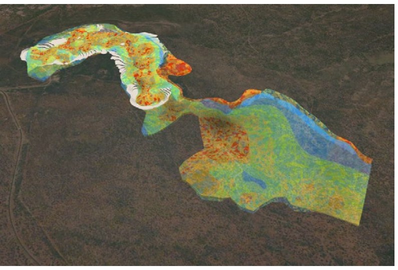
K.Hill Battery-Grade Manganese Project Preliminary Economic Assessment

National Instrument 43-101 Technical Report



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National Instrument 43-101 Technical Report

Prepared for:

Giyani Metals Corp.

Prepared by:

CSA Global South Africa (Pty) Ltd, an ERM Group company

Tetra Tech Europe

Knight Piésold Ltd.

Axe Valley Mining Consultants Ltd.

Qualified Persons:

Anton Geldenhuys, MEng, FGSSA, PrSciNat, CSA Global South Africa (Pty) Ltd, an ERM Group company

Howard Simpson, BSc Eng (Hons), BCom, FAusIMM(CP Mining), RPEQ, CSA Global South Africa (Pty) Ltd, an ERM Group company

Martiens Prinsloo, MSc, MBA, PriSciNat, ERM

Sifiso Siwela, PrSciNat, FGSSA, MSAIMM, CSA Global South Africa (Pty) Ltd, an ERM Group company

Rob Thomas, MSci ARSM, FAusIMM(CP), CEng, CSi, RenvP, MIMMM, CSA Global South Africa (Pty) Ltd, an ERM Group company

EUR ING Andrew Carter, BSc, CEng, MIMMM QMR, MSAIM, SME, Tetra Tech Europe

Matthew Randall, BSc (Hons), PhD, MIMMM, CEng, Axe Valley Mining Consultants Ltd.

Richard Elmer, BSc, MSc, CEng, MIMMM, MCSM, CEng, Knight Piésold Ltd.

Effective Date:

31st July 2023

Axe Valley Mining Consultants Ltd

Mob: +44 7891 108222

SIGNATURE PAGE

This report entitled *K.Hill Battery-Grade Manganese Project Preliminary Economic Assessment, National Instrument 43-101 Technical Report*, with an effective date of 31st July 2023, was prepared on behalf of Giyani Metals Corp. by the following Qualified Persons:

"signed and dated"	*"signed and dated"*
Anton Geldenhuys, MEng, FGSSA, PrSciNat CSA Global South Africa (Pty) Ltd, an ERM Group company	Howard Simpson, BSc Eng (Hons), BCom, FAusIMM(CP Mining), RPEQ CSA Global South Africa (Pty) Ltd, an ERM Group company
"signed and dated"	*"signed and dated"*
Martiens Prinsloo, MSc, MBA, PriSciNat ERM	Sifiso Siwela, PrSciNat, FGSSA, MSAIMM CSA Global South Africa (Pty) Ltd, an ERM Group company
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Rob Thomas, MSci ARSM, FAusIMM(CP), CEng, CSi, RenvP, MIMMM CSA Global South Africa (Pty) Ltd, an ERM Group company	EUR ING Andrew Carter, BSc, CEng, MIMMM QMR, MSAIM, SME Tetra Tech Europe
"signed and dated"	*"signed and dated"*
Matthew Randall, BSc (Hons), PhD, MIMMM, CEng Axe Valley Mining Consultants Ltd.	Richard Elmer, BSc, MSc, CEng, MIMMM, MCSM Knight Piésold Ltd.

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ABBREVIATIONS AND ACRONYMS

Ag	silver
Ai	Bond abrasion index test
AIMS	African Mineral Standards
Al	aluminium
Al_2O_3	aluminium oxide
As	arsenic
Au	gold
Axe Valley Mining	Axe Valley Mining Consultants Ltd.
Ba	barium
$Ba(OH)_2$	barium hydroxide
BaO	barium oxide
Be	beryllium
Bi	bismuth
BOCRA	Botswana Communications Regulatory Authority
BOD	biochemical oxygen demand
BOS	Botswana Bureau of Standards
BPC	Botswana Power Corporation
Bwi	Bond ball work index test
Ca	calcium
$CaCO_3$	limestone
CaF2	calcium fluoride

CAGR	compound annual growth rate
CaO	calcium oxide
CaOH	lime
Cd	cadmium
CDA	Canadian Dam Association
CDB	chert dolomite breccia
Ce	cerium
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
Co	cobalt
CO_2	carbon dioxide
COD	chemical oxygen demand
CONG	conglomerate
CPM	CPM Group LLC
Cr	chromium
CRM	certified reference material
CrO_2	chromium oxide
CSA	Canadian Securities Administrators
CSA Global	CSA Global South Africa (Pty) Ltd, an ERM Group company
csv	comma separated values
Cu	copper
CWi	Bond crushability work index test
D2EPHA	di(2-ethylhexyl)phosphoric acid
DAC	Design Acceptance Criteria
DC	direct current
DD	diamond drill (or drilling)
DDP	delivered duty paid
DEA	Department of Environmental Affairs
DGPS	differential global positioning system
EBITDA	earnings before interest, taxes, depreciation, and amortisation
EDS	energy dispersive spectroscopy
EIA	Environmental Impact Assessment
EIS	Environmental Impact Statement
EMP	Environmental Management Plan
EOH	end of hole
EPCM	engineering, procurement and construction management
ESMP	Environmental and Social Management Plans
FCA	free carrier
Fe	iron
$Fe_2(SO_4)_3$	Ferric sulphate
$Fe_2[SO_4]_3$	ferric sulphate
Fe_2O_3	iron oxide
FEL	felsite
FeO	ferrous oxide
Fe-shale	iron-shale

G&A	general and administrative
Ga	gallium
Ge	germanium
Giyani	Giyani Metals Corp.
GPS	global positioning system
H_2SO_3	sulphurous acid
H_2SO_4	sulphuric acid
HF	hydrofluoric acid
HPEMM	high-purity electrolytic manganese metal
HPMSM	high-purity manganese sulphate monohydrate
HPMSS	high-purity manganese sulphate solution
HSE	health, safety, and environment
ICP-OES	inductively coupled plasma-optical emission spectroscopy
IDW^2	inverse weighted distance to the power of two
In	indium
IP	induced polarisation
IRR	internal rate of return
ISO	International Organization for Standardization
JCI	Johannesburg Consolidated Investments Co. Ltd.
K	potassium
$K_2Mn_2(SO_4)_3$	manganolangbeinite
K_2O	potassium oxide
Knight Piésold	Knight Piésold Ltd.
La	lanthanum
Li	lithium
Loci	Loci Environmental (Pty) Ltd
LOI	loss on ignition
LOM	life of mine
LOP	life of project
Marble Lime	Marble Lime Associated Industries
MDG	Mmamokhasi Dam Group
Menzi Battery Metals	Menzi Battery Metals (Pty) Limited
Mg	magnesium
MgO	magnesium oxide
Mn	manganese
Mn	manganese
$Mn(OH)_2$	manganese hydroxide
Mn-clay (MCLAY)	manganiferous clay
MnO	manganese oxide
Mn-shale	manganiferous-shale
Mn-shale (MSH)	manganiferous-shale
Mo	molybdenum
MRE	Mineral Resource estimate
MSA Group	MSA Group (Pty) Ltd

MSM	manganese sulfate monohydrate
Na	sodium
Na_2O	sodium oxide
NaHS	sodium hydrosulphide
Nb	niobium
NFCF	net free cash flow
NH_3	ammonia
NH_4OH	ammonium hydroxide
Ni	nickel
NI 43-101	National Instrument 43-101
NMC	nickel-manganese-cobalt
NPV	net present value
OK	ordinary kriging
OLOM	original life of mine
P	phosphorus
P_2O_5	phosphorus pentoxide
Pb	lead
PEA	Preliminary Economic Assessment
PEP	Project Execution Plan
PL	prospecting licence
p-XRF	portable x-ray fluorescence
QA	quality assurance
QC	quality control
QEMSCAN	Quantitative Evaluation of Materials by Scanning Electron Microscopy
QP	Qualified Person
RC	reverse circulation
RES	Remote Exploration Services (Pty) Ltd.
RF	revenue factor
ROM	run of mine
RotsDrill	RotsDrill Botswana
RPEEE	reasonable prospects for eventual economic extraction
RTK	real-time kinematic
S	sulphur
SANAS	South African National Accreditation System
Sb	antimony
Se	selenium
SEP	Stakeholder Engagement Plan
SG	specific gravity
SGS Lakefield	SGS Canada Inc
SGS Randfontein	SGS Randfontein - Natural Resources
SHL	beige-cream shale
Si	silicon
SiO_2	silicon dioxide
Sn	tin

SO_2	sulphur dioxide
Sr	strontium
SRK	SRK Consulting (Kazakhstan) Ltd.
Ta	tantalum
Te	tellurium
Tetra Tech	Tetra Tech Europe
the K.Hill Project or the Project	K.Hill Battery-Grade Manganese Project
Ti	titanium
TiO_2	titanium dioxide
Tl	thallium
TMF	tailings management facility
TSS	total suspended solids
UTM	Universal Transverse Mercator
V	vanadium
V_2O_5	vanadium pentoxide
VoIP	Voice over Internet Protocol
WGS	World Geodetic System
WRD	waste rock dump
WTP	water treatment plant
WUC	Water Utilities Corporation
XRD	x-ray diffraction
XRF	x-ray fluorescence
XRF	x-ray fluorescence
Zn	zinc
Zr	zircon

MEASUREMENTS

$	dollars
%	percentage
<	less than
>	greater than
≤	less than or equal to
≥	greater than or equal to
°	degrees
°C	degrees Celsius
°F	degrees Fahrenheit
µm	microns
3D	three-dimensional
a	annum (year)
cm	centimetre
cm^3	cubic centimetre
D	Generalized Hoek-Brown disturbance factor
Eh	redox potential

g	gram
g/cm^3	gram per cubic centimetre
g/L	grams per litre
g/t	grams per tonne
Ga	billion years
GSI	geological strength index
ha	hectare
ha	horizontal
in	inch
kg	kilogram
kilovolt ampere	kVA
km	kilometres
km^2	square kilometre
kN	kilonewton
kN/m^3	kilonewton per cubic metre
kPa	kilopascal
kt	kilotonne
kt/a	kilotonne per year
kV	kilovolt
kW	kilowatt
kWh	kilowatt hour
L	litre
M	million
m	metre
m/m	metre by metre
m^2	square metre
m^3	cubic metre
masl	metres above sea level
masl	metres above sea level
mbgl	metres below ground level
mg/L	milligrams per litre
mi	Generalized Hoek-Brown constant
mm	millimetre
Mm^3	million cubic metres
MPa	megapascal
Mt	million tonnes
Mt/a	million tonnes per year
mV	millivolt
MW	megawatt
MWpdc	megawatt peak direct current
NTU	Nephelometric Turbidity Units
P	Botswana Pula
ppm	parts per million
R	South African Rand

t	tonne
t/a	tonnes per year
t/m^3	tonnes per cubic metre
UCS	unconfined compression strength
US$	US dollars
V	vertical
w/w	weight by weight (add to section)

1 SUMMARY

1.1 Introduction

Giyani Metals Corp. (Giyani), through its wholly owned local subsidiary Menzi Battery Metals (Pty) Limited (Menzi Battery Metals), intends to develop the K.Hill Battery-Grade Manganese Project (the K.Hill Project or the Project).

Giyani commissioned CSA Global South Africa (Pty) Ltd, an ERM Group company (CSA Global), Tetra Tech Europe (Tetra Tech), Axe Valley Mining Consultants Ltd. (Axe Valley Mining), and Knight Piésold Ltd. (Knight Piésold) to undertake a National Instrument 43-101 (NI 43-101) PEA-level Technical Report for the K.Hill Project.

The effective date of the Mineral Resource estimate (MRE) is 6th June 2023, and the effective date of the Preliminary Economic Assessment (PEA) is 31st July 2023.

Giyani has developed a bespoke process that can produce high-purity manganese sulphate monohydrate (HPMSM) directly from the high-grade K.Hill manganese oxide (MnO). HPMSM is a refined precursor material used to produce cathode powders for lithium-ion batteries for use in electric vehicles. This process avoids carbon-intensive calcination and electrorefining, providing the K.Hill Project with the opportunity to develop one of the lowest carbon footprints of any such facility globally. It is intended that the K.Hill project will be developed as an integrated mining and processing operation to manufacture HPMSM onsite.

The K.Hill Project name is derived from Kgwakgwe Hill, a manganese-rich outcrop located at the southern extent of the town of Kanye in southern Botswana. The orebody will be extracted from an open pit using conventional truck-and-shovel mining methods managed through a contractor mining execution strategy. The mine schedule focuses on early extraction of high-grade mineralized material. Mined mineralized material will be managed through dedicated high-, medium-, and low-grade stockpiles, which will ensure a continuous supply to the processing plant at the highest available grade.

This PEA evaluates a base case scenario that considers a single production line with a feed capacity of 200 kt/a to process manganese oxide material to produce HPMSM over a 57-year life of project (LOP; the LOP includes a 49-year life of mine [LOM] plus 8 years of stockpile rehandling). The PEA also evaluates an upside case, which assumes the construction of an additional production line from Year 5 of operations to increase total feed capacity to 400 kt/a, reducing the LOP to 31 years. The upside case is discussed in detail in Section 24.

The Project can deliver a strong return on investment over the 57-year LOP, as shown in an overview of the key project metrics presented in Table 1.1.

The K.Hill Project PEA includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. The results of the K.Hill Project PEA represent forward-looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the PEA. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially, are presented in the body of this report under each relevant section.

Table 1.1 Key Project metrics

Metric	Unit	Base case		
Project economics				
NPV (8% discount rate)	US$M	984		
IRR	%	29%		
Cumulative cash flow, undiscounted	US$M	5,283		
Project production		Year 1-5	Year 1-25	LOP
Total material mined	Mt	2.3	5.8	11.1
Average annual material processed	kt/a	170.0	194.0	196.0
Average plant feed grade	% MnO	19.1	17.3	15.2
Total HPMSM produced	kt	341	1,767	3,561
LOP	years	5	25	57
Net pricing assumptions				
Average realized HPMSM price (Years 1 to 5)	US$/t	3,559		
Average realized HPMSM price (Year 6 onwards)	US$/t	3,780		
Capital expenditure				
Total initial capital expenditure	US$M	282.6		
Total sustaining capital	US$M	18	142	288
Project cash flows				
Total revenue	US$M	1,214	6,620	13,387
Total operating costs (including royalty)	US$M	579	2,905	6,458
Total EBITDA	US$M	635	3,715	6,929

1.2 Property description and location

Giyani's licence extends over 1,960 km^2 of tenements, with the Project located in a 438 km^2 licence area, held under the Botswana-registered entity, Menzi Battery Metals. Giyani is the only shareholder and owns 100% of Menzi Battery Metals.

Giyani holds the exclusive right to engage in prospecting activities for "metals" within the Project area through prospecting licences (PLs) issued under Part IV of the Mines and Minerals Act of the Republic of Botswana. The Project is located within PL322/2016. All Giyani PLs, through its subsidiary Menzi Battery Metals, were renewed on 3^{rd} August 2022 and are valid until 30^{th} September 2024, with the exception of PL258/2017, which was renewed on 1^{st} January 2023 and expires on 31^{st} December 2024. According to the Mines and Mineral Act, the holder of a PL may, at any time not later than 3 months before the expiry of such licence, apply for renewal of the PL and shall be entitled to the grant of no more than two renewals thereof, each for the period applied for, which periods shall not in either case exceed 2 years, provided that (a) the applicant is not in default and (b) the proposed programme of prospecting operations is adequate (Government of Botswana 1999).

1.3 Accessibility, climate, local resources, infrastructure, and physiography

The Project area is well-networked via paved national roads, namely the A1 and A2 highways. The K.Hill Project is located within a few kilometres of the A2 highway, which runs from Buitepos at the Namibian border; through Jwaneng, Kanye, and Lobatse; to the South African border at Pioneer Gate, near Zeerust, South Africa. The A2 is a major component of the Trans-Kalahari Corridor, which is a highway corridor that provides a direct route from Maputo in Mozambique via Pretoria to central Namibia, to Windhoek and the port of Walvis Bay. The A1 runs from Gaborone (the capital of Botswana) past the junction with the A2. The Sir Seretse Khama International Airport in Gaborone provides links to major cities in southern and central Africa.

In Kanye, the summers are long, warm, and partly cloudy and the winters are short, cold, dry, and clear. Over the course of the year, the temperature typically varies from 4°C to 30°C and is rarely below 0°C or above 34°C. Kanye experiences significant seasonal variation in monthly rainfall. The climate is generally considered to be warm and arid with a summer rainfall season. Operations can continue throughout the year.

Kanye is at an elevation of approximately 1,300 masl, and Kgwakgwe Hill (after which the Project name is derived) forms a distinct topographical feature next to Kanye, with elevations reaching approximately1,500 masl. Various waste dumps, discards, stockpiled manganese mineralisation, and tailings occur in the K.Hill Project area, which together with cuts into the steep hillside from historical open pit mining, have disrupted the natural topography.

The K.Hill Project is located outside of Kanye, which has electricity supplied from the national grid. Kanye's water is supplied from the Mmamokhasi Dam, which is approximately 5 km from the Project site. Water from this source has been used for drilling activities.

1.4 Project history

The discovery of manganese in the Kanye area led to mining from 1957 to 1971 (Aldiss 1989). The first of many companies operating at K.Hill was Marble Lime Associated Industries (Marble Lime), which also developed the asbestos mine at Moshaneng, northwest of Kgwakgwe Hill. Marble Lime mostly mined the bedded-type mineralisation (described as manganiferous-shale [Mn-shale] during the Phase 1 mapping exercise completed by Giyani), which required beneficiation before it could be saleable (Aldiss 1989). Marble Lime ceased mining activities around 1967.

Further exploration work was carried out by Johannesburg Consolidated Investments Co. Ltd. (JCI), during the late 1960s and early 1970s.

In 1981, Rand London Manganese investigated the possibility of mining the Mn-shale deposits at K.Hill, Otse, and Gopane (near Lobatse), together feeding a single processing plant at Lobatse. The K.Hill deposit was considered to require further drilling for evaluation, but no further work was completed, and the licence was relinquished within a year (Aldiss 1989).

Historical production of manganese from K.Hill was reported by Baldock et al. (1977) to be 64,180 t from 1957 to 1967 and 131,563 t from 1968 to 1972. Variable prices were received for the product due to both metallurgical and high-grade battery-active products being supplied, with the latter attracting a premium at the time (Baldock et al. 1977).

1.5 Geology and mineralisation

1.5.1 Regional geology

The stratigraphy in the K.Hill Project 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 Archean basement in southeast Botswana is well studied on a regional scale, and maps are generally accurate. The prospect occurs within the mapped Transvaal Supergroup sediments, consisting of shales, quartzites, limestones, and conglomerates and in the vicinity of the Kanye Group (part of the Ventersdorp Supergroup), consisting of a variety of extrusive lavas and subordinate siltstones and shales.

1.5.2 Local geology

The mineralisation at the K.Hill Project 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, which suggests non-deposition of the intervening dolomites or a massive unconformity. The Kgwakgwe Chert Breccia Formation in the Kanye area can be subdivided into two main varieties:

- a dark-brown chert breccia with milk-white angular chert fragments, cemented together by brown haematitic material
- a reddish-brown chert breccia with abundant jaspilitic fragments and a high content of jasper in the matrix

Six lithologies were consistently intersected during drilling operations at the K.Hill Project. These include a chert or chert dolomite breccia unit, which occurs at the top of the stratigraphic sequence. Below that is an approximately 50 m thick package of shale identified to be part of the Black Reef Quartzite Formation, which hosts the manganese mineralisation within the shale units. Below this shale is an iron-rich shale, which is often intruded by manganese oxide material along fractures and joints. Below the iron shale (Fe-shale) lies a lower iron shale unit, typically a beige colour with no significant manganese content. A shale-containing manganese clay is also observed within the beige shale. Between the overlying shales and the felsite footwall unit lies a conglomerate marker unit, observed in almost all the drillholes. In some holes this marker conglomerate has a gritty texture with small clasts and is also mineralised with manganese oxides.

