWP 0.634 as mentioned above, will rise to BWP 0.669.

Table 8-1: Estimated capital cost total –infrastructure (*1)

Area	Total (USDm)
Project Infrastructure Facilities	1.9
Accommodation (not required)	-
Utilities	0.7
Bulk Earthworks	0.5
Site Roads	1.2
On-Site Primary Distribution (main subst to facilities) and 1.5 MW Emergency Power (*2)	2.0
Total	6.3

(*1) Excluded from the capital estimate in this section is: water supply infrastructure, plant support infrastructure, processing plant including civils and earthworks, contingency and Owner's costs.

(*2) The Company has confirmed that the capital costs for bulk power supply connection works will be met by the BPC and subsequently recouped within the supply tariff in the early years of operation. These connection costs are estimated to include the connection works at the assumed existing substation, a 32kV double circuit overhead line for 15km, and the main Project Substation 132kV step down to the site and plant distribution voltage. These costs are expected to be in the order of USD 7.1M.

8.6 Estimated Operating Costs

Operating costs for maintenance of building and civil construction are covered under the G&A costs for the Project. Maintenance of bulk power and water supply infrastructure are assumed to be incorporated into the supply cost.

8.7 Recommendations

In addition to typical study work and cost estimation normally recommended for a PFS, SRK has the following recommendations:

- Undertake a power supply study to confirm the best solution for power supply to the Project. This would take the form of a technical and economic trade-off. It will require liaison with the BPC to determine the best grid supply solution (including outage records from the proposed substation and effect on plant operating hours) and investigate a standalone power-plant / renewables strategy as an alternative
- Develop and maintain a Risk and Opportunities Register to allow the Project Team and third parties the opportunity to identify, review and eliminate risks as the Project develops further. The Risk and Opportunities Register is to be incorporated into a risk model to evaluate an appropriate level of contingency to be applied to the cost estimate(s). The main aim being to reflect the anticipated total cost of the Project, and to monitor the Projects status and the probability of completion at each phase
- Develop an implementation schedule with associated timing and dates for all project related infrastructure and investment. The estimated capital expenditure should then be updated within the financial model to reflect the implementation schedule

8.8 Product Logistics

Product will be loaded to barrels or bulk bags and placed in twenty-foot equivalent unit ("TEU") and transported by road to a port in Mozambique or South Africa; the transport distance will be in the order of 900-1000 km along national roads and highways. The product has a density of 7 t/m^3 and TEUs will carry a 27 t payload.

For this PEA study, it has been assumed that the off-take agreement with the Client to which the EMM will be supplied, will take care of insurance and transport costs. SRK notes that should such an agreement not be reached, then studies by the Client and SRK have shown that transport costs should be between USD65 and USD85 per tonne of EMM shipped. Port costs come on top of this, and will be in the order of USD7 /t.

9 ENVIROMENTAL AND SOCIAL

9.1 Introduction

SRK was not mandated to review the Environmental and Social (E and S) issues related to the Project as part of this PEA but notes that the Client has been working closely with the Government of Botswana in recent years and has taken a responsible approach to the Project. A key part of this has been the development of a local Environmental Management Plan (EMP) for the exploration work carried out to date at the Project. The EMP was formally approved by the Department of Environmental Affairs in Botswana in 2015 and was prepared by a firm of local consultants.

The Client intends to continue this approach in future studies and will be carrying out further detailed E and S work as the Project progresses.

SRK notes that the Project is located in an area where there are some receptors and a careful approach will be required to managing E and S issues. However, SRK considers that with the appropriate management measures during the planning stages there is no reason why a 'Social Licence to Operate' cannot be obtained.

9.2 Closure

For the PEA, SRK has assumed some closure costs of USD5 M for the Project and notes that the impacts on the environment should be able to be mitigated through a range of measures including water treatment and progressive closure during operations.

10 FINANCIAL EVALUATION

10.1 Introduction

This section represents a financial evaluation of an inferred resource, which are too speculative geologically to be categorised as mineral reserves. The reader is warned that mineral resources that are not mineral reserves do not have demonstrated economic viability. The following general assumptions have been applied to the Technical Economic Model ("TEM") for the Project:

- is expressed in real terms;
- is presented at 2019 money terms for Net Present Value (NPV) calculation purposes;
- applies a Base Case discount rate of 10%;
- is based on long term manganese prices of USD4700 /t EMM for a 99.9% product provided by the Client;
- is expressed in post-tax and pre-financing terms and assumes 100% equity;
- note the Client's tax advisers have indicated that they consider it likely that, instead of using the mining corporate tax formula, the flat Botswanan corporate tax rate of 22% can be applied due to the fact that the Company will be refining the extracted manganese ore into electrolytic manganese metal through a manufacturing process, accordingly a base corporate tax rate of 22 % has been used
- Government royalties have been applied at 3.0% of revenue; and
- for tax purposes capital investments are depreciated immediately and unredeemed capital is carried forward indefinitely as allowed for mining projects in Botswana.

10.2 Operating Costs

Table 9-1 summarises the expected operating costs based on the PEA level work completed for this study.

Table 9-1: Summary of Unit Operating Costs

Operating Costs	LoM (USD/t milled)
Mining	26.2
Rehandle	0.0
Processing	276.5
G&A	20.0
Selling Costs	37.3
Contingency	0.0
Total Operating Costs	**360.0**

10.3 Capital Costs

Total capital costs are estimated to amount to USD141.0 M over the Life of Project. Mining capital costs are estimated at USD3.6 M. Processing capital costs amount to USD95.9 M. Infrastructure capital amounts to USD6.3 M. Sustaining capital and closure cost provisions amount to USD9.3 M and USD5 M respectively. Contingency has been included at 15% and amounts to USD17.7 M. Table 9-2 summarises the capital costs over the Project life.

Table 9-2: Summary of Capital Costs

Capital Costs	LoM (USDM)
Mining	3.6
Processing	95.9
Tailings	2.7
Infrastructure	6.3
Sustaining Capital	9.9
Contingency - Capital	17.8
Closure Costs	5.0
Total Capital	**141.3**

10.4 Results

10.4.1 Cash Flow

The net cash flow is presented Figure 9-1 and Table 9-4 with key life of project parameters presented in Table 9-3.