The Mn-shale horizon extends at surface in outcrop in the north and to the south below surface, where its presence has been intersected in drillholes. To date, the known strike length of the horizon is 1.9 km. The stratigraphy has been duplicated by thrusting in places, and, to the south, the mineralised horizon has been extensively downfaulted by steeply dipping east-north-northeast trending faults. The shales have been intensely folded and slumped in the vicinity of these dislocations, and in addition, subparallel breccia zones and quartz veining may be evident.

The entire Transvaal package is cut out against the Waterberg sediments to the west of Kgwakgwe Hill along what are thought to be northerly trending faults.

The Mn-shale outcrops along the northerly scarp slope of the Kgwakgwe Hill and dips into the hill. The strata at the K.Hill Project dip gently toward the northwest, at an average of approximately 5° to 10°. Numerous outcrops display parasitic folding, with local dips varying from 45° to almost subvertical. This is not consistent with the overall shallow dip of the Mn-shale and adjacent units. Where the deposit outcrops in the north, the Mn-shale unit is kidney shaped. The unit varies between approximately 2 m to 15 m thick, with an average thickness in the order of 4 m and has a delineated extent of approximately 1,900 m north-south and 350 m east-west. Some of the thicker intersections may be local fold duplications of a single horizon. In general, the southern extension area shows greater thicknesses of the mineralisation than in the north.

1.6 Exploration

Early exploration at the K.Hill Project was designed to geologically map and geochemically sample the prospect licence area. Outcropping mineralisation, as well as exposed mineralisation in the historical mining pits, were sampled. Various mineralisation styles were observed and sampled. Grab samples were taken from the K.Hill shale within and away from the Mn-shale outcrop.

Channel chip sampling was conducted at the K.Hill Project at two locations. The aim was to collect representative samples from the Mn-shale without any loss in material. The intended use of the samples was for future metallurgical test work.

Giyani engaged Remote Exploration Services (Pty) Ltd. (RES) to complete high-resolution ground gravity and ground magnetic surveys over the K.Hill Project. RES (2018) described the geophysical surveys as follows:

- The K.Hill Project ground gravity grid comprised 1,987 planned gravity stations. All gravity grids were planned with 50 m × 50 m spaced stations. Ground magnetic data were collected on the same survey lines as the gravity grids along north-northeast to south-southwest oriented lines at the K.Hill Project. All data were collected in continuous surveying (Walkmag) mode, which translates to a reading every 1 m to 2 m on 50 m spaced survey lines. In total, 101 km of magnetic data were collected over the K Hill Project. Three, 1 km induced polarisation (IP)/direct current (DC) traverses were undertaken based on the results from the gravity and magnetic data.
- Ground gravity station positions were measured using a Trimble R6 real-time kinematic (RTK) differential global positioning system (DGPS). Coordinates for the beacons were provided by the Botswana Department of Surveys and Mapping in Cape LO25 format with orthometric heights. These coordinates were transformed to World Geodetic System (WGS)84 Universal Transverse Mercator (UTM) 35S. Gravity station positions were marked and measured by walking a pattern that included approximately 5% internal repeats as well as approximately 5% external repeats. Following RTK surveying, gravity readings were taken over all stations. A Scintrex CG-5 Autograv gravity meter was used to complete the gravity survey.
- Magnetic data were collected using a GEM Systems Inc. (GEM) GSM19 Overhauser magnetometer in Walkmag at a 1-second sampling interval. A GEM proton precession magnetometer was used to monitor and correct for diurnal variations. Location data were collected with handheld GPS, which was time synchronised with both Walkmag and base station magnetometers.
- With regards to IP/DC data collection and to evaluate the effectiveness of IP/DC techniques, three (approximately), 1 km lines of IP/DC were collected. IP/DC traverses were designed to extend from felsic volcanic basement (Kanye Volcanic Formation), over the basal unit of the Transvaal

quartzites (Black Reef Quartzite Formation), into Lower Transvaal shales (Kgwakgwe Shale Formation), and into Upper Transvaal chert breccia (Kgwakgwe Chert Breccia Formation). The results of the ground gravity data were used to assist with the survey design. IP/DC data were collected in dipole-dipole configuration with a = 50 m and n = 1-7. A Zonge GDP32 receiver, a GGT10 10 kVA transmitter, and a ZMG 7.5 kVA generator were employed.

Drilling was undertaken in five programmes at the K.Hill Project: an initial diamond drilling (DD) programme of 18 holes completed in June 2018 totalling 1,109 m, a follow-up infill programme of 96 reverse circulation (RC) holes (including 6 redrills) that commenced in November 2020 and was completed in June 2021 totalling 3,346 m, and a synchronous DD programme of 11 holes for 346 m. The 2020/2021 DD programme was primarily completed for the purposes of geotechnical analysis, and as such, this core has not been subject to sampling and assaying.

Density determinations completed on the geotechnical holes were used in assigning density to the Mineral Resource model. The fourth phase (Phase 4) of drilling to delineate the southern area was completed in August 2021 and comprised 28 RC holes and 3 DD control holes, for a total of 2,126 m.

The final phase (Phase 5) was completed in August 2022 and included 6,116 m of RC and 216.60 m of DD. In total, 75 RC holes and 7 DD holes were completed. The aim of this drilling campaign was to increase the amount of drilling in the south and decrease the spacing between drillholes on a regular grid across extension of the deposit.

1.7 Sampling preparation, analysis, and data verification

1.7.1 Sample Preparation

Samples were dispatched in batches of approximately 100 samples. The sample batches were exported by Aramex (for the DD programme) and Pinnacle Express (for the RC programme) to SGS Randfontein in South Africa for geochemical analysis. The chain-of-custody was maintained by signature at every point in which the samples changed hands, from the core shed in Kanye, where the samples were stored, to the laboratory. As part of the laboratory procedure, all samples were weighed. All persons involved in the chain-of-custody were required to submit a copy of their receipt of handover of the samples to the project manager for record keeping on site.

Diamond drill core samples were prepared and analysed at SGS Randfontein. This is an independent commercial laboratory, which is International Organization for Standardization (ISO) 17025 accredited by the South African National Accreditation System for chemical analysis. The sample preparation method code is PRP87.

Reverse circulation sample preparation follows a procedure detailed in the LT20051R standard operating procedure.

1.7.2 Analysis

All samples were assayed at SGS Randfontein using method XRF76V, which assays major element oxides by x-ray fluorescence (XRF) using borate fusion. The oxides assayed included aluminium oxide (Al_2O_3), calcium oxide (CaO), chromium oxide (CrO_2), iron oxide (Fe_2O_3), magnesium oxide (MgO), manganese oxide, sodium oxide (Na_2O), phosphorus pentoxide (P_2O_5), potassium oxide (K_2O), silicon dioxide (SiO_2), titanium dioxide (TiO_2), and vanadium pentoxide (V_2O_5) reported in percent. Loss on ignition (LOI) was also determined.

1.7.3 Data verification

An extensive QA/QC programme was implemented to support assay data. The results of the QA/QC measures applied for the K.Hill Project do not indicate significant contamination and demonstrate a high degree of accuracy and precision. The check laboratory assays confirm the primary laboratory assays within close limits. Sampling and sample preparation methodologies are aligned with standard industry practice. Assay methodology is appropriate for the type of mineralisation.

The Qualified Person (QP), Mr. Anton Geldenhuys, MEng, FGSSA, PrSciNat, conducted a site visit for Mineral Resources from 20th to 21st April 2023. The QP accessed the K.Hill Project area by vehicle and explored on foot. Several outcrops and historic mining excavations, both surface and underground, were observed. Collar locations of four drillholes were verified in the field. The core processing facility was inspected with regards to core storage and quality. Drilling was not taking place at the time of the visit; therefore, physical sampling of core was not observed. Drill core and RC drilling chips from four holes were inspected relative to the original geological logs and digital geological logs contained within the supplied database. All logging data in the original and digital logs were verified by means of cross checks against the two log types and the core. During the inspection of the four holes by the QP, assays contained in the database were checked relative to geological logging and the core for signs of mineralisation. Mineralisation was evident in the core, and in all instances, mineralisation occurring in the core was evident in the recorded logging and assays.

In 2022, SRK undertook an analysis of the twin drilling completed by Giyani, comparing Phase 1 DD holes against the Phase 2 RC holes. CSA Global updated the study with newly acquired twin data.

The QP considers the drilling, sampling, assaying, and QA/QC procedures utilised by Giyani have resulted in data that is of sufficient quality to support a MRE. The QP confirms verification the data to the extent described above and confirms its suitability and adequacy as such.

1.8 Mineral processing and metallurgical testing

Based on historical metallurgical test work reports and reference materials, Tetra Tech implemented a test work and metallurgical development programme to develop a flow sheet for the metallurgical extraction of manganese (Mn) from the K.Hill Project manganese deposit.

Documentation was available from SRK Consulting (Kazakhstan) Ltd. (SRK), MSA Group (Pty) Ltd, Vietti Slurrytec, Lab 4 Inc., and Dalhousie University. The documentation included mineralogy, chemical analysis, leach testing, and solvent extraction testing.

The test work initiated by Tetra Tech was undertaken at Mintek, Johannesburg, South Africa and included assays, specific gravity (SG), mineralogy, comminution, solid-liquid separation, leach optimisation, jarosite precipitation, iron (Fe) and aluminium (Al) precipitation, other base metal precipitation, calcium (Ca) and magnesium (Mg) precipitation, fluoride removal, crystallisation, and manganese hydroxide ($Mn[OH]_2$) precipitation.

In addition, samples of the B Horizon material, a separate mineralised zone within the K.Hill deposit, were also tested; these samples were subjected to mineralogical tests, chemical assaying, and leach extraction test work. The purpose of this series of tests was to understand any differences in metallurgical extraction characteristics between the B Horizon material and the original LOM (OLOM) composite sample selected for test work.

A modelled (i.e., non-measured) mine plan (Section 16) indicates that mined material may differ somewhat from the OLOM sample that was used to complete the process design; however, the impacts on the process design are within the existing design allowances and are not expected to influence the process estimates.

1.8.1 Comminution development

Comminution test work indicates that the K.Hill mineralized material, from both the OLOM sample and B Horizon material, displays comminution characteristics that vary from soft to hard in terms of crushability and grinding indices, and are generally low in relation to abrasiveness.

1.8.2 Solid/liquid separation development

The solid/liquid testing indicates that post-leach, the K.Hill Project material settles poorly and produces low-density thickener underflows. On this basis, the decision was made to use filtration for washing and separation rather than conventional thickening.

1.8.3 Leach development

The leach performance was investigated and indicates that a high temperature (90˚C) reductive leach in sulphate media produces excellent extraction results of between 95% and 99%. Initial work conducted by Lab 4 Inc. using sucrose as the reductant was replaced by sulphur dioxide (SO_2) as a reductant, which provided benefits in terms of cost, practicality, and reduced acid consumption. The B Horizon material showed very similar results to the OLOM material.

1.8.4 Precipitation and purification development

Test work indicates that the stage-wise precipitation of contaminants is effective and efficient. Manipulation of pH and addition of aqueous reagents, as well as the use of activated alumina, allow the production of a high-purity manganese stock solution that is suitable for crystallisation of HPMSM.

1.8.5 Crystallisation development

Two routes were pursued for the basis of design development:

- use of synthetic solution produced from analytical reagents
- work from Mintek to produce a solution from representative mineralized material that had been processed step-by-step through the purification scheme

The synthetic solution provided a means for rapid and inexpensive production of crystalliser feed, allowing the development of initial process feed purity requirements and expected operational criteria. The Mintek work yielded a fully representative stock solution, which provided the opportunity to complete the design of the crystalliser and provide a sample of a significant amount of process accurate HPMSM for acceptance testing and trials with customers.

1.8.6 Overall recovery

The primary purpose of the processing plant is to produce HPMSM for the rapidly developing and growing battery metals market. From the test work and process development, an overall recovery of 88.5% of manganese is anticipated.

1.9 Mineral Resource estimates

CSA Global completed an MRE for Giyani's K.Hill Project deposit. Ms. Susan Oswald (Senior Resource Consultant) conducted the modelling, estimation, and Mineral Resource classification. Mr. Anton Geldenhuys (Principal Resource Consultant) peer reviewed the MRE and is the QP.

The current MRE has an effective date of 6th June 2023, was prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) *CIM Definitions and Standards on Mineral Resources & Mineral Reserves* (CIM 2014) and is reported in accordance with the Canadian Securities Administrators (CSA) NI 43-101 *Standards of Disclosure for Mineral Projects* (CSA 2016a), Companion Policy 43-101CP (CSA 2016b), and Form 43-101F1 (CSA 2011).

This MRE is based on interpretations from assays and geological logging. Apart from the initial sample data preparation and intermediate spreadsheet processing, all interpretations, modelling, estimation, and model validation was conducted using Leapfrog™ and Datamine StudioRM™ software. Snowden Supervisor™ software was used to conduct statistical analysis.

The MRE workflow can broadly be summarised as follows:

- data validation and preparation
- interpretation of the geology and mineralisation domains
- coding, compositing, and capping of composites
- exploratory data analysis and statistical analysis
- variogram analysis
- block model construction
- grade interpolation
- block model validation
- density assignment
- reasonable prospects for eventual economic extraction
- Mineral Resource classification and Mineral Resource reporting

CIM *Definition Standards for Mineral Resources & Mineral Reserves* (CIM 2014) require that resources have reasonable prospects for eventual economic extraction (RPEEE). This generally implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off grade, considering possible extraction scenarios and processing recoveries. To satisfy the requirement of reasonable prospects for economic extraction by open pit mining, reporting pit shells were determined based on conceptual parameters and costs supplied by Giyani and reviewed for reasonableness by the QP.

The Mineral Resource is reported above a cut-off grade of 7.3% manganese oxide and comprises 8.6 Mt of Indicated material at a grade of 15.2% manganese oxide and 6.1 Mt of Inferred Material at a grade of 14.1% manganese oxide. Mineralisation below the reporting pit shell is not considered as Mineral Resource, as it does not have reasonable prospects for economic extraction at the time of reporting (Table 1.2).

Table 1.2 K.Hill Project Mineral Resource at a cut-off grade of 7.3% manganese oxide as of 6th June 2023

Mineral Resource Classification	Tonnage (Mt)	MnO (%)	Al_2O_3 (%)	SiO_2 (%)	Fe_2O_3 (%)	LOI (%)
Indicated	8.6	15.2	9.1	49.5	12.2	7.2
Inferred	6.1	14.1	8.0	53.5	11.4	6.7

Notes:
a) The Mineral Resource has been classified and reported under the guidelines defined by the Canadian Institute of Mining, Metallurgy and Petroleum in their document *CIM Definition Standards for Mineral Resources and Mineral Reserves* (CIM 2014).
b) Mineral Resources are not Mineral Reserves and have not demonstrated economic viability.
c) Mineral Resources are stated as in situ dry tonnes; figures are reported in metric tonnes.
d) Figures have been rounded to the appropriate level of precision for the reporting of Mineral Resources.
e) Estimation has been completed within six different mineralization domains.
f) Mineral Resources are reported assuming open pit mining methods.
g) The Mineral Resource is reported within a conceptual pit shell determined using a price of US$3,800/t HPMSM (equivalent to US$9,054/t manganese oxide), conceptual parameters, and costs to support assumptions relating to reasonable prospects for eventual economic extraction.
h) The Mineral Resource is reported at a cut-off grade of 7.3% manganese oxide.
i) The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. CSA Global is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other any other relevant factors affecting the MRE.
j) HPMSM price quoted is based on 2022 market data, which was available at the time of reporting the Mineral Resource. Additional pricing information will be available for input into subsequent technical studies, and this may impact on the Mineral Resource reported.

Reported Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no guarantee that all or any part of a Mineral Resource will be converted into a Mineral Reserve.

1.10 Mining methods

The K.Hill Project will be developed as an open pit mining operation with a target processing plant feed rate of 200 kt/a. Just over 11.1 Mt of mineralized material is above a cut-off grade of 7.4% manganese oxide, which will be mined and processed at an average diluted head grade of 15.2%

manganese oxide. The average waste stripping ratio is 11:1. In the initial 5 years of production, the plant will process 817kt of mineralized material at an average diluted head grade of 19.1% manganese oxide.

It should be noted that the PEA mine plan includes Inferred material; consequently, the term mineral inventory has been used to define material that can be sent to the processing plant and processed economically. The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized.

Due to the relatively continuous nature of the deposit and the minor low-grade mineralisation that exists along much of the mineralised boundary, the impact of both dilution and loss is expected to be minimal to the Project economics. Modifying factors of 5% dilution and 95% recovery were assumed in the open pit optimisation and were subsequently updated to 3% dilution and 98% recovery in the mineral inventory estimate.

The pit optimisation was run using an HPMSM price of US$3,800/t and a total processing cost (including a general and administrative [G&A] cost) of US$531/t processed to generate a series of nested pit shells by varying the revenue factor (RF). The pit limit was then selected at an RF between 0.6 and 0.7, which equates to a metal price of US$2,280/t to US$2,660/t of HPMSM.

A series of five pit stages, or pushbacks, were designed to simulate the proposed mining sequence. The mineral inventory for each pit stage was then calculated for each 10 m bench and used to develop the mine schedule in quarterly (for the first 5 years) and annual periods.

Mining will occur through one, 12-hour day shift, 7 days per week, using 10 m benches (divided into flitches of 2.5 m or 5 m, as required, to minimise material loss and waste dilution) with a maximum slope angle of 41°. The near surface weathered material is generally soft in nature and, with some exceptions, can be mined without blasting.

As the mine progresses to the south, the deeper horizons are expected to be harder and will require ripping and/or blasting. To ensure continuous operations with minimal downtime due to unforeseen hardness, drilling and blasting have been assumed for up to 50% of the material mined.

Medium-sized excavators (3.8 m^3) will be used in combination with 30 t articulated dump trucks to feed either the ROM pad or the low-grade stockpile located near the processing plant. The material will then be rehandled using a wheel loader and fed into the primary crusher.