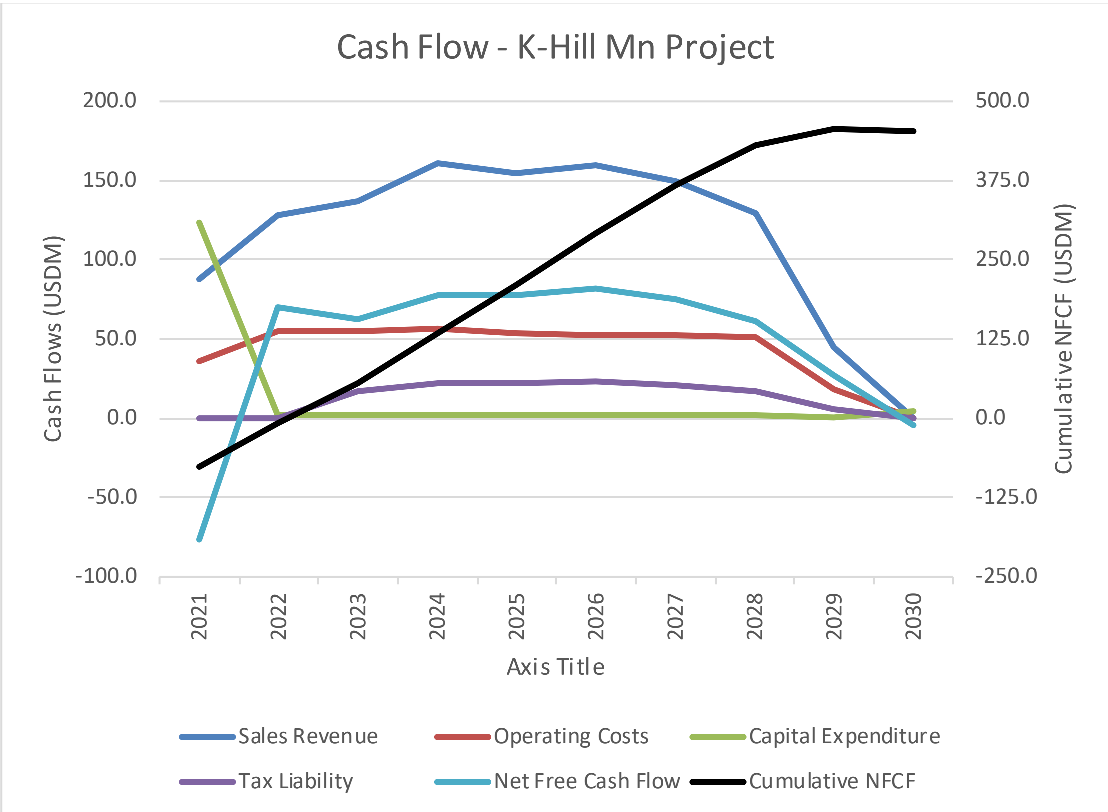


Figure 9-1: Net Cash Flow

Table 9-3: Summary of Key Life of Project Parameters

Summary of Key Parameters		
Revenue	(USDM)	1,152
Operating Costs	(USDM)	430
Operating Profit	**(USDM)**	**721**
Tax Liability	(USDM)	129
Capital Expenditure	(USDM)	141
Cash Flow	**(USDM)**	**451**
RoM Mined	(kt)	1,196
Waste Mined	(kt)	7,876
RoM Processed	(kt)	1,196
MnO Content	(kt)	354
MnO Recovered	(kt)	316
EMM Tonnes Produced	(kt)	245
Revenue	(USD/t EMM)	4,700
Operating Costs	(USD/t EMM)	1,756
Operating Profit	**(USD/t EMM)**	**2,944**

Table 9-4: Cash Flow Summary

Year			Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Base Case	Units	Total/Ave										
Production												
O/P												
Waste Mined	(kt)	7,876	1,405	1,292	1,216	1,806	891	416	443	337	71	0
RoM Mined	(kt)	1,196	93	150	150	148	150	150	150	150	55	0
Total Material Moved	(kt)	9,072	1,498	1,442	1,366	1,953	1,041	566	593	487	126	0
Stripping Ratio	(tw:to)	6.59	15.11	8.62	8.11	12.23	5.94	2.77	2.95	2.25	1.29	0
Rehandled	(kt)	0	0	0	0	0	0	0	0	0	0	0
RoM MnO Grade	(%)	29.64%	29.15%	26.28%	28.16%	33.55%	31.59%	32.76%	30.65%	26.45%	25.26%	0
MnO Content in Ore Mined	(kt)	354.4	27.1	39.4	42.2	49.5	47.4	49.1	46.0	39.7	13.9	0
Processing												
RoM Processed	(kt)	1,196	93	150	150	148	150	150	150	150	55	0
MnO Grade Processed	(%)	29.64%	29.15%	26.28%	28.16%	33.55%	31.59%	32.76%	30.65%	26.45%	25.26%	0
MnO Content	(kt)	354.4	27.1	39.4	42.2	49.5	47.4	49.1	46.0	39.7	13.9	0
MnO Process Recovery	(%)	89.3%	89.3%	89.3%	89.3%	89.3%	89.3%	89.3%	89.3%	89.3%	89.3%	0%
MnO Recovered	(kt)	316.4	24.2	35.2	37.7	44.2	42.3	43.9	41.1	35.4	12.4	0
EMM Tonnes Produced	(kt)	245.1	18.8	27.3	29.2	34.3	32.8	34.0	31.8	27.4	9.6	0
Macro Economics												
EMM Product Price	(USD/t)	4,700	4,700	4,700	4,700	4,700	4,700	4,700	4,700	4,700	4,700	0
Revenue												
EMM Revenue	(USDM)	1,151.9	88.1	128.1	137.3	161.0	154.0	159.7	149.4	129.0	45.1	0
Total Revenue	**(USDm)**	**1,151.9**	**88.1**	**128.1**	**137.3**	**161.0**	**154.0**	**159.7**	**149.4**	**129.0**	**45.1**	**0**
OPERATING COSTS, Real												
Mining	(USDM)	31.4	5.63	5.23	4.78	6.56	3.48	1.81	1.88	1.57	0.41	0.00
Rehandle	(USDM)	0.0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Processing	(USDM)	330.5	25.71	41.47	41.47	40.81	41.47	41.47	41.47	41.47	15.20	0.00
G&A	(USDM)	23.9	1.86	3.00	3.00	2.95	3.00	3.00	3.00	3.00	1.10	0.00
Selling Costs	(USDM)	44.6	3.41	4.96	5.32	6.24	5.97	6.19	5.79	5.00	1.75	0.00
Contingency	(USDM)	0.0	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Contingency Rate	(%)	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Total Operating Costs	**(USDM)**	**430.4**	**36.6**	**54.7**	**54.6**	**56.6**	**53.9**	**52.5**	**52.1**	**51.0**	**18.5**	**0.0**
CAPITAL COSTS, Real												
Mining	(USDM)	3.6	3.2	0.0	0.0	0.4	0.0	0.0	0.0	0.0	0.0	0.0
Processing	(USDM)	95.9	95.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Tailings	(USDM)	2.7	1.9	0.0	0.5	0.3	0.0	0.0	0.0	0.0	0.0	0.0
Infrastructure	(USDM)	6.3	6.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Sustaining Capital	(USDM)	9.9	0.0	1.4	1.4	1.4	1.3	1.3	1.3	1.3	0.5	0.0
Subtotal Capital	(USDM)	118.5	107.4	1.4	1.9	2.1	1.3	1.3	1.3	1.3	0.5	0.0
Contingency - Capital	(USDM)	17.8	16.1	0.2	0.3	0.3	0.2	0.2	0.2	0.2	0.1	0.0
Contingency Rate	(%)	15%	15%	15%	15%	15%	15%	15%	15%	15%	15%	0%
Closure Costs	(USDM)	5.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	5.0
Total Project Capital	**(USDM)**	**141.3**	**123.5**	**1.6**	**2.2**	**2.4**	**1.6**	**1.5**	**1.5**	**1.5**	**0.5**	**5.0**
Economics, Real: BASE DATE												
Sales Revenue	(USDM)	1,152	88.1	128.1	137.3	161.0	154.0	159.7	149.4	129.0	45.1	0.0
Operating Costs	(USDM)	430	36.6	54.7	54.6	56.6	53.9	52.5	52.1	51.0	18.5	0.0
Operating Profit - EBITDA	(USDM)	721	51.5	73.5	82.8	104.4	100.1	107.3	97.3	78.0	26.7	0.0
Tax Liability	(USDM)	129	0.0	0.0	17.7	22.4	21.7	23.3	21.1	16.8	5.8	0.0
Capital Expenditure	(USDM)	141	123.5	1.6	2.2	2.4	1.6	1.5	1.5	1.5	0.5	5.0
Working Capital	(USDM)	0	4.5	1.9	0.8	1.8	-0.4	0.6	-0.9	-1.7	-6.8	0.0
Net Free Cash Flow	**(USDM)**	**451**	**-76.5**	**69.9**	**62.1**	**77.7**	**77.2**	**81.9**	**75.6**	**61.3**	**27.2**	**-5.0**

10.4.2 Net Present Value

Net Present Values (NPV) of the cash flows are shown in Table 9-5 using discount rates from 0% to 15% in a post-tax context. SRK notes that at 10% discount rate the post-tax NPV for the Project is USD 285 M. NPV's in a pre-tax context are shown in Table 9-6. The Project IRR is 91% with payback in year 3.

Table 9-5: Summary of NPV's – Post Tax pre-finance

Base Case Summary of NPV's						
Discount Rate	**0%**	**5%**	**8%**	**10%**	**12%**	**15%**
NPV (USDM)	451	356	311	285	261	230

Table 9-6: Summary of NPV's – Pre-Tax pre-finance

Base Case Summary of NPV's						
Discount Rate	**0%**	**5%**	**8%**	**10%**	**12%**	**15%**
NPV (USDM)	580	459	402	369	339	300

10.4.3 Sensitivity Analysis

Figure 9-2 shows an NPV sensitivity chart for the Project operating costs; capital expenditure and revenue. The Project's NPV is most sensitive to revenue (grade or commodity price) as illustrated by the blue line in Figure 9-2. The Project has lower sensitivity to operating costs and is least sensitive to capital as indicated by the red line and green lines in Figure 9-2.