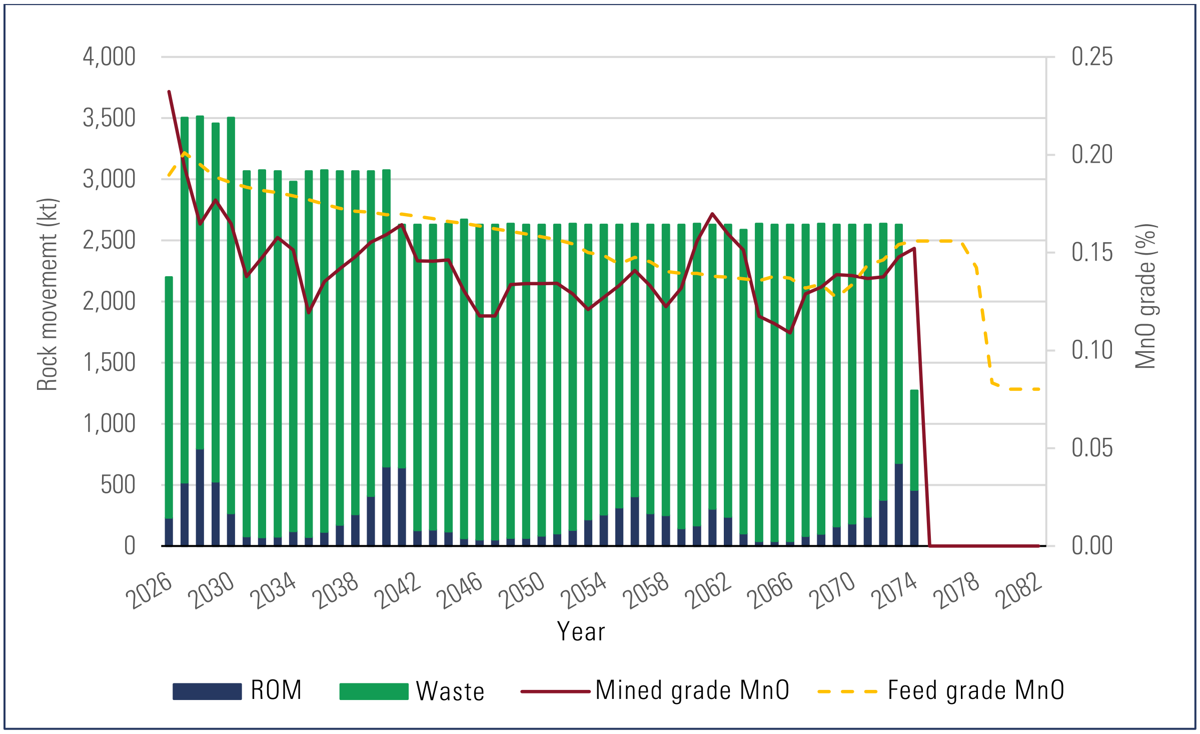
The ROM pad will be separated into several “fingers” separating material by grade and material type to facilitate blending. For this PEA, the mineral inventory has been split into waste (<7.4% manganese oxide), low grade (<9.0% manganese oxide), medium grade (<10% manganese oxide), and high grade (≥10% manganese oxide) so that a high grading policy can be used in the early years to maximise Project value. In the early years of the Project, lower-grade material (<10% manganese oxide) will be stockpiled separately from the ROM pad.

Waste will be transported to the nearest waste rock dump (WRD), located to the east of the open pit. Material will be dumped and levelled using a designated track dozer.

The LOM production schedule runs over a period of 49 years, in which both direct and rehandled material will be fed to the processing plant. The total mining rate has been capped at approximately 3.5 Mt/a to provide the required material quantities over the LOM. This 49-year period is followed by 8 years of stockpile rehandling to fill the processing plant to the end of mine life, making the total LOP 57 years.

The processing plant's ramp-up, scheduled over a period of 2 years, will be achieved by directly feeding the required material from the open pit to the processing plant. The manganese oxide head grade will remain relatively stable throughout years of mining and will remain constant during the stockpile rehandling period. The annual mining schedule is shown on Figure 1.1.

Figure 1.1 Life of mine schedule



1.10.1 Hydrological considerations

General hydrological conditions

The K.Hill Project site is located at the top of Kgwakgwe Hill, which acts as a localised hydrological catchment divide, with runoff draining in a both an eastern and western direction into local, non-perennial streams. Runoff from the eastern side of Kgwakgwe Hill, which also drains the southern slopes of Kanye, flows approximately 2 km into the Mmamokhasi Dam, which has a capacity of approximately 250,000 m^3. Runoff from the western slopes of the Kgwakgwe Hill flows approximately 3 km south into the Nneneke River, which then converges with runoff from the eastern slopes (via the Mamakhosi Dam) into the Matlhapise River, which eventually flows into the Taung River.

General hydrogeological conditions

The Project area is underlain by a low-permeability fractured rock aquifer system that is associated with shales. In general, this aquifer is a poor yielding aquifer with a blow yield of approximately

0.5 m^3/hour); however, at greater depths where fractures are encountered, higher yields can be expected. Overall, the aquifer underlying the mining area is not regarded as being suitable for groundwater supply purposes. The area to the south and to the west of the mining area comprises mainly dolomitic aquifers in which significantly higher yields and high-flow yields of up to 50 m^3/hour can be expected.

The depth to groundwater on a regional scale ranges between 30 mbgl to over 150 mbgl. The deeper groundwater levels measured in the dolomitic aquifers around water supply wells are interpreted as being due to groundwater abstraction, which has caused depressions in the groundwater table and do not necessarily represent steady-state levels. Near the open pit, groundwater levels (measured in 2020) are approximately 45 mbgl. In the vicinity of the open pit, the groundwater flow is generally in a south-southeast direction.

The groundwater quality in the region is generally regarded as being good, when compared to the Botswana Bureau of Standards (BOS) drinking water specification. Only slightly elevated manganese (Mn) and iron concentrations exceeding the drinking water standards were found within the groundwater supply wells that were sampled. All other ions complied with BOS 32:2015 standards.

Mine inflows and dewatering

The surface water runoff into the pit was calculated based on the available updated open pit layout at the end of life of operations and is expected to represent the maximum expected inflow volume. Surface runoff into the open pit area is expected to be approximately 85,850 m^3/day in the event of a 1:100-year return period storm event (equivalent to 3,577 m^3/hour). This does not take into consideration possible backfilling of the open pit. It should be noted that this calculation was completed a PEA level of accuracy, and it is recommended that a full surface runoff calculation be completed as part of further studies.

With the natural groundwater level within the footprint of the open pit, lying at approximately 1,300 masl, it is assumed that any portion of the open pit that extends below 1,300 masl will intercept the groundwater level and require dewatering. The open pit design indicates that will include the central and southern portions of the main open pit. The southern extension pit is shown to have a minimum elevation of approximately 1,297 masl; therefore, some groundwater inflows into the southern extension pit can also be expected.

Groundwater inflow volumes into the excavated main open pit at the end of the LOM were calculated using analytical methods in the range of 430 m^3/day to 690 m^3/day. Groundwater inflows into the southern extension pit are expected to be approximately 40 m^3/day to 50 m^3/day. Analytical methods are not ideally suited to calculating groundwater inflows in a dynamic scenario, such as an active mine; therefore, it is recommended that inflow calculations be updated using a 3D numerical groundwater flow model.

1.10.2 Geotechnical considerations

Available data was reviewed and used to support a geotechnical assessment of open pit slopes suitable for a PEA level of analysis; this includes data from the previously completed Technical Report (SRK and Tetra Tech 2022) and additional geotechnical logging data more relevant to the southern, deeper extent of the expanded MRE.

A geotechnical model was developed based primarily on the geological model developed for the MRE and the rock mass conditions derived from available geotechnical data. In some areas, the proposed pit walls are located beyond the limits of the geological model; where this is the case, the lithological unit contacts were extrapolated and are of lower confidence.

Limit equilibrium methods were used in stability analyses to determine the suitability of the proposed open pit slope angles. Stability analyses on representative sections of the open pit design indicate that the sections meet design acceptance criteria; therefore, the open pit design is considered appropriate for a PEA.

1.11 Recovery methods

The processing plant will treat 200 kt/a of ROM mineralised material from the open pit at an average grade of 15.2% manganese oxide to produce HPMSM. In the initial 5 years of production, an average grade of 19.1% manganese oxide is treated.

The mineralised material comprises manganese and iron shales and is moderately hard and amenable to reductive acid leaching in sulphate media using sulphur dioxide as a reductant. The process comprises crushing and grinding to reduce the ROM mineralised material to a characteristic grind (P_{80}) of 150 µm, an acid reductive leach in sulphate media at an elevated temperature using sulphur dioxide as a reductant, and a sequential purification process for the removal of metal impurities. Fluoride polishing is undertaken to further improve the purity of the solution. The purified solution then undergoes evaporative crystallisation followed by filtration and drying of the product to produce an HPMSM final product. The solids removed during sequential purification and fluoride polishing will be disposed of in the tailings management facility (TMF) or stored as an intermediate product. All liquors removed in the treatment of the mineralized material will be treated for reuse or, where they meet required environmental standards, used for haul road dust suppression.

1.12 Project infrastructure

The K.Hill Project will require the development of multiple infrastructure items. The locations of Project facilities and other infrastructure items were selected to take advantage of local topography, accommodate environmental considerations, and ensure efficient and convenient operation of the mine haul fleet.

The following Project infrastructure and facilities include:

- crushing facility, including ROM pad and stockpiles, three-stage crushing plant, and a crushed material bin
- processing area, including grinding, extraction, purification, fluoride polishing, crystallisation, product storage and handling, water treatment, reagent storage and tails handling; sulphur dioxide plant and plant infrastructure and utilities, including steam and air plants and low-voltage switch rooms
- additional infrastructure, including gatehouse and weighbridges; laboratory, maintenance workshop; tyre and lube storage; administration building, including first-aid and firefighting facilities; explosives storage; and fuel farm
- water systems to supply, treat, and distribute plant water, fire water, and potable water

- site and haul roads
- electrical high- and medium-voltage substations and power distribution to all facilities via two, 11 kV feeder circuits
- communications infrastructure
- temporary construction facilities
- off-site infrastructure, including a solar plant and access roads

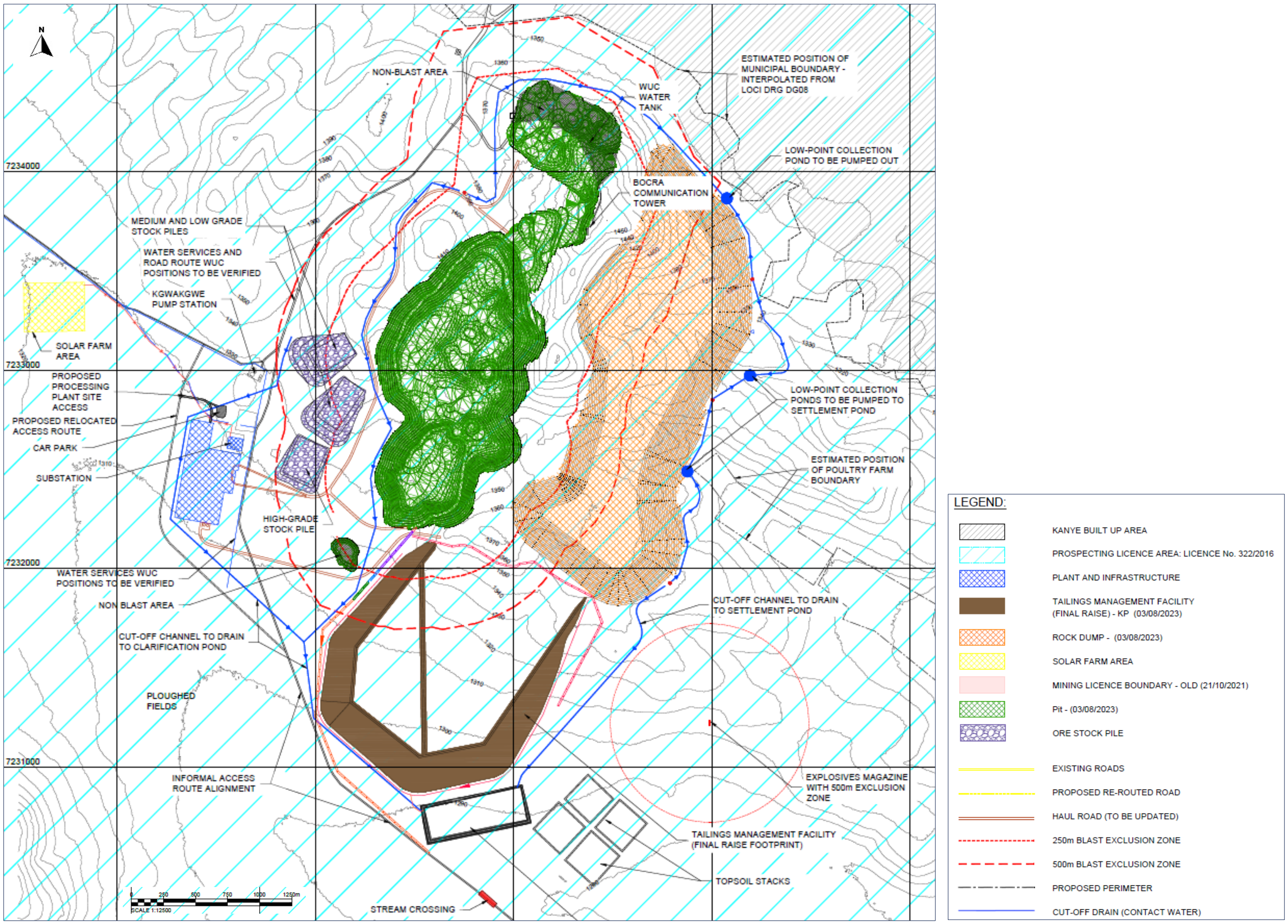
A 4.5 MWp_{dc} solar plant, covering 7.6 ha, will be constructed 1.2 km west-northwest of the processing plant entrance gate. Power generation models predict excellent solar production at the K.Hill Project, greater than 74% of the theoretical maximum production, a very good value in solar generation.

Botswana Power Corporation (BPC) will install a 33 kV high-voltage power line from the main Kanye municipal substation, located north of the town, to a new high-voltage substation next to the processing plant. BPC will also install the 33 kV high-voltage substation, converting the incoming high voltage supply from 33 kV to 11 kV.

Water for mineralized material processing will be supplied from the municipal water supply to limit the volume of impurities entering the process. The processing plant will be self-contained and a zero-runoff facility. Additional rainwater, collected in the processing plant bunded areas, will be pumped back into the process.

An overall site layout for the Project is shown on Figure 1.2.

Figure 1.2 K.Hill Project site layout



Source: Tetra Tech

1.13 Tailings management facility

The TMF has been designed to accommodate an estimated filtered tailings production of 23.3 Mt. Due to the process of extracting manganese from the mineralized material, the tailings mass to be stored is estimated to be double the mass of the mined mineralized material.

For the PEA, Knight Piésold have designed a conventional tailings impoundment. Based on a reported solids content of 63%, Knight Piésold has assumed that the filtered tailings will form a highly thickened material with a geotechnical moisture content (59%) that exceeds its liquid limit and optimum moisture content, and as such a "dry stack" facility is not considered for the PEA.

Table 1.3 presents the design criteria used in the TMF design.

Table 1.3 Tailings management facility design criteria

Description	Criteria	Source
Total LOP throughput (Mt)	23.3	Tetra Tech
LOP (years)	57	Axe Valley Mining
Annual production rate (t/a)	416,880	Tetra Tech
Total required capacity (Mm^3)	16.2	Calculated
Tailings transportation method	Trucked in tankers	Knight Piésold design
Tailings solids content by mass (%)	63	Tetra Tech
Tailings deposition method	Direct discharge into impoundment	Knight Piésold design
Tailings density (t/m^3)	1.44	Adopted from SRK and Tetra Tech (2022)
Crest width (m)	8	Knight Piésold design
Embankment slope	1V:3H	Knight Piésold design
Tailings storage method	Within downstream raise impoundment	Knight Piésold design

Notes:
V - vertical
H - horizontal

Tailings will be transported using tankers and deposited into the TMF along the perimeter embankment and from an inter-cell bund.

Perimeter embankments will be raised using a downstream method of construction. At this stage, it is envisaged that waste rock from the open pit will be suitable to construct the embankment; however, to confirm this, testing (geotechnical and geochemical) in the next phase of the Project will need to be conducted.

Knight Piésold adopted a filtered tailings density of 1.44 t/m^3 for the PEA based on material testing. It is recommended that confirmation of the density be completed at the next stage of this project.

A consequence classification of the TMF was undertaken based on the methods provided in the *Global Industry Standard on Tailings Management* (GTR 2021). The TMF has been assigned a classification of high, which is considered conservative, as the design could include mitigation measures to reduce the consequence classification during later stages of the project when more information on the tailing's properties is known.

The TMF will be constructed as a fully lined facility to prevent seepage of contact water from the tailings to the environment. Such an option will improve the water recovery and storage at the settlement pond in order for water to be reused in the process.

The stability analysis results show that all the scenarios assessed achieved a factor of safety above the Canadian Dam Association guideline values.

Diversion channels have been designed to intercept and divert non-contact water before it reaches the TMF and perimeter channels, and a drain tower within the TMF impoundment has been included to capture contact water and direct it to the settlement pond.

The TMF will be closed by placing the removed subsoil and topsoil over the new TMF landform, which will be vegetated to minimise water ingress and erosion while decreasing the potential for dust generation once the final raise has been constructed.

1.14 Market studies and contracts

As the automotive industry increases electric vehicle production as part of a global move toward decarbonisation and electrification, CPM Group LLC (CPM; 2022) reports that demand for lithium-ion batteries used in electric vehicles is projected to grow by 25% annually between 2021 and 2031. CPM demonstrates that the resulting demand for HPMSM in lithium-ion batteries will increase nearly 30-fold between 2021 and 2036, reaching 1.8 Mt on a contained metal basis which will increase up to 4.5 Mt by 2050. Currently, more than 90% of global HPMSM production capacity is in China, and only six non-Chinese high-purity manganese projects are forecast to come on stream in the next 5 years.

1.15 Environmental studies, permitting, and social or community impact

1.15.1 Environmental Impact Assessment

Loci Environmental (Pty) Ltd (Loci) prepared an Environmental Impact Assessment (EIA) for the K.Hill Project and an Environmental Impact Statement (EIS) based on the work completed for the EIA. The EIS was submitted to the Botswana Department of Environmental Affairs (DEA) in July 2023 (Giyani 2023). The EIA includes a series of specialist studies covering archaeology and cultural heritage, biodiversity, hydrogeology and geochemistry, hydrology, traffic, noise, air quality, landscape and visual amenity, waste management, health, social, and rehabilitation/mine closure.

Public consultation was initiated during the EIA scoping stage for public consultation methodology. Consultation continued throughout the EIA process as the Project scope was refined and additional information became available.

The K.Hill Project will comply with Botswana legislation and current policies, legal requirements, regional conventions, and international obligations to which Botswana is a signatory. Giyani has opted to conform with the International Finance Corporation (IFC) *Performance Standards on Environmental and Social Sustainability* (IFC 2012) and the *Equator Principles EP4* (Equator Principles Association 2020).

The EIA assesses only the first 11 years of mine operations; impact assessments beyond the first 11 years of the LOM will need to be assessed by an independent environmental consultant. It is anticipated that any changes would not be material in terms of the EIA process and could be done through amendments to the existing EIA. The process of amending the EIA is well stipulated by the DEA.

1.15.2 Environmental Management Plan

A Project Environmental Management Plan (EMP) was developed to manage the environmental and social impacts predicted during the EIA. An EMP provides a framework for mitigating environmental impacts associated with the Project and its activities. This includes a summary of all potential environmental impacts expected during all phases of the Project (i.e., construction, operations, and rehabilitation/closure) and mitigation measures that address each impact and assign roles and responsibilities for personnel who will implement the EMP.

1.15.3 Monitoring

A Monitoring Plan will be put in place to make sure the Project complies with environmental requirements, as per legislation and standards. For each possible impact, monitoring includes the parameter to be monitored, monitoring locations, key performance indicators, the agent responsible for monitoring, monitoring methodology, frequency, reporting mechanism, threshold, and recommended action when threshold is exceeded. Monitoring will also give an indication of the effectiveness of mitigation measures and provide for improvements where required.

1.15.4 Closure and decommissioning

A Mine Closure Plan will be developed for the Project, with the primary goal of leaving behind an enduring and positive legacy after mining, in which closure planning has been adequately resourced and integrated to facilitate effective transition of the mine lease area back to land authorities or other third parties. Post-Project land use and landscaping will form part of the consultation exercise for closure.

1.16 Capital and operating costs

1.16.1 Capital costs

Capital cost estimates were prepared for initial, sustaining, and closure capital.

The total initial capital cost estimate for the K.Hill Project is US$282.6M, including a contingency of US$62.5M. The initial capital cost estimate includes direct and indirect costs associated with project execution. In addition, the operating cost for the mine pre-strip is capitalised, totalling US$1.1M.

Capital costs incurred after start-up are designated as sustaining capital and are projected to be paid out of operating cash flows. The annual sustaining capital allowance for each cost area was estimated over the 57-year LOP and totals US$287.7M. The closure cost estimate is US$8.4M.

The capital cost estimate breakdown by cost area is shown in Table 1.4.

Table 1.4 Capital cost summary by cost area

Cost area	Cost (US$M)
Mining	2.0
Processing	94.1
Infrastructure and Services	37.1
TMF	9.2
Indirect costs	48.8
Construction overheads	20.7
Owner's costs	8.2
Total initial capital cost (excluding contingency)	**220.1**
Contingency (28%)	62.5
Total initial capital cost (including contingency)	**282.6**
Sustaining capital (including in contingency)	288.0
Closure cost	8.4
Total Project capital costs	**579.1**

Note: The sum of costs may differ from the total due to rounding.

1.16.2 Operating costs

The operating cost estimate for the K.Hill Project consists of mining, processing, TMF, and G&A costs, as summarised in Table 1.5.

The total estimated LOP average unit operating cost is US$579/t of mineralised material processed (Table 1.5). Mining accounts for 7.4% of the total operating cost, processing accounts for 89.4%, G&A accounts for 3.0%, and the TMF accounts for 0.2%.

Table 1.5 Average unit operating cost summary by cost area

Cost area	Unit Cost (US$/t processed)	Contribution to operating cost (%)
Mining	43	7.4
Processing	517	89.4
TMF	1	0.2
G&A	18	3.0
Total	**579**	**100**

Note: The sum of costs may differ from the total due to rounding.

1.17 Economic analysis

In collaboration with Tetra Tech and Axe Valley Mining, CSA Global prepared an economic analysis for the K.Hill Project. Cash inflows are based on annual production and revenue projects, while cash outflows consist of capital costs, operating costs, royalties, and taxes. The modelling period covers the LOP of 57 years, including a 2-year construction period for the processing plant, a 2-year ramp up to full production following processing plant commissioning, and an 8-year processing period after mining has ceased, in which stockpiled material will be fed to the processing plant.

The Project NPV was calculated by discounting back cashflow projections through the LOP at 8%. Key project metrics are presented in Table 1.6.

The K.Hill Project PEA includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. The results of the K.Hill Project PEA represent forward-looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the PEA. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially, are presented in the body of this report under each relevant section.

Table 1.6 Key Project Economic Metrics

Metric	Unit	Base case		
Project economics				
NPV (8% discount rate)	US$M	984		
IRR	%	29%		
Cumulative cash flow, undiscounted	US$M	5,283		
Project production		Year 1-5	Year 1-25	LOP
Total material mined	Mt	2.3	5.8	11.1
Average annual material processed	kt/a	170.0	194.0	196.0
Average plant feed grade	% MnO	19.1	17.3	15.2
Total HPMSM produced	kt	341	1,767	3,561
LOP	years	5	25	57
Net pricing assumptions				
Average realized HPMSM price (Years 1 to 5)	US$/t	3,559		
Average realized HPMSM price (Year 6 onwards)	US$/t	3,780		
Capital expenditure				
Total initial capital expenditure	US$M	282.6		
Total sustaining capital	US$M	18	142	288
Project cash flows				
Total revenue	US$M	1,214	6,620	13,387
Total operating costs (including royalty)	US$M	579	2,905	6,458
Total EBITDA	US$M	635	3,715	6,929

1.18 Interpretations and conclusions

1.18.1 Geology and Mineral Resources

The updated MRE contained in the Technical Report was prepared using data from 187 RC and DD holes from all drilling campaigns conducted over the Project since 2018, totalling 10,710 m. This additional drilling included 40 step-out holes along strike into a previously untested, but mineralized, section to the south.

This collated and complete set of drilling data, together with a new density determination programme and updated optimization parameters for the constrained pit shell, resulted in the conversion of Inferred to Indicated Mineral Resources resulting in an overall increase in Indicted Mineral Resources of more than 300% and an increase of more than 100% in Inferred Mineral Resources. The MRE reported has been restricted to all classified material located within an optimized pit shell.

The Mineral Resource is reported above a cut-off grade of 7.3% manganese oxide and comprises 8.6 Mt of Indicated material at a grade of 15.2% manganese oxide and 6.1 Mt of Inferred Material at a grade of 14.1% manganese oxide. Mineralisation below the reporting pit shell is not considered as Mineral Resource, as it does not have reasonable prospects for economic extraction at the time of reporting.

1.18.2 Processing

Mineralogical investigations indicate that most of the manganese is associated with the minerals cryptomelane and bixbyite. When subjected to a reducing leach using sulphur dioxide in sulphate media, manganese extraction is greater than 90%.

Test work undertaken included assays, SG, mineralogy, comminution, solid-liquid separation, leach optimisation, jarosite precipitation, iron and aluminium precipitation, other base metal precipitation, calcium and magnesium precipitation, fluoride removal, crystallisation, and manganese hydroxide precipitation.

The metallurgical investigations also confirm the findings of the mineralogical studies, where the optimal grind size is estimated to be a P_{80} of 150 µm, yielding a manganese extraction of 97%. The manganese losses during the purification process were found to be 0.5% during the precipitation stage and 4.0% in the fluoride polishing stage; together with a 97% recovery by precipitation from the crystalliser bleed recycle stream yielded an overall manganese recovery of 88.5%.

1.18.3 Mining

The final pit design contains a mineral inventory, including Inferred material, of 11.1 Mt at a diluted grade of 15.2% manganese oxide, above a marginal cut-off of 7.4% manganese oxide. The pit design inventory includes a total contained metal quantity of 1,691 kt manganese oxide. The assumed pit design parameters are supported by the geotechnical assessment implementing a 41° overall slope.

The planned processing plant throughput rate is 200 kt/a; therefore, the LOM plan has 49 years of mining with another 8 years of stockpile rehandling of low-grade material, which results in a total LOP of 57 years.

Due to the relatively continuous nature of the deposit, the impact of both dilution and loss, at 3% dilution and 98% recovery in the mineral inventory estimate, is expected to be minimal to the Project economics. The pit optimisation was run using an HPMSM price of US$3,800/t and a total processing cost of US$531/t to generate a series of nested pit shells by varying the RF. The pit limit was then selected at an RF between 0.6 and 0.7, which equates to a metal price of US$2,280/t to US$2,660/t of HPMSM.

The processing plant's ramp-up, scheduled over a period of 2 years, will be achieved by directly feeding the required material from the open pit to the processing plant. The Project will use a contract mining operation and the average mining costs have been estimated at US$2.50/t mined for waste and the US$3.00/t for ROM mineralized material.

Geotechnical

A geotechnical model was developed based primarily on the geological model developed for the MRE and the rock mass conditions derived from available geotechnical data. In some areas, the proposed pit walls are located beyond the limits of the geological model; where this is the case, the lithological unit contacts were extrapolated and are of lower confidence.

Limit equilibrium methods were used in stability analyses to determine the suitability of the proposed open pit slope angles. Stability analyses on representative sections of the open pit design indicate that the sections meet design acceptance criteria; therefore, the open pit design is considered appropriate for a PEA.

Hydrology/hydrogeology

The updated mining sequence will breach the groundwater level at approximately 1,300 masl leading to groundwater inflows into the open pit. The expected inflow volumes will range 430 m^3/day to 690 m^3/day. There will also be surface water runoff into the pit, and it is calculated that surface runoff into the open pit area is expected to be approximately 85,850 m^3/day in the event of a 1:100-year return event storm. The combined groundwater and surface water inflows will require dewatering.

1.18.4 Project Infrastructure

The locations of Project facilities and other infrastructure items were selected to take advantage of local topography, accommodate environmental considerations, and ensure efficient and convenient operation of the mine haul fleet. Permanent infrastructure has been positioned outside the 250 m blast radius of all open pits, and the administration and maintenance areas are outside the 500 m blast radius of all pits. Additional major infrastructure, such as the explosives storage, have been positioned in accordance with standard African mining practice, and the fuel farm has been located on the corner of the ROM pad.

Site utilities, which include water, power, and communications, will run alongside the public road from Kanye and will enter the site close to the processing plant. The water supply will be piped to the water treatment area, power will be distributed via two, 10 kV ring mains and communication masts will be strategically positioned around the site.

1.18.5 Tailings management facility

The TMF has been designed to accommodate a significant increase in estimated filtered tailings production of 23.3 Mt as compared to previous studies. Due to the process of extracting manganese from the mineralized material, the tailings mass to be stored is estimated to be double the mass of the mined mineralized material. The TMF will be located southeast of the processing plant. Tailings material will leave the processing plant in the form of highly thickened tailings at an estimated 60% to 70% solids ratio. The tailings will be trucked in tankers to the filtered TMF. A conventional tailings impoundment has therefore been adopted for the PEA. The TMF will consist of a fully lined facility and a settlement pond in which surface runoff and seepage water will be collected.

The TMF embankment will be constructed in five phases using the downstream construction method. The embankment will be constructed of borrowed waste rock. An inter-cell bund will be constructed for decanting and tailings deposition. An assessment of embankment slope stability was conducted as part of the PEA design to demonstrate that the TMF external slopes meet the accepted criteria set out in the Canadian Dam Association (CDA) guidelines.

1.18.6 Environmental studies, permitting, and social or community impact

The current EIA indicate that the K.Hill Project will cause environmental impacts, both negative (e.g., dust raised in working areas) and positive (e.g., employment opportunities and increased economic activity). Mitigation measures will vary with each phase of the Project, with limited additional impacts during operations. Regular monitoring by Giyani will be required to make sure that the Project environmental standards are achieved (e.g., water, dust).

The EIA assesses only the first 11 years of the LOP; impact assessments beyond the first 11 years will need to be assessed by an independent environmental consultant. It is anticipated that any changes would not be material in terms of the EIA process and could be done through amendments to the existing EIA. The process of amending the EIA is well stipulated by the DEA.

1.19 Recommendations

It is recommended that post-PEA work be completed to advance the Technical Report to a Feasibility level for the current MRE and mitigate risk, including the following work:

- It is reasonable to expect that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued infill drilling. Infill drilling is recommended to improve the Mineral Resource classification.
- Metallurgical test work should include variability sampling in line with the mine plan and test work completed on the samples to confirm material extraction, solution purification requirements, and reagent consumption data as well as further detailed engineering studies in relation to mill design, bulk materials handling requirements, slurry rheology, dissolution, and precipitation performance. It is further recommended that Giyani evaluate production of agricultural-grade manganese sulphate as a potential outlet for off-specification material set aside for repurification or as a potential source of high-grade manganese in the event of major process interruptions.
- Open pit optimisations should be rerun when additional operating cost and/or geological information is available to establish the optimal pit limit to be used for pit design as well as estimation of modifying factors for material loss and dilution.
- A numerical groundwater model in which inflows into the pit are simulated should be undertaken. The model can also be used to calculate the zone of influence of groundwater level drawdown and pore pressure in the pit walls for input into the pit design.
- The existing mine water balance should also be updated to ensure that enough water storage capacity is designed and installed at the processing plant site to effectively manage the additional volume of storm water and groundwater pit inflows. Giyani should re-evaluate processing plant water demand to ensure that the identified water supply options will be sufficient to meet mining and processing plant water demands.
- A geotechnical investigation of units in the vicinity of the walls of the proposed pit should be conducted. Geotechnical studies for the potential for reducing waste stripping requirements by steepening the open pit slopes should be evaluated through a comprehensive geotechnical analysis of both North and South pit areas. Utilising information gained from the proposed

geotechnical drilling programme to define the strength properties, geotechnical parameters for the WRD and the low-grade stockpile.

- A geotechnical assessment for the site infrastructure, buildings, TMF, and WRD areas as well as mineralised material and waste geochemistry testing and acid rock drainage test work.
- A TMF ground investigation to determine potential contaminants and evaluate other possible solutions to prevent seepage to the environment. Piezometer installations, sampling and monitoring at revised TMF locations to determine geotechnical and groundwater conditions. Materials testing to inform the in-situ density for updated capacity analysis and strength parameters for updated stability analysis.
- The demonstration plant should be set up to validate the processing flowsheet, mitigate commercial processing plant risk, and facilitate off-take qualification.
- Approval of the EIA for LOM extension needs to be obtained. Specialist environmental studies need to be completed upon receipt of the EIS approval from the DEA.
- A geochemical assessment of the material that will be mined, processed, and stored must be conducted in order to determine the potential for producing acid rock drainage, as well as the long-term quality of leachate emanating from the different stockpiles and facilities.
- Further detailed technical and economic analyses is warranted to increase the degree of accuracy in the estimates and the economics of the Project. Further work on capital and operating cost estimation will be required for the next level of study. The true cost of mining contracting needs to be investigated, especially around mobilization and demobilization over the life of the Project. Updating of costs can further refine the upside potential of the K.Hill Project.

Recommendations and associated budgets are detailed in Section 26.

2 INTRODUCTION

2.1 Issuer

Giyani trades on the TSX Venture Exchange, part of the Toronto Stock Exchange, under the stock symbol EMM, and on the Frankfurt Stock Exchange, under the stock symbol KT9: GR. Giyani's strategy is to become a responsible, low-carbon producer of battery materials for the electric vehicle industry and currently focuses on the development of manganese projects in the Republic of Botswana.

2.2 Terms of reference

Giyani commissioned CSA Global, Tetra Tech, Axe Valley Mining, and Knight Piésold to undertake an NI 43-101 PEA-level Technical Report for the K.Hill Project, located in the Republic of Botswana, in which Giyani holds a 100% interest through its local, wholly owned subsidiary, Menzi Battery Metals.

This NI 43-101 Technical Report has been prepared to comply with disclosure and reporting requirements set forth in the (CSA *National Instrument 43-101 Standards of Disclosure for Mineral Projects* (CSA 2016a), *Companion Policy 43-101CP* (CSA 2016b), *Form 43-101 F1 Technical Report* (CSA 2011), and the CIM *CIM Definition Standards for Mineral Resources & Mineral Reserves* (CIM 2014).

2.2.1 Independence

Neither CSA Global, Tetra Tech, Axe Valley Mining, Knight Piésold, nor the QPs of this Technical Report has or have had previously any material interest in Giyani or the mineral properties in which Giyani has an interest. The relationship with Giyani is solely one of professional association between client and independent consultant. This report is prepared in return for professional fees based upon agreed commercial rates, and the payment of these fees is in no way contingent on the results of this report.

2.2.2 Element of Risk

The interpretations and conclusions reached in this report are based on current geological theory and the best evidence available to the author at the time of writing. It is the nature of all scientific conclusions that they are founded on an assessment of probabilities and, however high these probabilities might be, they make no claim for absolute certainty. Any economic decisions which might be taken on the basis of interpretations or conclusions contained in this report will therefore carry an element of risk.

2.2.3 Units

Unless otherwise stated, all monetary figures expressed in this report are in US dollars (US$) or Botswana Pula (P).

The metric system, or International System of Units, is the primary system of measure and length used in this Technical Report and is generally expressed in kilometres (km), metres (m), and centimetres (cm); volume is expressed as cubic metres (m^3), mass expressed as metric tonnes (t), area as hectares (ha) or square kilometres (km^2).

The coordinate reference frame for this Project is WGS84 and the grid reference is UTM 35 South.

2.3 Principal sources of information

CSA Global, Tetra Tech, Axe Valley Mining, and Knight Piésold based their technical work on information provided by Giyani, along with technical reports from consultants, government agencies, and other relevant published and unpublished data. Section 27 includes sources of information used in this PEA. The QPs have endeavoured, by making all reasonable enquiries, to confirm the authenticity and completeness of the technical data upon which the PEA is based.

2.4 Qualified Person section responsibility

The QPs for this NI 43-101 Technical Report are listed by section in Table 2.1.

Table 2.1 Qualified Persons by report section

Section no.	Section name	Consultant	QP
1	Summary	All	All
2	Introduction	CSA Global	Anton Geldenhuys
3	Reliance on other experts	CSA Global CSA Global	Anton Geldenhuys Sifiso Siwela
4	Property description and location	CSA Global	Anton Geldenhuys
5	Accessibility, climate, local resources, infrastructure, and physiography	CSA Global	Anton Geldenhuys
6	History	CSA Global	Anton Geldenhuys
7	Geological setting and mineralisation	CSA Global	Anton Geldenhuys
8	Deposit types	CSA Global	Anton Geldenhuys
9	Exploration	CSA Global	Anton Geldenhuys
10	Drilling	CSA Global	Anton Geldenhuys
11	Sample preparation, analyses, and security	CSA Global	Anton Geldenhuys
12	Data verification	CSA Global	Anton Geldenhuys
13	Mineral processing and metallurgical testing	Tetra Tech	Andrew Carter
14	Mineral resource estimates	CSA Global	Anton Geldenhuys
15	Mineral reserve estimates	Axe Valley Mining	Matthew Randall
16	Mining methods	Axe Valley Mining CSA Global CSA Global	Matthew Randall Rob Thomas Martiens Prinsloo
17	Recovery methods	Tetra Tech	Andrew Carter
18	Project infrastructure	Tetra Tech CSA Global Knight Piésold	Andrew Carter Martiens Prinsloo Richard Elmer
19	Market studies and contracts	CSA Global	Sifiso Siwela
20	Environmental studies, permitting and social or community impact	CSA Global	Sifiso Siwela
21	Capital and operating costs	CSA Global Tetra Tech Knight Piésold	Howard Simpson Andrew Carter Richard Elmer

Section no.	Section name	Consultant	QP
22	Economic analysis	CSA Global	Howard Simpson
23	Adjacent properties	CSA Global	Anton Geldenhuys
24	Other relevant data and information	Axe Valley Mining CSA Global	Matthew Randall Howard Simpson
25	Interpretations and conclusions	All	All
26	Recommendations	All	All
27	References	All	All

2.5 Qualified Person site inspections

The following QPs visited the K.Hill Project site:

- Mr. Anton Geldenhuys, MEng, FGSSA, PrSciNat, conducted a site visit for Mineral Resources from 20th to 21st April 2023.

3 RELIANCE ON OTHER EXPERTS

3.1 Geology and Mineral Resources

Although Giyani has provided additional background information, it should be noted that the QP, Mr. Anton Geldenhuys, MEng, MGSSA, PrSciNat, has not relied on any information provided by Giyani concerning legal, political, environmental. or tax matters relating to the K.Hill Project. This information has been supplied to the QP through personal communications with Giyani staff, provision of technical information and data, and uploading relevant information to a project data room during 2023. Technical conversations via email and online teleconferencing have been regularly held with various Giyani staff, primarily Mr. Luhann Theron, Chief Geologist, and Mr. Jacques du Toit, Vice President, Technical Services, in 2023. The QP has been provided with information regarding permits; however, the QP has not independently verified the status of, nor legal titles relating to, the mineral concessions. The QP has also not independently verified nor undertaken any due diligence regarding the legal and tax aspects relating to the Project.

3.2 Environmental matters

Mr. Sifiso Siwela, PrSciNat, FGSSA, MSAIMM, of CSA Global, relied on Mr. Tom Steytler, B.Sc., E&S Lead with Giyani Metals Corp., and Ms. Chrisna Klopper, Director, Biodiversity with Loci, for information regarding environmental studies, permitting, and social or community impact for the K.Hill Project. This reliance is based on the EIS Giyani submitted to the Botswana DEA in July 2023 (Loci 2023), which is summarised in Section 20.