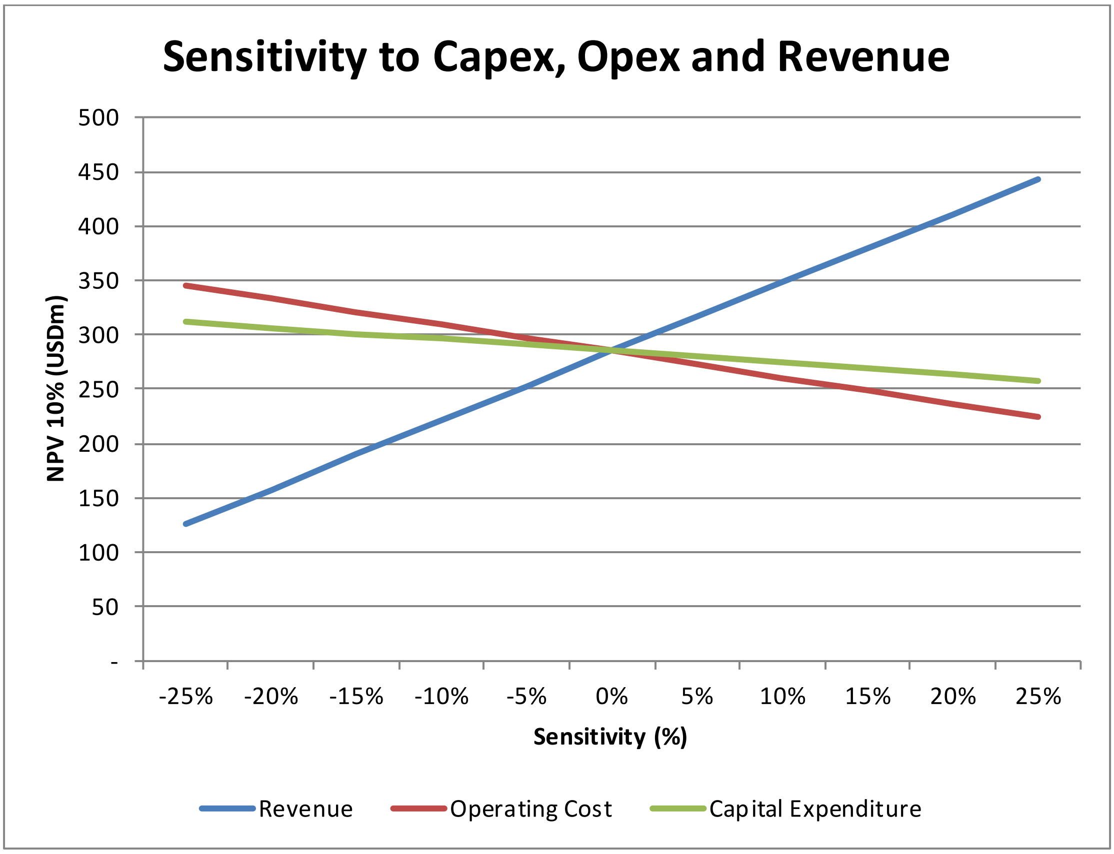


Figure 9-2: NPV Sensitivity

10.5 Conclusions

Based on the work carried out for this PEA, SRK concludes the following:

- The Project reflects a positive NPV of USD285 M at a 10% discount rate. Further investment and technical studies are therefore warranted
- Average operating costs have been estimated at USD360 /t milled
- Total capital expenditure has been estimated at USD141 M
- The Project NPV is most sensitive to revenue (grade or commodity price) but has lower sensitivity to operating costs and is least sensitive to capital costs
- The Project critically depends on achieving production of a high grade EMM product. If this grade of product cannot be achieved the value of the saleable product falls significantly
- The Client's tax advisers have indicated that they consider it likely that, instead of using the mining corporate tax formula, the flat Botswanan corporate tax rate of 22% can be applied due to the fact that the Company will be refining the extracted manganese ore into electrolytic manganese metal through a manufacturing process, accordingly a base corporate tax rate of 22 % has been used in the financial model. SRK notes that if the Company would decide to sell the extracted manganese ore as is, without refining, then the mining corporate tax formula is applied and the tax payable will be significantly higher.