No warranty or guarantee, be it express or implied, is made by CSA Global with respect to the completeness or accuracy of the legal or environmental aspects of this document. CSA Global does not undertake or accept any responsibility or liability in any way whatsoever to any person or entity in respect of these parts of this document, or any errors in or omissions from it, whether arising from negligence or any other basis in law whatsoever.

3.3 Market studies

Mr. Sifiso Siwela, PrSciNat, FGSSA, MSAIMM, of CSA Global, relied on a market study completed by CPM (2022), which is summarised in Section 19 of this Technical Report. The CPM market study assesses the supply and demand dynamics of the HPMSM market, based on projected consumption by the electric vehicle battery market versus known and estimated sources of future production to determine a long-term average price for HPMSM.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location of property

The K.Hill Project is located adjacent to the town of Kanye, which is the administrative centre of the Southern District of the Republic of Botswana. The Project is accessible via a short section of unpaved roads and tracks from a network of paved national roads, namely the A1 and A2, with the A2 highway located just a few kilometres from the Project. Gaborone, the governmental and economic capital city of Botswana, is approximately 100 km by paved road from Kanye (Figure 4.1).

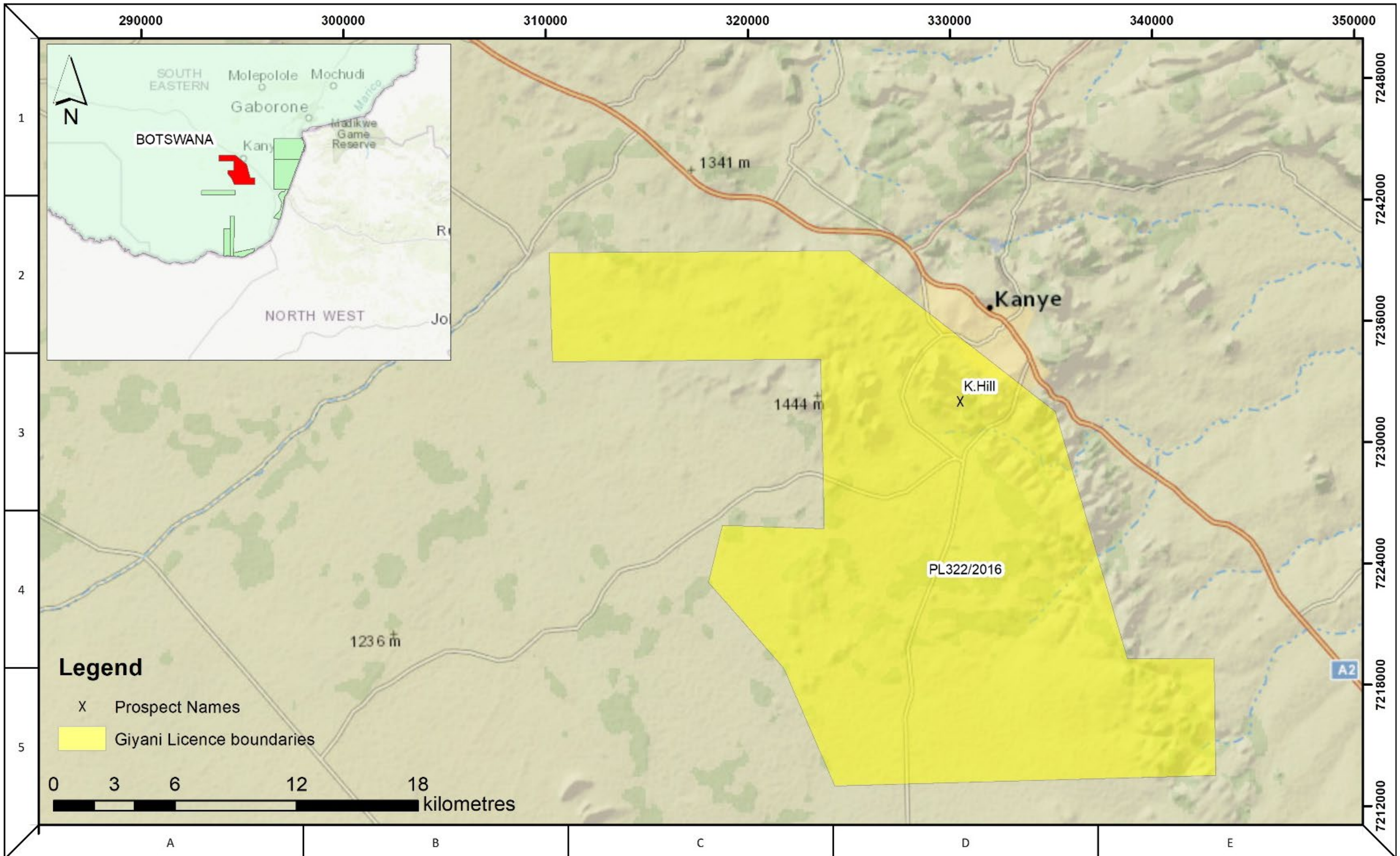
Giyani's licence extends over 1,960 km^2 of tenements, with the Project located in a 438 km^2 licence area, held under the Botswana-registered entity, Menzi Battery Metals. Giyani is the only shareholder and owns 100% of Menzi Battery Metals.

4.2 Mineral tenure and surface rights

Giyani holds the exclusive right to engage in prospecting activities for "metals" within the Project area through PLs issued under Part IV of the Mines and Minerals Act of the Republic of Botswana. The Project is located within PL322/2016. All Giyani PLs, through its subsidiary Menzi Battery Metals, were renewed on 3rd August 2022 and are valid until 30th September 2024, with the exception of PL258/2017, which was renewed on 1st January 2023 and expires on 31st December 2024 (Table 4.1). According to the Mines and Mineral Act, the holder of a PL may, at any time not later than 3 months before the expiry of such licence, apply for renewal of the PL and shall be entitled to the grant of no more than two renewals thereof, each for the period applied for, which periods shall not in either case exceed 2 years, provided that (a) the applicant is not in default and (b) the proposed programme of prospecting operations is adequate (Government of Botswana 1999).

The location of PL322/2016 in relation to Giyani's other PLs is shown on Figure 4.2. The PL is on land classified as tribal and is managed directly by the Kanye Sub Land Board which falls under the Ngwaketse Land Board, which in turn is overseen by the Ngwaketse Tribal Administration with the leadership of the Paramount Chief Kgosi Malope II. The Ngwaketse Tribal Administration manages the land on behalf of the community across parts of the larger area surrounding the Project. Water for drilling has been sourced from the Mmamokhasi Dam, which is approximately 3.4 km from the Project. Agreements were signed with the Mmamokhasi Village Development Committee in 2018 (now expired) and 2021 (now expired), which gave Giyani permission to extract water for an agreed sum of P15,000 per month. The Mmamokhasi Village Development Committee is an elected tribal authority that represents the local community in an area bordering the dam. A third water agreement was signed in 2022 with the Mmamokhasi Dam Group (MDG); the MDG is the body that has overall responsibility for the dam. The Chief of the area in which the Mmamokhasi Dam is located, facilitated a meeting with the Mmamokhasi Village Development Committee, MDG, and Giyani representatives, and it was agreed this third agreement should be with the MDG. This agreement was specifically for the use of water during drilling activities and has since expired. Giyani does not hold permission to extract water from the dam at the effective date of this report.

Figure 4.1 Location of the K.Hill Project and prospecting licence number PL322/2016



Rev	Date	Revision Description	DRAWN	CHCK'D	APPV'D
A	22/10/19		CN		

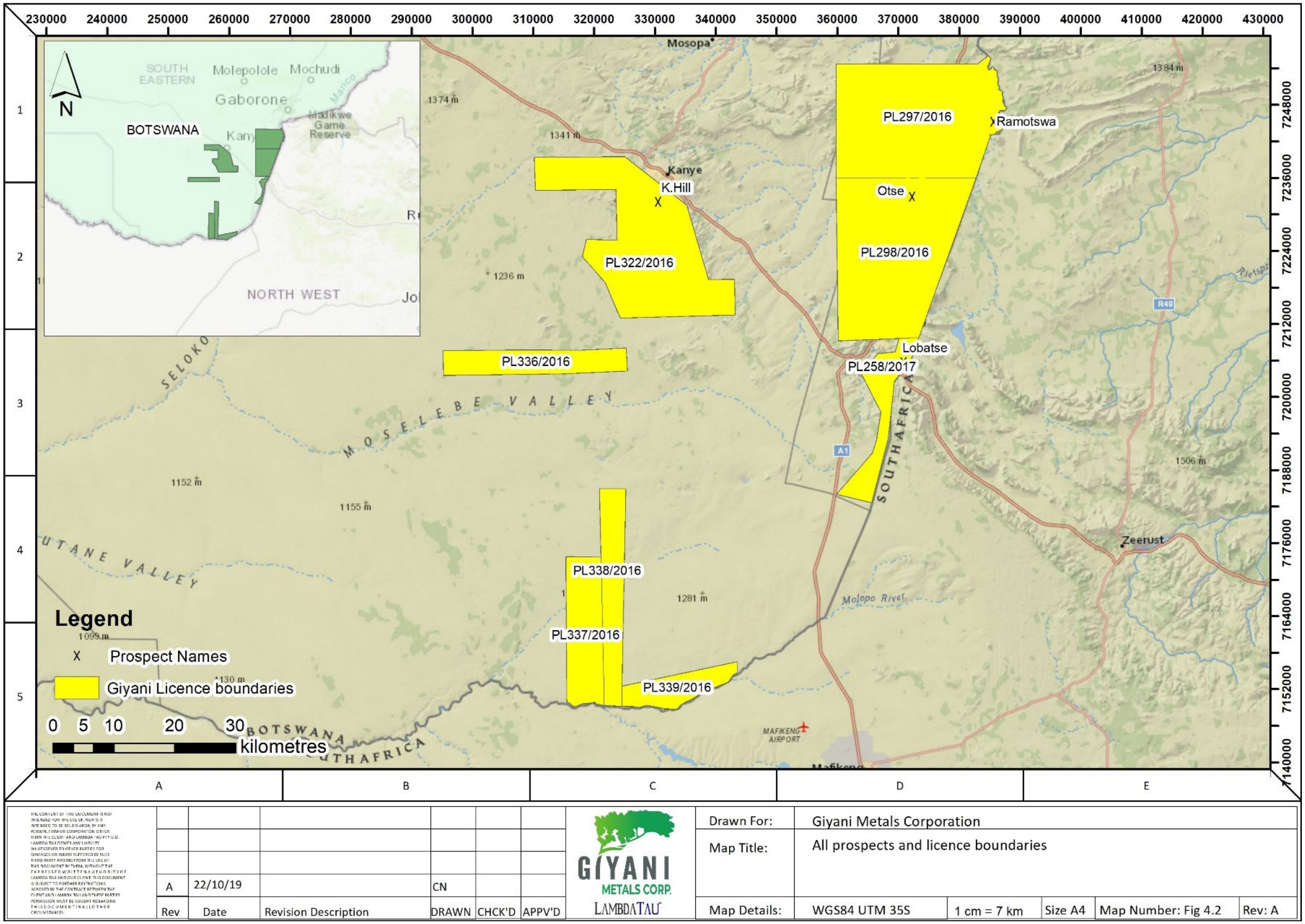
GIYANI METALS CORP.

LAMBDATAU

Drawn For:	Giyani Metals Corporation
Map Title:	Location Map of K.Hill licence boundary
Map Details:	WGS84 UTM 35S · 1 cm = 2 km · Size A4 · Map Number: Fig 4.1 · Rev: A

Source: Lambda Tau (2022)

Figure 4.2 Location of Giyani's prospecting licence holdings in Botswana



Source: Lambda Tau (2022)

Table 4.1 Summary of Giyani's prospecting licence holdings in Botswana

Licence number	Licence holder	Issue date	Licence type	Expiry date	Size (km^2)
PL297/2016	Menzi Battery Metals	01-July-2020	Prospecting	30-September-2024	482.9
PL298/2016	Menzi Battery Metals	01-July-2020	Prospecting	30-September-2024	478.4
PL322/2016	Menzi Battery Metals	01-July-2020	Prospecting	30-September-2024	437.7
PL336/2016	Menzi Battery Metals	01-July-2020	Prospecting	30-September-2024	118.1
PL337/2016	Menzi Battery Metals	01-July-2020	Prospecting	30-September-2024	144.1
PL338/2016	Menzi Battery Metals	01-July-2020	Prospecting	30-September-2024	127.1
PL339/2016	Menzi Battery Metals	01-July-2020	Prospecting	30-September-2024	76.8
PL258/2017	Menzi Battery Metals	01-January-2021	Prospecting	31-December-2024	95.0
Total					1,960.1

4.3 Environmental liabilities

In August 2018, Giyani contracted the services of Loci to undertake environmental screening studies for its projects in Botswana, including the K.Hill Project. No significant issues were identified during this process. Giyani received a request from the Botswana DEA to complete an EMP for the K.Hill prospect area. Under this EMP, Giyani has clearance to conduct exploration and evaluation work, including, but not limited to, geophysics and other non-invasive exploration techniques, drilling, and sampling. The approval of the EMP for the K.Hill and Otse prospect areas was granted in July 2019 and was valid until 2021. The EMPs were renewed for one more year in 2021 (valid until July 2022). In September 2022, the DEA approved the EMPs for two additional years (expiring September 2024). Giyani is required to complete a detailed EIA before any mining (which, by definition, includes construction) and/or processing can commence. Please see Section 20 for further details on the EIA.

4.4 Property obligations and agreements

According to Section 70 of the Mines and Minerals Act of the Republic of Botswana, the Licence Holder, at the time of issue of this licence and on each anniversary thereafter, is required to pay to the Office of the Director of the Department of Mines an annual charge equal to P5.00, multiplied by the number of square kilometres in the licence area, subject to a minimum annual charge of P1,000. Menzi Battery Metals has fulfilled all obligations on licence expenditure. Giyani, through Menzi Battery Metals, is expected to carry out, and has carried out, the prospecting operations set out in PL322/2016 (Table 4.2).

Table 4.2 Programme of prospecting operations for K.Hill PL322/2016

Programme of prospecting operations	Proposed minimum expenditure (P)	Proposed minimum expenditure (US$)
Year 1: • brownfields exploration • Mineral Resource expansion drilling • completion of Feasibility Study • post-Feasibility Study engineering • various economic and financial analysis	2,000,000	159,617
Year 2: • mining and mineral processing optimisation • decision to mine • addition to existing Mineral Resource base	2,500,000	199,521

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Accessibility

The Project area is well networked via paved national roads, namely the A1 and A2 highways. The K.Hill Project is located within a few kilometres of the A2 highway, which runs from Buitepos at the Namibian border; through Jwaneng, Kanye, and Lobatse; to the South African border at Pioneer Gate, near Zeerust, South Africa. The A2 is a major component of the Trans-Kalahari Corridor, which is a highway corridor that provides a direct route from Maputo in Mozambique via Pretoria to central Namibia, to Windhoek and the port of Walvis Bay. The A1 runs from Gaborone (the capital of Botswana) past the junction with the A2. The Sir Seretse Khama International Airport in Gaborone provides links to major cities in southern and central Africa.

The K.Hill Project site perimeter is accessible from the national road via paved public roads. Access on the site is mostly unpaved old mining tracks that have been opened for exploration access. Some areas remain overgrown and degraded by erosion.

Local communities own the surface areas around the K.Hill Project, and access is granted by notification only. The larger project area consists of a combination of privately-owned land and communal/tribal land.

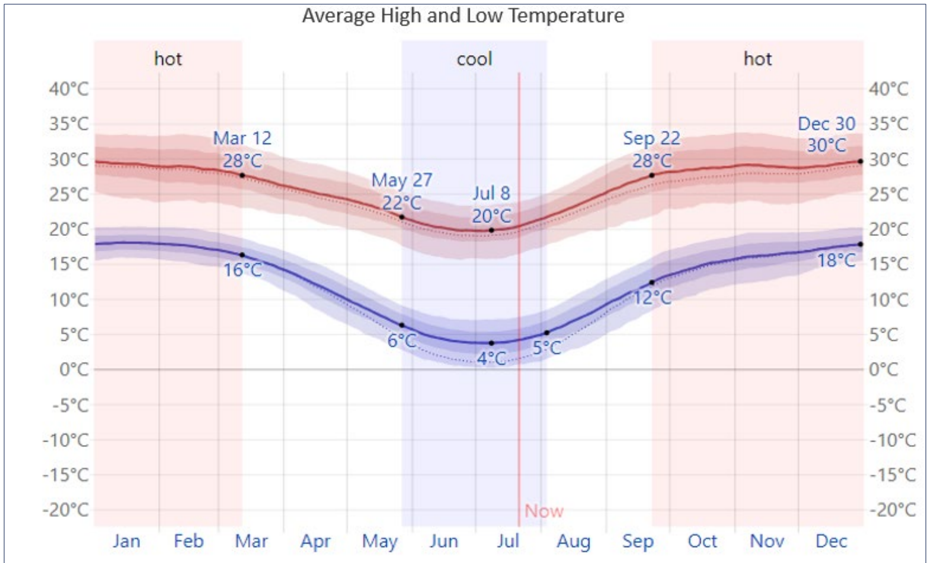
5.2 Climate and physiography

5.2.1 Climate

In Kanye, the summers are long, warm, and partly cloudy and the winters are short, cold, dry, and clear. Over the course of the year, the temperature typically varies from 4°C to 30°C and is rarely below 0°C or above 34°C.

The hot season lasts for close to six months, from 22nd September to 12th March, with an average daily high temperature above 28°C (82°F). The hottest month of the year is January, with an average high of 29°C (85°F) and low of 18°C (64°F). The cool season lasts for just over 2 months, from 27th May to 3rd August, with an average daily high temperature below 22°C (71°F). The coldest month of the year is July, with an average low of 4°C (39°F) and high of 20°C (69°F; Figure 5.1).

Figure 5.1 Annual average high and low temperature in Kanye

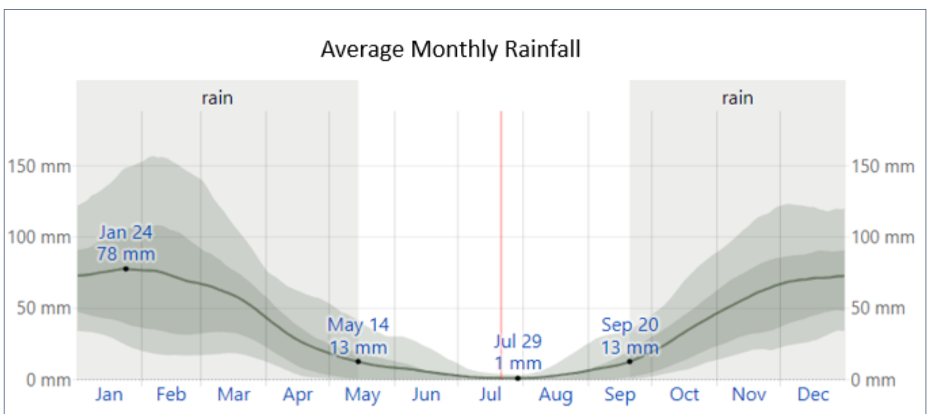


The daily average high (red line) and low (blue line) temperature, with 25th to 75th and 10th to 90th percentile bands. The thin dotted lines are the corresponding average perceived temperatures

Source: www.weatherspark.com

Kanye experiences significant seasonal variation in monthly rainfall. The rainy period of the year lasts for almost 8 months, from 20th September to 14th May, with a 31-day average rainfall of at least 31 mm (0.5 in). The month with the most rain is January, with an average rainfall of 73 mm (3.0 in). The rainless period of the year lasts for 4 months, from 14th May to 20th September. The month with the least rain in Kanye is July, with an average rainfall of 1 mm (0.0 in; Figure 5.2).

Figure 5.2 Average rainfall variation at Kanye



The average rainfall (solid line) accumulated over the course of a sliding 31-day period centered on the day in question, with 25th to 75th and 10th to 90th percentile bands. The thin dotted line is the corresponding average snowfall.

Source: www.weatherspark.com

The climate is generally considered to be warm and arid with a summer rainfall season. Operations can continue throughout the year.

5.2.2 Physiography

The land surface of Botswana is mostly flat or gently undulating, with the greatest topographical relief located in the southern parts of the country. Kanye is at an elevation of approximately 1,300 masl, and Kgwakgwe Hill (after which the Project name is derived) forms a distinct topographical feature next to Kanye (Figure 5.3 and Figure 5.4, with elevations reaching approximately1,500 masl.

Various waste dumps, discards, stockpiled manganese mineralisation, and tailings occur in the K.Hill Project area, which together with cuts into the steep hillside from historical open pit mining, have disrupted the natural topography (Figure 5.5).

Generally, Botswana can be divided into three main physiographic regions:

- the Wetland region around the Okavango Delta to the north
- the Hardeveld region, with outcropping metamorphic geology in the southeast, in which the K.Hill Project area lies
- the Sandveld region, which comprises the central Kalahari sands

Most of southern Botswana is covered in some form of savanna. In the K.Hill Project area, common shrubs and small thorn trees are present. No protected or scarce trees, such as the Baobab, Marula, Mopane or Fig trees, have been observed (c).

Figure 5.3 View from K.Hill toward Kanye, taken from a drilling site

Source: MSA Group (2018)

Figure 5.4 View of K.Hill showing historical spoil heaps

Source: MSA Group (2018)

Figure 5.5 Birds-eye view of the K.Hill Project area

Source: Giyani (2018)

5.3 Local resources and infrastructure

The K.Hill Project is located outside of Kanye, which has electricity supplied from the national grid. Kanye's water is supplied from the Mmamokhasi Dam, which is approximately 5 km from the Project site. Water from this source has been used for drilling activities.

Formal mining in Botswana for copper (Cu), nickel (Ni), coal, gold (Au), and diamonds has taken place from the 1900s to present, as well as historical manganese mining in the Kanye area. Both underground and open pit mining skills are available in the country as well as skills gained from migrant labour in neighbouring South Africa. Kanye is located 70 km from Jwaneng Mine, a large open pit diamond mine. A historical tailings area is located on the Project site (Figure 5.5).

6 HISTORY

6.1 Project and exploration history

The property ownership and previous exploration results are unknown for the K.Hill Project. The only available historical data are summarised in the following sections.

6.2 Historical production

The discovery of manganese in the Kanye area led to mining from 1957 to 1971 (Aldiss 1989). The first of many companies operating at K.Hill was Marble Lime, which also developed the asbestos mine at Moshaneng, northwest of Kgwakgwe Hill. Marble Lime mostly mined the bedded-type mineralisation (described as Mn-shale during the Phase 1 mapping exercise completed by Giyani), which required beneficiation before it could be saleable (Aldiss 1989). Marble Lime ceased mining activities around 1967.

Further exploration work was carried out by JCI, during the late 1960s and early 1970s.

In 1981, Rand London Manganese investigated the possibility of mining the Mn-shale deposits at K.Hill, Otse, and Gopane (near Lobatse), together feeding a single processing plant at Lobatse. The K.Hill deposit was considered to require further drilling for evaluation, but no further work was completed, and the licence was relinquished within a year (Aldiss 1989).

Historical production of manganese from K.Hill was reported by Baldock et al. (1977) to be 64,180 t from 1957 to 1967 and 131,563 t from 1968 to 1972 (Table 6.1). Variable prices were received for the product due to both metallurgical and high-grade battery-active products being supplied, with the latter attracting a premium at the time (Baldock et al. 1977). It is not stated how much of this material was mined from open pit or underground sources, and these figures have not been verified by the QP.

Table 6.1 Manganese production from the K.Hill Project

Year	Amount (t)	Value (R)
Total 1957-1967	64,180	798,678
1968	39,751	16,863
1969	16,732	290,433
1970	40,488	695,396
1971	34,387	140,655
1972	205	9,970
Total 1968-1972	131,563	-
Grand total	195,743	-

Notes:
R - South African Rand
Source:
After Baldock et al. (1977)

6.3 Previous Mineral Resource estimates

JCI estimated that about 120,000 t of "marketable ore" could be extracted from the shale horizon at K.Hill (1960s and early 1970s). Later, Rand Mines estimated that approximately 1.6 Mt, of which 60% could be extracted, resides within the shale-type "ore" at K.Hill (Aldiss 1989). The reader is cautioned that these figures are provided for historical background purposes, and no conclusions as to the prospectively of the Project should be drawn from them. No details are known to the QP on the basis of the estimates, and the level of information available is not consistent with the definition of a "historical estimate," as defined in Section 2.4 of NI 43-101.

Historical estimates reported according to NI 43-101 are summarised further:

- In 2018, MSA Group completed a MRE for the K.Hill Project. Table 6.2 summarises the results.

Table 6.2 K.Hill Mineral Resource reported at an 18% manganese oxide cut-off (2018)

Category	Tonnes (Mt)	MnO (%)	Al_2O_3 (%)	SiO_2 (%)	Fe_2O_3 (%)	LOI (%)
Inferred Mineral Resources	1.1	31.2	8.9	26.3	16.9	8.8

Source: MSA Group (2018)

- In 2021, SRK completed an MRE for the K.Hill Project. Table 6.3 summarises the results.

Table 6.3 K.Hill Mineral Resource reported at a 7.3% manganese oxide cut-off (2021)

Category	Tonnes (Mt)	MnO (%)	Al_2O_3 (%)	SiO_2 (%)	Fe_2O_3 (%)	LOI (%)
Inferred Mineral Resources	1.6	22.0	10.9	35.7	16.5	7.9
Indicated Mineral Resources	1.4	13.9	9.6	51.4	13.1	6.3

Source: SRK (2021)

- In 2022, SRK completed an updated MRE for the K.Hill Project as part of an NI 43-101 Feasibility Study Technical Report. Table 6.4 summarises the results.

Table 6.4 K.Hill Mineral Resource reported at a 7.3% manganese oxide cut-off (2022)

Category	Tonnes (Mt)	MnO (%)	Al_2O_3 (%)	SiO_2 (%)	Fe_2O_3 (%)	LOI (%)
Inferred Mineral Resources	2.1	19.3	10.56	41.17	16.66	7.37
Indicated Mineral Resources	3.1	16.9	7.6	14.70	22.50	7.46

Source: SRK (2022c)

7 GEOLOGICAL SETTING AND MINERALISATION

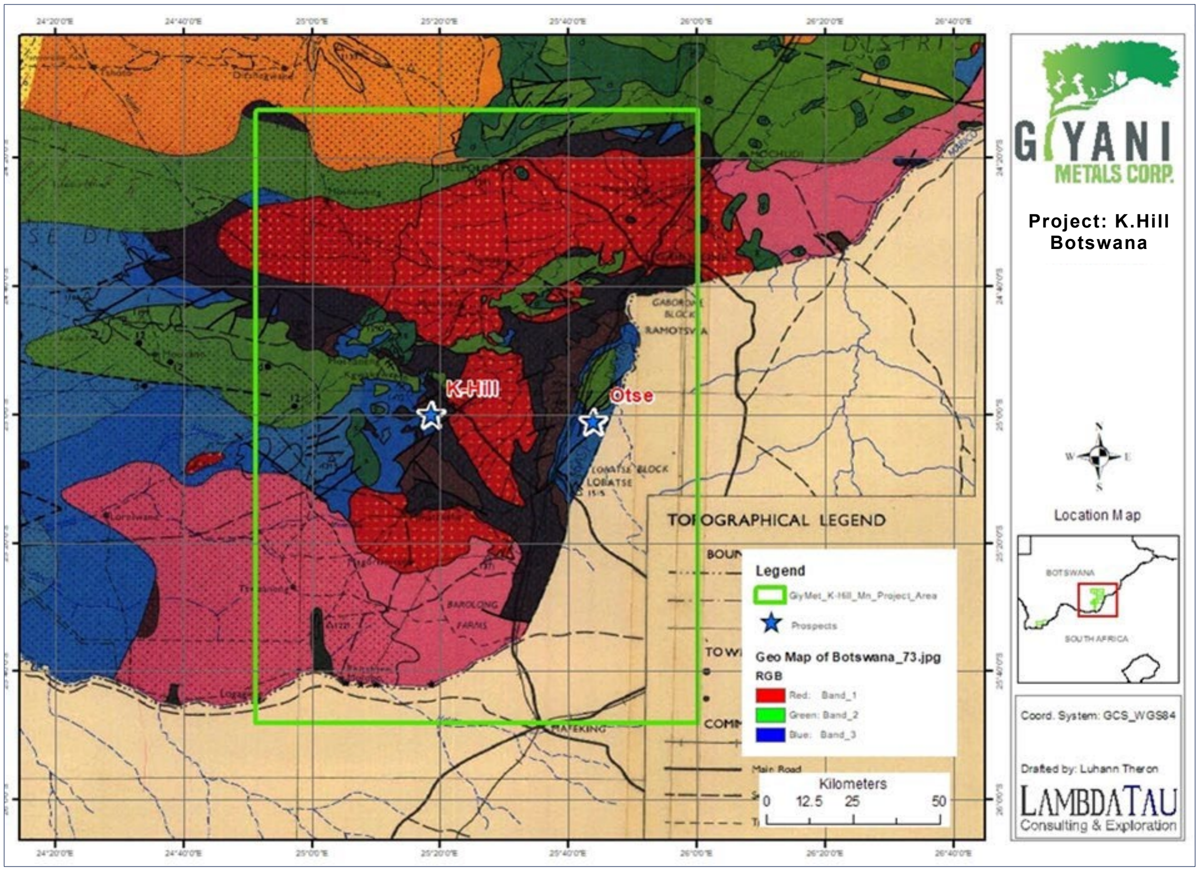
7.1 Regional geology

The stratigraphy in the K.Hill Project 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 location of the K.Hill prospect is shown superimposed on the 1973 regional geological map of Botswana in Figure 7.1. The Archean basement in southeast Botswana is well studied on a regional scale, and maps are generally accurate. The prospect occurs within the mapped Transvaal Supergroup sediments, consisting of shales, quartzites, limestones, and conglomerates (blue shading in Figure 7.1) and in the vicinity of the Kanye Group (part of the Ventersdorp Supergroup), consisting of a variety of extrusive lavas and subordinate siltstones and shales (purple shading in Figure 7.1).

To the east, the early-Precambrian intrusive igneous units of the Gaborone Granite suite occur (red and pink shading in Figure 7.1). The youngest succession in the K.Hill Project area is the Waterberg Group sediments (green shading in Figure 7.1). In 1998, Roger M. Key and Neil Ayres published a subsequent regional (1:1,000,000 scale) geological map of Botswana; however, most of the government regional mapping in the K.Hill Project area was completed prior to 1980, so for this area, the 1973 map is still the most accurate and relevant (Key and Ayres 2000).

The Moshaneng-Kanye area contains outcrops of a Transvaal-age sedimentary sequence, which forms part of the so-called Kanye Basin. This basin is oval shaped, with its long axis orientated northwest-southeast and extends over the Kgomodikae, Segwagwa, and Moshaneng areas. It is separated from the Transvaal sediments in the Lobatse area by Ventersdorp-age rocks, which comprise the Lobatse Volcanic Group, remnants of Archaean basement rocks and the Gaborone Granite, all of which define the north-south trending Vryburg Arch between Kanye and Lobatse. At Kanye, the arch begins to swing westward through Moshaneng toward Jwaneng. Although partially separated by a veneer of Waterberg-age rocks, the Transvaal Supergroup sequence around Moshaneng forms a structural continuum with that exposed around Kanye, as evidenced by the isolated ridges of chert breccia between the two. The Transvaal sediments outcrop in a poorly defined ring structure near the village of Moshaneng. The core of this structure is composed of alkaline igneous rocks of the Moshaneng Complex.

Figure 7.1 Location of the K.Hill prospect on the regional geological map of Botswana, 1973



Source: Lambda Tau (2021)

In the Kanye area, only the lower parts of the Transvaal succession, the Black Reef Quartzite Formation and the Taupone Dolomite Group, are present. The sediments of the Black Reef Quartzite Formation constitute a sequence that fines upwards from conglomerates at the base through arenites to shales and mudstones at the top. This suggests a progressive increase in water depth consistent with a major marine transgression. The discontinuous clast-supported conglomerates are thought to have been deposited in localised stream channels within an active tidal beach environment. This conclusion is supported by the occurrence of very mature quartz arenites overlying the conglomerates.

The Black Reef Quartzite Formation passes gradationally through an interval of bluish-grey shales and mudstones into the overlying carbonates. The carbonates are interpreted as being predominantly deposited on a tidal flat. Local sandstone lenses may occur in the upper portions of the dolomite sequence and mark the start of a regressive cycle in which terrigenous sediment was reintroduced.

The predominance of the Paupone Dolomite Group and the extensive development of chert breccias, as opposed to banded iron formation, suggests that the deposition of the Transvaal Supergroup rocks took place in a tectonically active basin. Tectonic instability is also evident during the deposition of the Black Reef Quartzite Formation, as indicated by the development of localised fault-scarp conglomerates as its base.

7.2 Local stratigraphy

7.2.1 Summary

The mineralisation at the K.Hill Project 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. The Kgwakgwe Chert Breccia Formation in the Kanye area can be subdivided into two main varieties:

- a dark-brown chert breccia with milk-white angular chert fragments, cemented together by brown haematitic material
- a reddish-brown chert breccia with abundant jaspilitic fragments and a high content of jasper in the matrix

Figure 7.2 Chert breccia at the K.Hill Project

Source: MSA Group (2018)

7.2.2 Lithologies

Six lithologies were consistently intersected during drilling operations at the K.Hill Project. These include a chert or chert dolomite breccia unit, which occurs at the top of the stratigraphic sequence. Below that is an approximately 50 m thick package of shale identified to be part of the Black Reef Quartzite Formation, which hosts the manganese mineralisation within the shale units. Below this shale is an iron-rich shale, which is often intruded by manganese oxide material along fractures and joints. Below the Fe-shale lies a lower iron shale unit, typically a beige colour with no significant manganese content. A shale-containing manganese clay is also observed within the beige shale. Between the overlying shales and the felsite footwall unit lies a conglomerate marker unit, observed in almost all the drillholes. In some holes this marker conglomerate has a gritty texture with small clasts and is also mineralised with manganese oxides.

7.2.3 Chert dolomite breccia

This unit consists of angular chert (dolomite) clasts within a haematite-rich matrix. The chert dolomite breccia (CDB) unit decreases in grain size and abundance vertically. In addition, the unit becomes laminated before transitioning into the underlying and softer manganiferous-shale (Mn-shale) unit. The contact is typically broken and is assumed to be sharp and erosional. Minor overburden of typically less than 1 m in thickness is intersected at the top of the sequence.

7.2.4 Manganiferous-shale

The Mn-shale (MSH) lithology is the primary mineralisation host. The lithology consists of massive and homogenous manganese oxide mineralisation, with a steel grey, submetallic lustre. The horizons are generally associated with higher magnetic susceptibility readings when compared to adjacent shale and CDB units, especially when manganese oxides are observed visually. The MSH lithology has a dark black, friable shale/clay component, which makes up the bulk of the unit. These dark black sections are thought to contain manganese wad, a high-grade manganese dioxide complex. Laminae of kaolin clay are also observed. The MSH horizons are typically 3 m to 4 m thick. Core recovery can be poor due to the friable nature of these rocks.

7.2.5 Iron-shale

The Fe-shale (FSH) lithology, being rich in iron, is found above and below the MSH lithology and is, generally, the most extensive lithology in the K.Hill area. The unit hosts sporadic, vein and fissure fill mineralisation of manganese oxide and can also occur as thin 1 cm to 3 cm thick bands. Mineralisation is confined to the upper sections of the unit and grades into alternating weak to moderate non-mineralised Fe-shales downhole. Haematite and bleaching alteration often occur.

7.2.6 Beige-cream shale

The beige-cream shale (SHL) unit is a non-mineralised, homogenous unit, occurring below the FSH units. The SHL units have gradual contacts and are interspersed between a manganese clay and/or conglomerate unit.

7.2.7 Manganiferous clay

The manganiferous clay (Mn-clay or MCLAY) unit contains low-level mineralisation present in the form of laminations or pervasive blotches. Moderate limonite alteration is typical of this unit.

7.2.8 Conglomerate

The conglomerate (CONG) unit is the primary marker of the footwall. This unit appears to be weakly mineralised in the south of the K.Hill Project region. The CONG units within the north have a coarser grain size than the units in the south. The unit is characterised by a granular/sugary texture, with subhedral clasts of quartz. This unit is often interspersed by 10 cm bands of cream shale, with sharp upper and lower contacts.

7.2.9 Felsite

This felsite (FEL) unit is a fine-grained, silica-rich igneous extrusive rock forming the footwall to the shale and conglomerate units. The exact composition of the FEL unit is unknown and is referred to as felsite. The unit is not mineralised and contains pervasive-blotchy haematite alteration and localised-pervasive limonite alteration.

7.3 Property geology

The Mn-shale horizon extends at surface in outcrop in the north and to the south below surface, where its presence has been intersected in drillholes. To date, the known strike length of the horizon is 1.9 km.

The stratigraphy has been duplicated by thrusting in places, and, to the south, the mineralised horizon has been extensively downfaulted by steeply dipping east-north-northeast trending faults. The shales have been intensely folded and slumped in the vicinity of these dislocations, and in addition, subparallel breccia zones and quartz veining may be evident.

The entire Transvaal package is cut out against the Waterberg sediments to the west of Kgwakgwe Hill along what are thought to be northerly trending faults.