10.6 Recommendations

Based on the work carried out for this study, SRK recommends the following:

- Staged progression of further technical work into a Feasibility Study
- Additional technical and field studies to determine that the key technical parameters that underpin the Project profitability are investigated in further detail. The critical technical works outstanding are the plant flowsheet design, operating costs, product prices and recoveries into a saleable EMM product
- Further drilling is required to bring the inferred resource up to indicated. The current gap between the two pits is the likely result of the lack of drilling in that area. Infill drilling may well prove the resource is present in this gap, and a programme targeting this area is recommended to potentially increase the resource
- As the Project progresses commence negotiation with the GoB to confirm the applicable tax regime for the Project

For and on behalf of SRK Consulting (UK) Limited

Mike Beare
Corporate Consultant (Mining)
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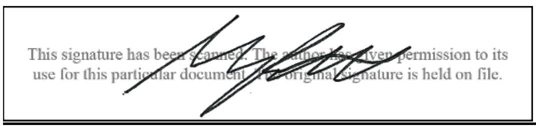

Onno ten Brinke
Principal Consultant (Mining Engineering)
SRK Consulting

APPENDIX

A ELECTROREFINING REPORT

Electro-Refining PEA Summary

The Process

Manganese produced from sulphate solution electrowinning typically doesn't achieve grades greater than about 99.5% as it incorporates or encapsulates a small part of the electrowinning solution deposits minor amounts of sulphur. In addition, iron, nickel, cobalt, lead or other metals are strongly selected against but may still grades in the tenths of a percent. To achieve a higher grade the manganese produced in the electrowinning is reprocessed. This allows the minor metal contaminates to be mostly eliminated and switching to a halide system helps to reduce the sulphur.

In this system, manganese electrode, previously plated, is dissolved into the halide solution. The dissolved Mn is the re-plated. This can be done at higher current densities, 800 A/m2 as opposed to 400 A/m2 at greater efficiency; 80% vs 40%. This decreases the size of the electrolysis plant and electrical consumption when compared to the first stage electrowinning. However, the ammonia used is consumed thus cannot be recycled and the minor amount of chlorine formed at the anode must be contained and dealt with. The halide acids are more corrosive and will require specialized materials.

The following chemical reactions are performed in this system:

Location	Reaction
Cathode	$Mn^{2+} + 2e^- = Mn$ $2H_2O + 2e^- = H_2 + 2(OH^-)$
Anode	$2Cl- = Cl_2 + 2e^-$ $3Cl_2 + 2NH_3 = N_2 + 6HCl$ $Mn^{2+} + 2H_2O = MnO_2 + 4H^+ + 2e^-$

Lower temperatures result in higher cathodic current efficiency and better deposition quality but this will be balanced by the cost of cooling. Conventional sulphur-based systems used temperatures in the range of 25-40C; the best temperature for the halide system, in terms of performance, is about -16C.

- Suggested anodes are graphite and cathodes are brass.
- NH^{4+} concentration of 2.0 mol/L
- Catholyte $MnCl_2.4H_2O$ at 298 g/L
- Diaphragm No. 84 filter paper or vinylidene chloride canvas
- pH 4.5~7.0
- Temperature below -10 °C

The higher current density and solution concentration means that the electro-refining tanks are likely to be about 40% the size of the electrowinning system. A furnace and roll will be needed to create the electrodes from the electrowinning product. Refrigeration will be required to reduce the solution temperature.

Scoping Study Cost Estimate

The total estimated cost of the hydrometallurgy electro-refining section is approximately 16 million USD not including contingency. This estimate is based on the raw materials for construction and fabrications costs. The circuit cost is based on a residence time of 8 hours and a current density of 800 A/m2 with a solution containing 10M Mn or 550 grams per liter.

Materials: the materials required for the equipment for all circuits was estimated. About 60% of the materials needs to be stainless steel. Refining tanks should be RFP or PVC due to the harsh nature of the chemicals. The average cost between these elements was assumed to be 3000 USD for materials and 3000 USD for fabrication.

Pipe and valves costs were estimated simply as 35% of the tank costs. This does not include the costs of hydraulic lines, water lines, or reagent lines that are included in other headings.

Hydraulics are the hydraulics controls, pumps, and lines to run the mixers, all filters, and pumps. Electrical systems may be considered as an alternative. This assumes one fixed hydraulic pump and one variable hydraulic pump to supply the motive forces for most of the plant.

Coatings are important for all surfaces that are, or could be in contact with bases or acids to prevent corrosion. This includes many of the tanks, but also the support framework of the equipment and anywhere that spillage could reach. It also includes coatings on building structures and other elements as small droplets of acid resulting from the mixing will cause widespread corrosion. Any locations close to mixers or moving parts should also have anti-slip coatings. The coating cost was assumed to be equal to 35% of the materials cost.

Pumps. Ten pumps are assumed to be required to pump acid, bases, and many also solids. All pumps will have spare parts and spare pumps should be stocked. This estimate assumes 20 pumps at an average cost of 10,000 USD.

Filters are typical belt filters modified to allow washing. These include leach remainders, iron and other metal precipitates, and two cases of MnO2 filtering; once filter for each line. Probably 10 filters at an average cost of 40,000. These do vary considerably in size.

Controls includes everything from monitoring and detection of pressure, pH, conductivity, level, flow rate, density, voltage, power draw, and others, to variable speed controls, distributed control systems, variable valves and connection hardware.

Ventilation includes duct work, control of pressures, air blowers and dust control. Ventilation is important whenever acids or bases are used. It is a critical system for electro-refining as minor amounts of Cl2 are evolved in the process. Ventillation and dust control was assumed to be equal to 1/6 of the prior plant system costs.

Transformers includes the main transformer for the plant, and sub-transformers for the elecro-winning and electro-refining. A 50% increase in the electrical capacity is approximately a 25% increase in costs over the base costs of the rest of the circuit.

Wires and buses are the conductors that power the electrowinning and electrorefining cells. These are typically 1" or larger diameter copper wire and heavy bars of copper. This number does not include plant wiring.

Rectification is the conversion of line AC to DC used by the electro-winning and electro-refining circuits. A cost of approximately 1.0/Amp was used. This is about a 50% increase in the conversation requirements, or an additional 25% on costs.

Structural is all the supports, walkways, access points, ladders, stairs and other elements that are not integral to the buildings themselves. These costs are assumed equal to the materials cost.

Regents systems includes the storage tanks, preparation tanks, metering pumps and controls, valves and distribution system of the reagents. This includes hydrochloric acid and sodium hydroxide. A cost of 50,000 for each system was assumed.

Recycle recovery includes the precipitation of unwanted metal ions such as iron and others, the removal of cations such as ammonia and the cleaning of recycled waters and organics within and between each sub-circuit.

Safety includes containment areas, pressure relieve, control of failure, eye wash stations and other health and safety requirements when dealing with acids and bases.

Building assumes a ground leveling, main building, ware house, offices, and all infrastructure support such as outside wiring, water and sewage. It doesn't include electrical substation. This number is highly variable depending on the site chosen. This value was assumed to be 10% of all prior building costs. A significant amount of this cost will be for insulation as the electro-refining rooms will be cold.

Electrode manufacturing furnace and roll for flatting, costs of approximately $400,000.

Refrigeration is required to lower the temperature of the solutions to approximately -7C. C. Costs associated with the cooling is an estimate subject to considerable error based on the costs of ice rink refrigeration systems.

Element	Electro-refining
Materials	500
Pipes/Values	250
Mixers	
Hydraulics	
Coatings	200
Pumps	200
Filters	100
Controls	
Ventilation	150
Transformers	50
Wires/Buses	50
Rectification	50
Controls (Electro)	50
Structural	500
Reagents Systems	50
Recycle/Recovery	200
Safety	100
Building	500
QC Lab	
Chemical storage	
Input Storage	
Warehouse	
Electrode manufacture	400
Refridgeration	500
Total	3150
Shipping	945
Taxes	945
Fabrication	11025
Total	16,065

Operating Costs

Electrical consumption as measured per tonne of refined manganese produced is approximately 3.5 MWhr for the electro-refining, 0.2 MWhr for refrigeration and 0.3 MWhr for electrode creation. The operating costs of the chemicals is less than the error in the electrical costs thus has not been considered.

Assuming the electricity costs are $0.06 per kWhr, or $60 per MWhr, and the feed grade is 30% Mn, the cost per ROM is $60*0.3 or $18 per ROM tonne.

Dr. Ian Flint
03-Aug-2019