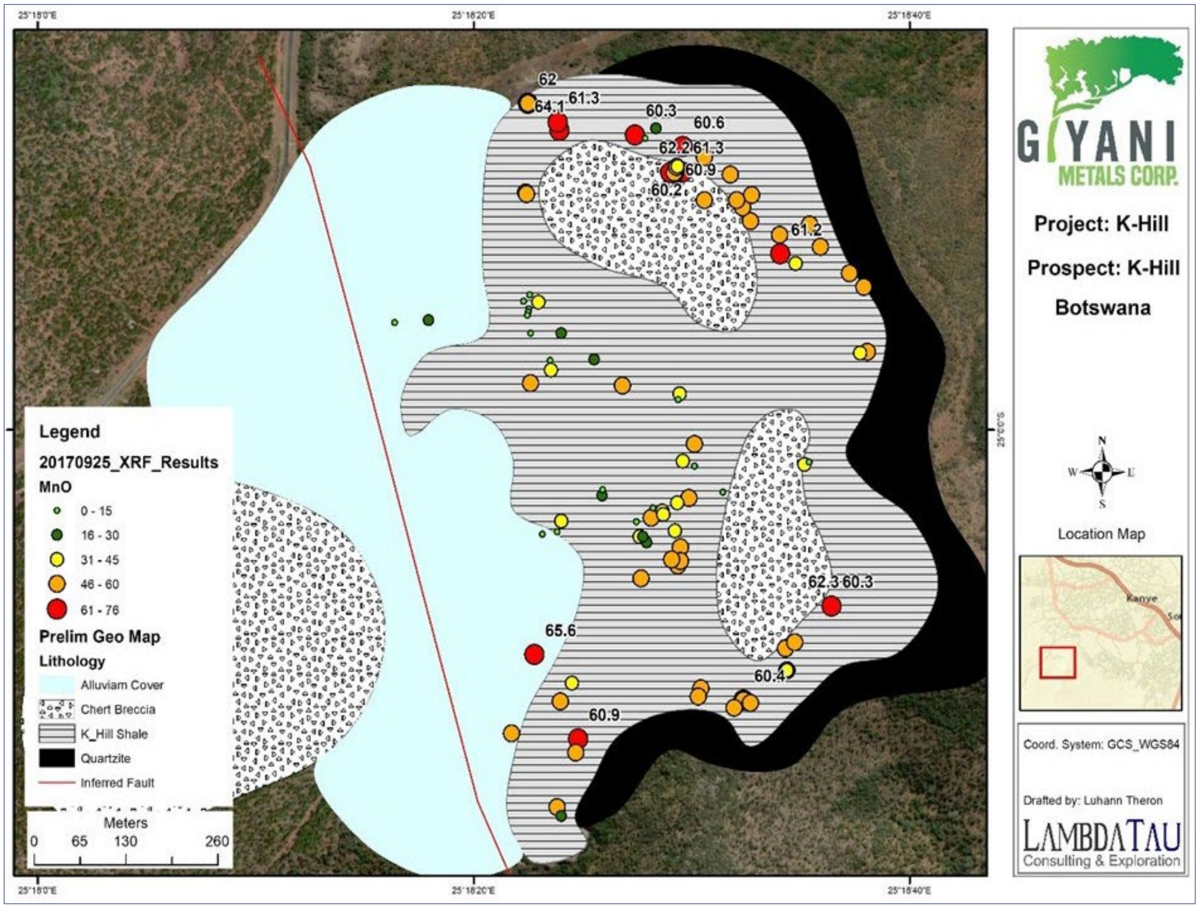
The Mn-shale outcrops along the northerly scarp slope of the Kgwakgwe Hill and dips into the hill (Figure 7.3). The strata at the K.Hill Project dip gently toward the northwest, at an average of approximately 5° to 10°, and S and Z parasitic folding is common. Numerous outcrops display parasitic folding, with local dips varying from 45° to almost subvertical. This is not consistent with the overall shallow dip of the Mn-shale and adjacent units. Where the deposit outcrops in the north, the Mn-shale unit is kidney shaped. The unit varies between approximately 2 m to 15 m thick, with an average thickness in the order of 4 m and has a delineated extent of approximately 1,900 m north-south and 350 m east-west. Some of the thicker intersections may be local fold duplications of a single horizon. In general, the southern extension area shows greater thicknesses of the mineralisation than in the north.

A simplified geological map of the mineralised area, including the positions of surface grab samples of manganiferous material, is shown in Figure 7.4. The outcrop defining the limit of the Mn-shale is well defined in the east, with the areas to the west being partially covered by alluvium and/or exposed by historical surface mining into the hillside. Little or no outcrop is observed in the areas south of northing 7,233,525. In this southern area, the extents of mineralisation are defined by drilling only.

Figure 7.3 Manganiferous-shale (black unit) and overlying shale (pale brown unit) exposed at the entrance to artisanal workings

Source: MSA Group (2018)

Figure 7.4 Simplified geological surface map of the K.Hill Project area



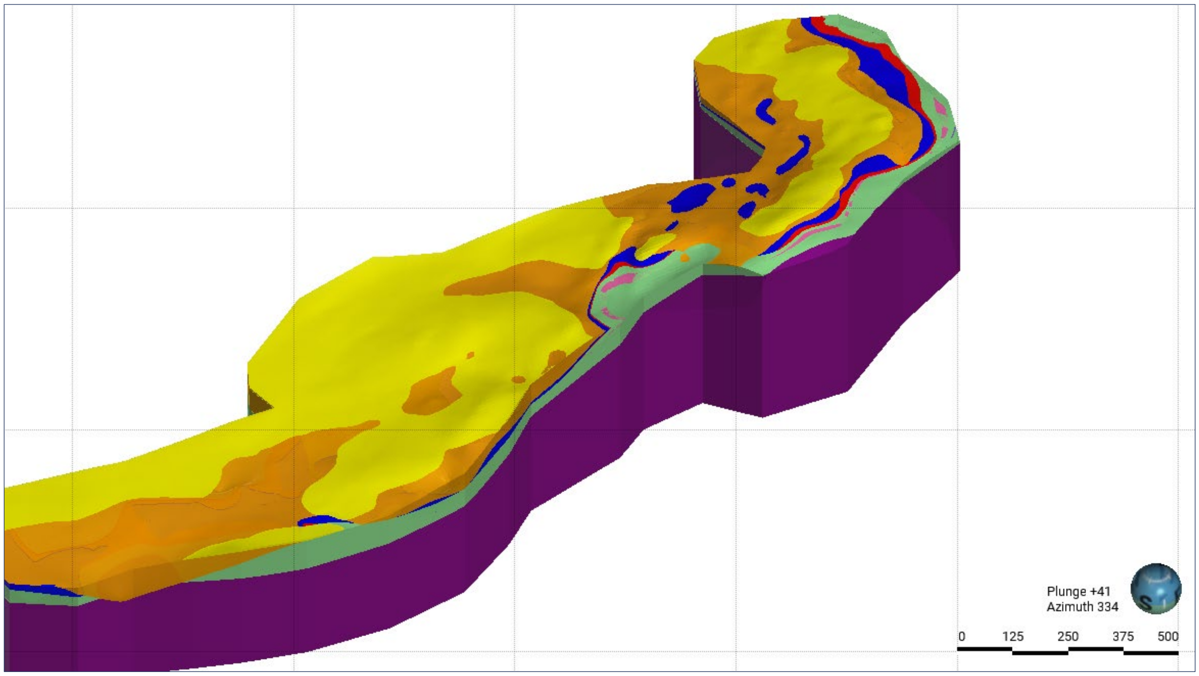
Source: Lambda Tau (2017)

Figure 7.5 presents an oblique view of the geological interpretation for the northern area, with a typical simplified cross section provided in Figure 7.6, showing the main Mn-shale horizons and how they have been coded for the purposes of modelling the Mineral Resource. This interpretation represents the shale horizons as broadly stratiform, shallowly dipping units (5° to 10°), where the interpretation is based on the mapped mineralisation outcrops and drillhole intercepts.

The horizons have been coded from A to C, with the highest layer coded as A, central layer as B, and lowest layer as C. In addition, A Horizon has been subdivided into the A1, A2, and A3 horizons, reflecting changes in manganese content across the layer.

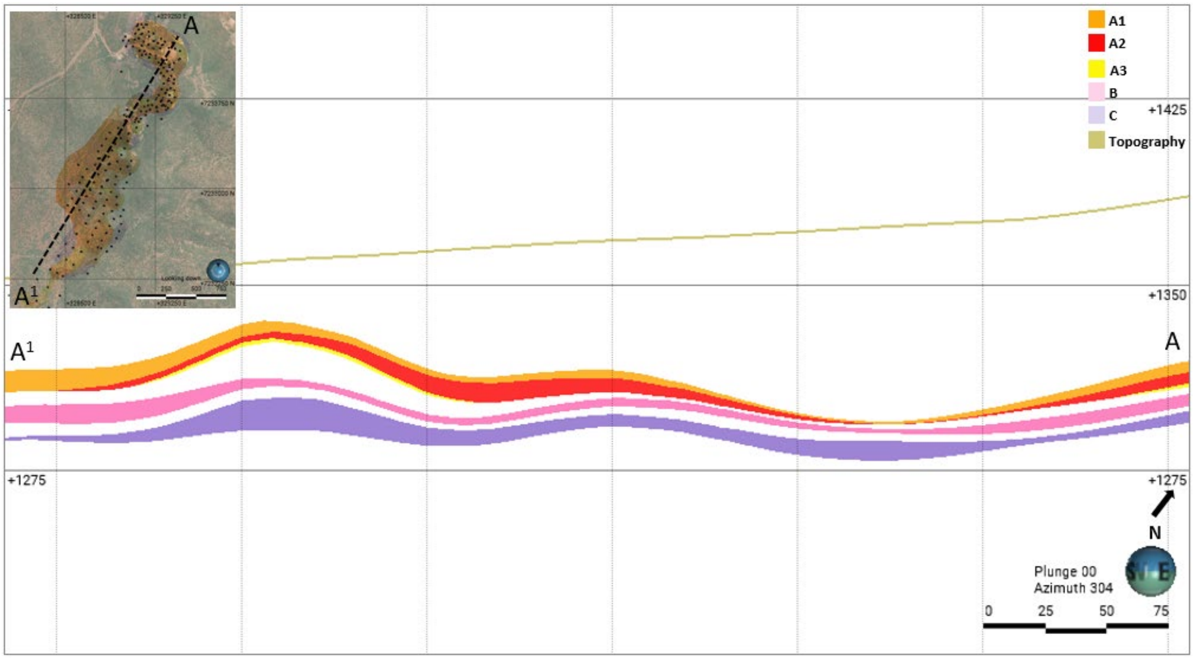
Based on previous field observations, it is possible that the Mn-shale horizon is disrupted and discontinuous due to thrusting, causing local folding and faulting. This alternative interpretation, which requires further investigation prior to incorporation in an updated model, is unlikely to materially impact the overall continuity of the individual horizons and, therefore, the global volumes of the horizons, but it may impact on the short-scale morphology and grade distribution of the horizons, and, in turn, may explain the variable thickness observed in the horizons.

Figure 7.5 Oblique view of the K.Hill geology model northern area (looking west)



Source: CSA Global (2023)

Figure 7.6 Cross section through the K.Hill manganiferous-shale horizons, plan view indicating the section line



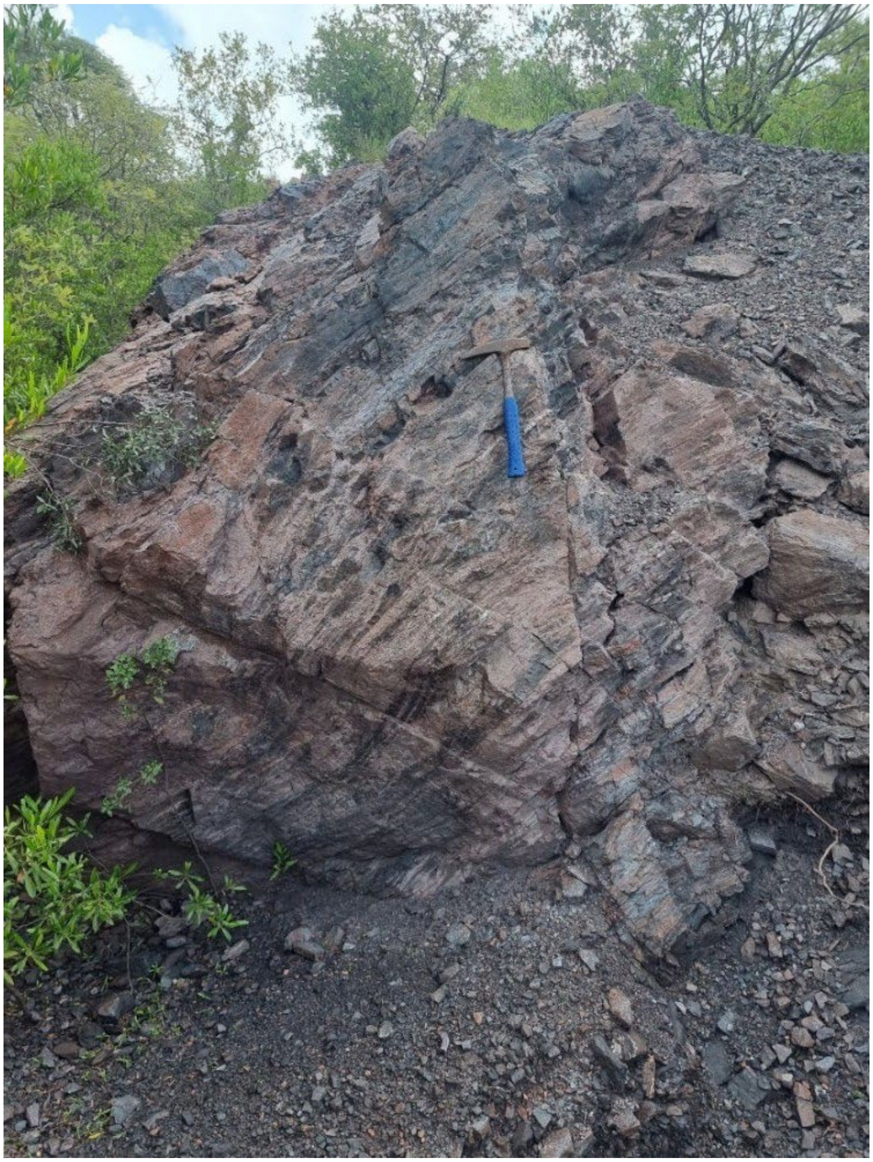
Source: CSA Global (2023)

7.4 Structural geology

The mineralised horizons have been interpreted as two packages: the Upper Mn-shale A Horizon and the Lower Mn-shale B Horizon, both of which dip shallowly (5° to 10°) toward the northwest. The current interpretation considers the horizons as continuous stratiform bodies based on outcrop (where available) and drillhole data.

Field observations of outcrops on the hill where the main deposit occurs show numerous examples of parasitic Z, S, and anticlinal folds. To have such folds and yet maintain such shallow uniform dips poses a problem with the spatial accommodation of the Mn-shales. The stratigraphy itself is Paleoproterozoic in age (Transvaal Supergroup, 1.2 Ga). Elsewhere in the region, stratigraphy of this age displays significant evidence of deformation. Most outcrops visited display evidence of widespread parasitic folding (Figure 7.7). Structural reading of exposed fold axial surfaces (hinges) shows strikes of between north-northwest to south-southeast to north-south, almost parallel to the overall strike of the beds with plunges of between 5° and 10°.

Figure 7.7 Outcrop of folded, steeply dipping, manganiferous-shale unit at the K.Hill Project



Source: SRK and Tetra Tech (2022)

Based on field observations, it appears the stratigraphy may have been duplicated by thrusting. This may well explain how the parasitic folding has been accommodated, and the overall dip of the stratigraphy remains shallow (5° to 10°) and dips to the west, whilst locally it can vary from 45° to almost vertical. The presence of a possible basal and upper thrust (now eroded) may have formed a series of detachment fold duplexes that has given rise to parasitic folding and associated faulting. Figure 7.8 shows an example of this interpretation in folded and faulted sediments from another location.

Figure 7.8 Example of localised folding and faulting similar to that seen at the K.Hill Project

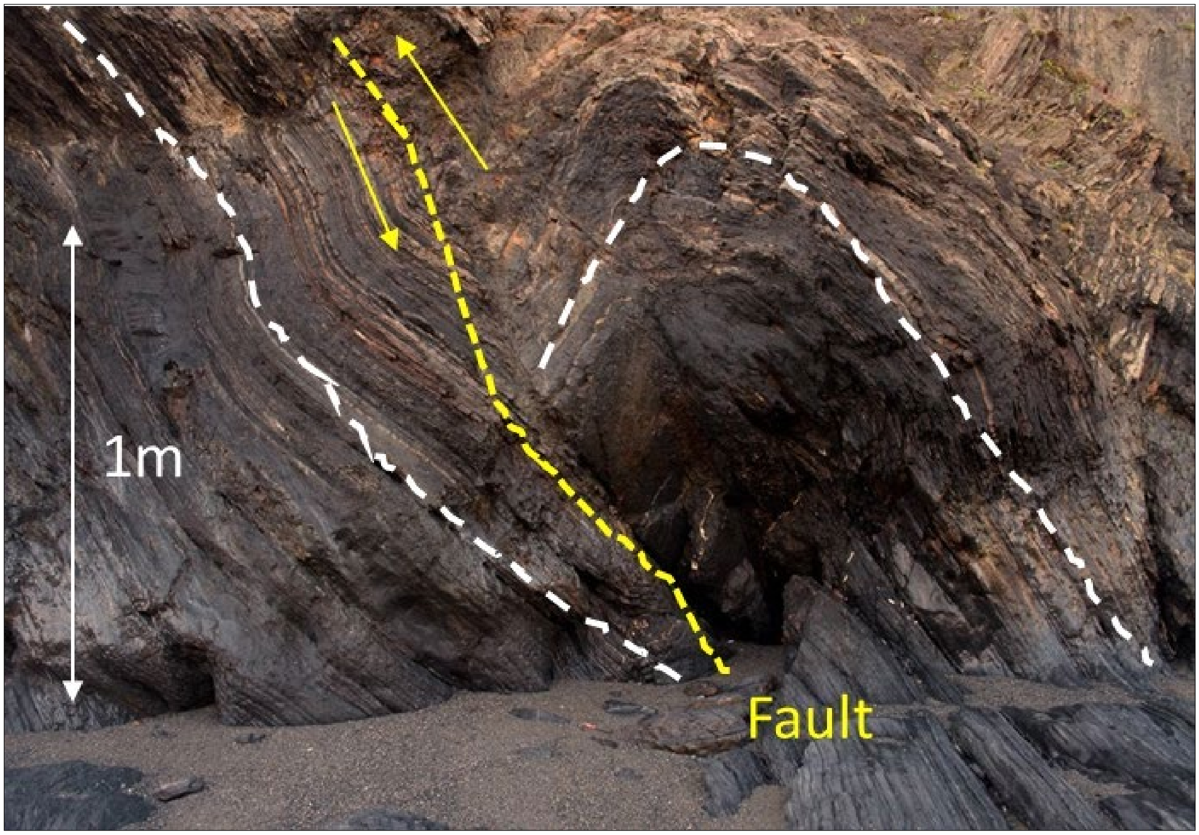
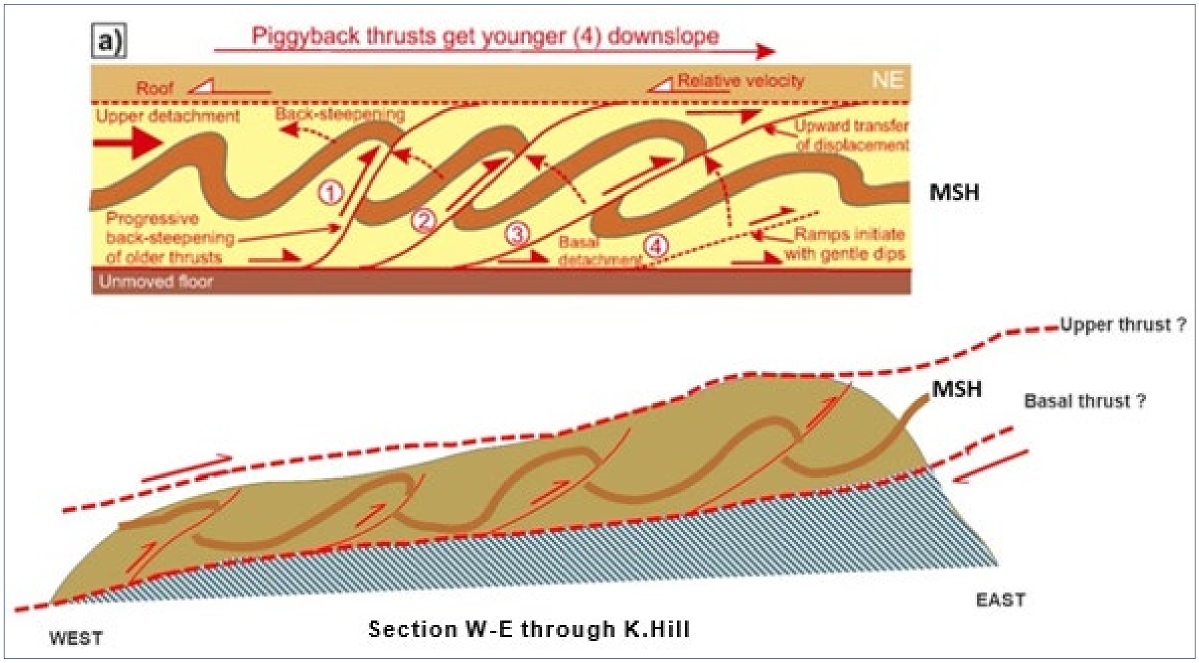


Figure 7.9 shows an idealised west-east cross section through the main K.Hill deposit at approximately northing 7,233,900, presenting the possible development of such structures caused by local thrusting in the footwall and hanging wall of the mineralised horizon. The development of parasitic fold duplexes caused by thrusting of the Mn-shale and associated stratigraphy may give rise to the development of localised lenses of thicker (and possibly higher grade) manganese mineralisation. It has also been noted that many of the historic workings do not follow the manganiferous mineralisation down dip to the west. It appears, based on site observations, that the historic adits may have preferential mined manganese mineralisation along fold hinges and trend in a more northerly direction.

An alternative to a sedimentary stratiform interpretation or the parasitic folding due to thrusting may be syn-sedimentary mass flow movement of material (sedimentary slumping). However, although this would account for the range of dips observed at outcrop, this is thought less likely, as no typical soft sediment deformation features have been noted (flame structures, clastic dykes, slumps).

Figure 7.9 Possible development of detachment fold duplexes associated with faulting at the K.Hill Project



Source: SRK and Tetra Tech (2022)

The implication for adopting a revised interpretation incorporating parasitic folding due to thrusting on mineralisation, resources, and mining at the K.Hill Project would be:

- less continuity of mineralisation down dip than is currently envisaged and with preferential thickening of localised lenses with a dominant north-south to north-northwest to south-southeast trend
- revision of the mineralisation domain models at a local/short scale (i.e., closer than current drill spacing), where globally the volumes and tonnes would be similar, but that there may be local-scale modifications to the geometry to reflect the revised interpretation and adjustments to the local-scale grade variability; this type of detail might only be seen in mined exposures
- this local-scale variability may require a greater degree of grade control definition drilling to accurately define the Mineral Resource at higher levels of confidence such as Measured
- in terms of mining, a more selective approach may be required to mine mineralisation that maybe less continuous at a local scale than originally anticipated

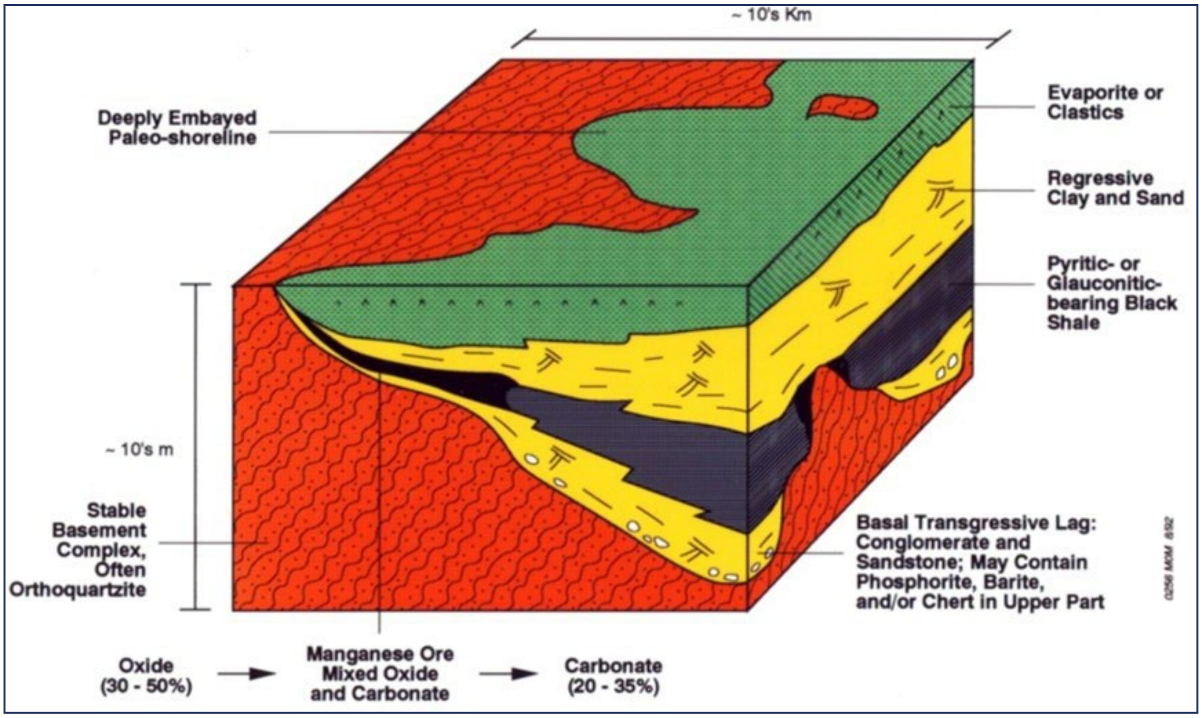
8 DEPOSIT TYPES

The manganese mineralisation at the K.Hill Project occurs as a supergene-enriched shale (Mn-shale) within the Black Reef Quartzite Formation.

The Mn-shale itself appears to represent a primary manganese deposition in a shallow marine basin, as per the Canon and Force (1986) model referred to in Figure 8.1. As is typical for most manganese deposits, there is clear evidence of upgrading by means of supergene enrichment. This evidence includes the following:

- Only manganese oxide mineralogy has been noted to date.
- Observed manganese mineral textures are consistent with secondary precipitation.
- The presence of fine manganese wad and the presence of cavities and vugs are intersected in the drillholes in the Mn-shale.

Figure 8.1 Cannon and Force (1986) model for sedimentary manganese mineralisation



Source: Cannon and Force (1986)

There are two possible time intervals during which weathering and supergene enrichment could have occurred: the recent period of exposure as well as the ancient period of exposure associated with the unconformity at the base of the chert breccia. If supergene enrichment is only related to the current exposure, then the enrichment could be limited in extent by the current geomorphology and may only extend under the hill to the limit of weathering. If there was supergene enrichment during the period associated with the unconformity, then more extensive supergene enrichment may have occurred. The latter possibility appears to be supported by the drillhole intersections. Two mineralising processes are evident, namely an initial phase of mineralisation by precipitation and diagenesis during sedimentation (forming the Mn-shale) followed by one (and possibly two) phases of redistribution and concentration during weathering and supergene enrichment.

Interpretation of the gravity map allows the recognition of a very clear marine basin embayment paleogeography of the type typically recognised in the Cannon and Force (1986) model. The Archaean floor to the basin seems to be well described by gravity anomalies. Interpretation of the gravity map indicates the possible presence of two shallow marine embayments with a north-south trending paleo-shoreline shoaling to the east and deepening westward.

8.1 Mineralisation style

Key elements of the Canon and Force (1986) model and this mineralisation style may be summarised as follows:

- The Cannon and Force (1986) model is a depositional model for many sedimentary manganese deposits that have their formation during times of high sea level stand and stratified sea columns in common. Manganese precipitation occurs at intersections of horizontal oxidation-reduction interfaces with shallow marine substrates within shallow marine embayments.
- The manganese occurs in typically thin, flat-lying stratiform and stratabound layer(s), often of enormous lateral extent. They are stratiform marine basin-margin deposits, which may be present in oxide and (or) carbonate facies, tend to be in condensed stratigraphic sequences, result from low-energy deposition, and have little clastic dilution. Characteristically, the manganese horizon is a thin but laterally extensive stratigraphic condensed sequence-type interval.
- Basin analysis has shown that these types of deposits typically occur in settings characterised by deposition in localised basins or shallow marine embayments around littoral paleo-islands, peninsulas, and shoals. It is important to delineate barrier island, embayments, and shoal settings because of the role of basin sills play in isolating anoxic seas and favouring black shale formations. The relationship to the basin's basement and floor is critical. Major deposits are usually formed close to the basin margin. Deposits are often less than 100 m above the basement.
- The host rocks are typically sandstone (or orthoquartzite), siltstone and claystone, shales, and black shale. Subordinate, poorly consolidated limey sediments (marls) occur. Diatomaceous clays, radiolarian sediments, and shell beds are common. Black organic- and pyrite-rich shales and glauconitic sands are common in footwall rocks.
- Mineralogically, they typically include an oxide facies, oxide-carbonate facies, and a carbonate facies. Currently there is only evidence of oxide facies in the K.Hill deposit:
  - The oxide-facies manganese mineralisation most commonly includes cryptomelane-group minerals and pyrolusite and over 40 oxide minerals. Less-oxic deposits contain manganite, braunite, and kutnohorite. Psilomelane and wad (primary and supergene iron and manganese oxide mineral intergrowths) are commonly listed in older literature. Gangue typically includes clay minerals (commonly montmorillonite), carbonate minerals, glauconite, quartz, chert, and biogenic silica.
  - If developed, the oxide-carbonate facies include psilomelane, manganite, manganoan calcite, and rhodochrosite.

    - Carbonate facies mineralisation typically includes rhodochrosite, kutnohorite, and manganoan calcite; siderite; mixed manganese and iron-carbonate minerals; pyrite; and wad. Gangue typically includes clay, calcium, and calcium-magnesium carbonate minerals; glauconite; organic matter; pyrite; quartz; and biogenic silica.

- Secondary superimposed weathering and supergene processes are common. In-situ weathering and oxidation enhance both the oxide and carbonate primary mineralisation. Manganese carbonates may weather to a brown non-descript rock. Black secondary oxides are common. Penecontemporaneous erosion, oxidation, and sedimentary reworking (tidal lag) of oxide and oxidised carbonate mineralisation can give rise to higher grades. Contacts between primary and supergene-enriched zones are typically sharp.

This mineralisation type typically presents manganese grades as follows:

- primary: 20% to 40% manganese, low iron
- secondary: 30% to 50% manganese, low iron
- iron content: host rocks are low in iron
- examples of this mineralisation type include:
    - Groote Eylandt, Australia
    - Nikopol & Bolshoi Tokmakskoe, Ukraine
    - Chiatura, Georgia
    - Obrochischte, Bulgaria

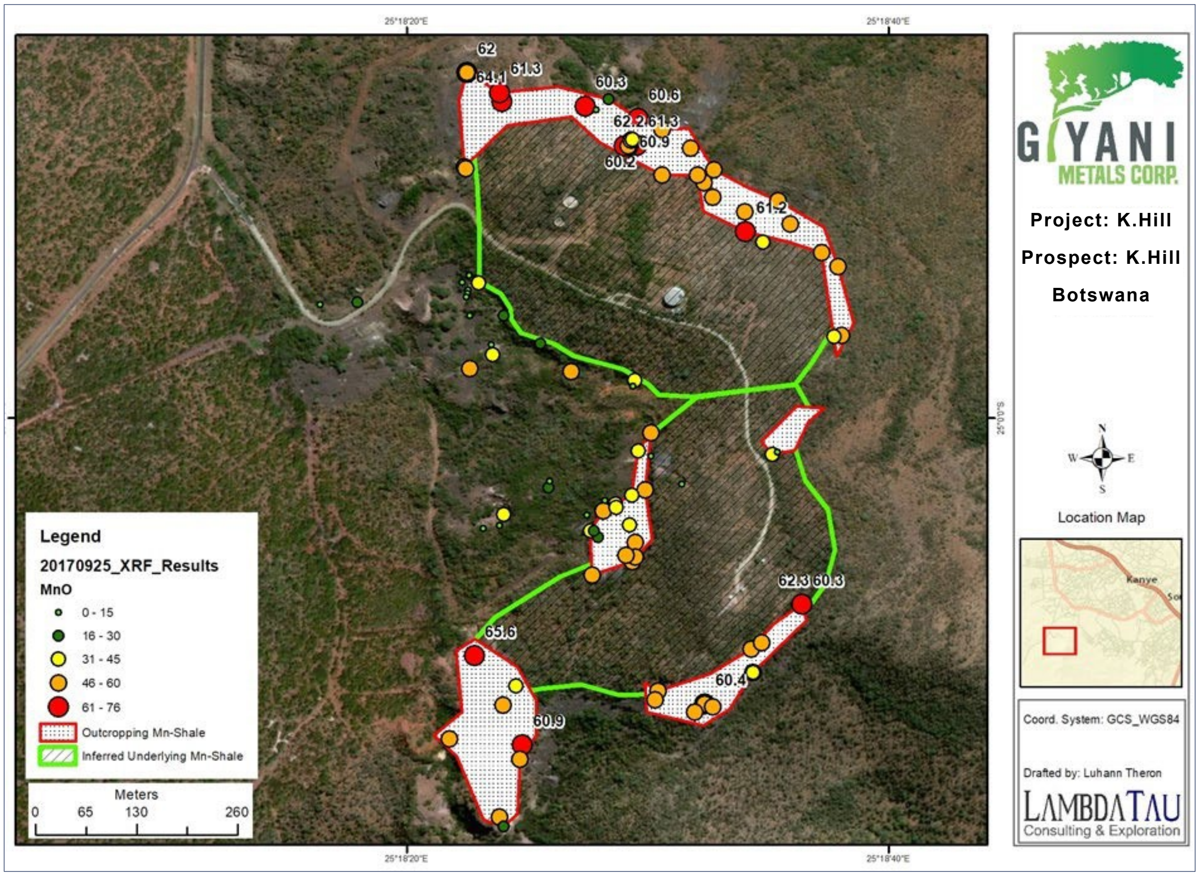
9 EXPLORATION

9.1 Mapping and sampling

Early exploration at the K.Hill Project was designed to geologically map and geochemically sample the prospect licence area. Geologists equipped with a Garmin GPSMAP 64S GPS, a Brunton Compro Pocket Transit compass, and RockLogger Android software, spent 13 days mapping and sampling in and around the main prospect area.

A preliminary stratigraphic column was interpreted from the field observations. It is important to note that the Mn-shale and the K.Hill shale are not the same unit. The K.Hill shale, as shown in Figure 9.1, refers to the entire shale unit. This unit varies in composition and character but represents the same geological event. The Mn-shale refers to a specific horizon within the K.Hill shale that contains elevated concentrations of manganese mineralisation.

Figure 9.1 Location of outcropping manganiferous-shale within the area of interpreted manganiferous-shale that underlies K.Hill



Source: Lambda Tau (2017)

Historical mining efforts focused on the central area, with unmined outcropping mineralisation at the outer northern and north-western edges. Adits from artisanal mining are common in areas containing outcrops.

Outcropping mineralisation, as well as exposed mineralisation in the historical mining pits, were sampled. Various mineralisation styles were observed and sampled. Grab samples were taken from the K.Hill shale within and away from the Mn-shale outcrop. These were taken where outcrops were present, and access allowed. A total of 97 grab samples were collected from the K.Hill prospect; additionally, 25 samples were submitted as duplicates to test variability in sampling technique and analyses. All samples were taken from surface and within old open pit faces except for two samples that were taken from artisanal adits. Sampling focused on mineralised units, although some samples were collected from the non-mineralised footwall and hanging wall units. Rock units sampled included Mn-shale, ferruginous shale, quartzite, chert breccia, and siliceous/silicified shale.

SGS Randfontein - Natural Resources (SGS Randfontein) in Randfontein, South Africa, analysed the grab samples using borate fusion followed by x-ray fluorescence (XRF).

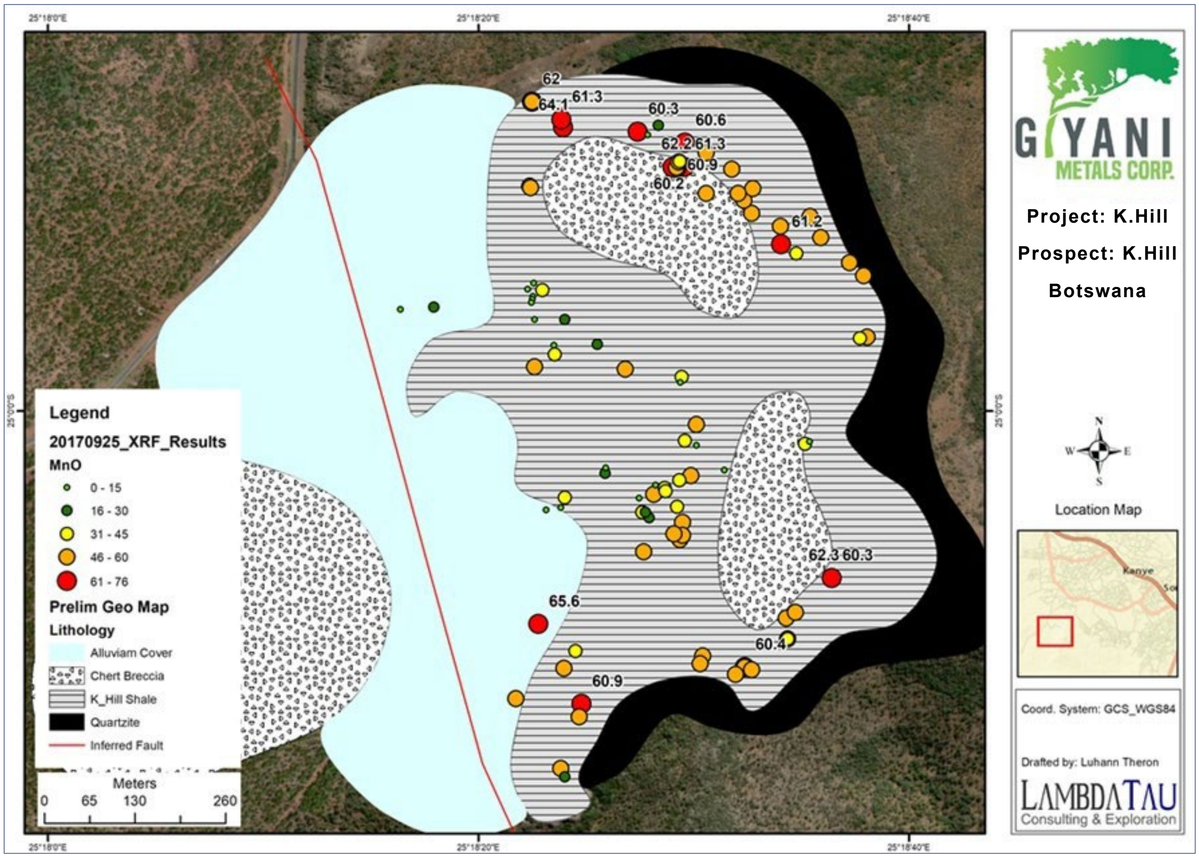
The purpose of grab sampling was to identify the location and nature of the mineralisation. Samples of this nature are not suitable for estimating the grade of the deposit, as they are not representative of the grade of the complete mineralised unit, and sampling of this nature is inherently biased, generally toward higher-grade mineralisation.

Samples collected from outcropping Mn-shale units yielded the highest manganese grade, as can be expected. Lower-grade and non-mineralised footwall and hanging wall units were sampled where they were exposed by historical workings. Instances of high-grade manganese oxide (>50%) occur along the entire approximate 1.25 km of Mn-shale outcrop. Due to the biased nature of the grab sampling, this should not be misconstrued as the in-situ grade of the Mn-shale unit but does illustrate that occurrences of high-grade manganese mineralisation occur within the outcrop.

A preliminary geological map was interpreted (Figure 9.2). More detailed work is required to refine the interpretation.

Table 9.1 summarises the average manganese oxide values from the grab sampling programme at the Project. It should be noted that the figures shown in Table 9.1 are the average results of a grab sampling exercise focused on identifying the location of elevated concentrations of manganese mineralisation and the potential of different units to contain elevated levels of manganese. These figures are not intended to represent an estimate of the grade of the units.

Figure 9.2 Preliminary geological surface map of the K.Hill prospect area with the grab sample locations and grades



Source: Lambda Tau (2017)

Table 9.1 Summary of analytical results from the K.Hill grab sampling programme

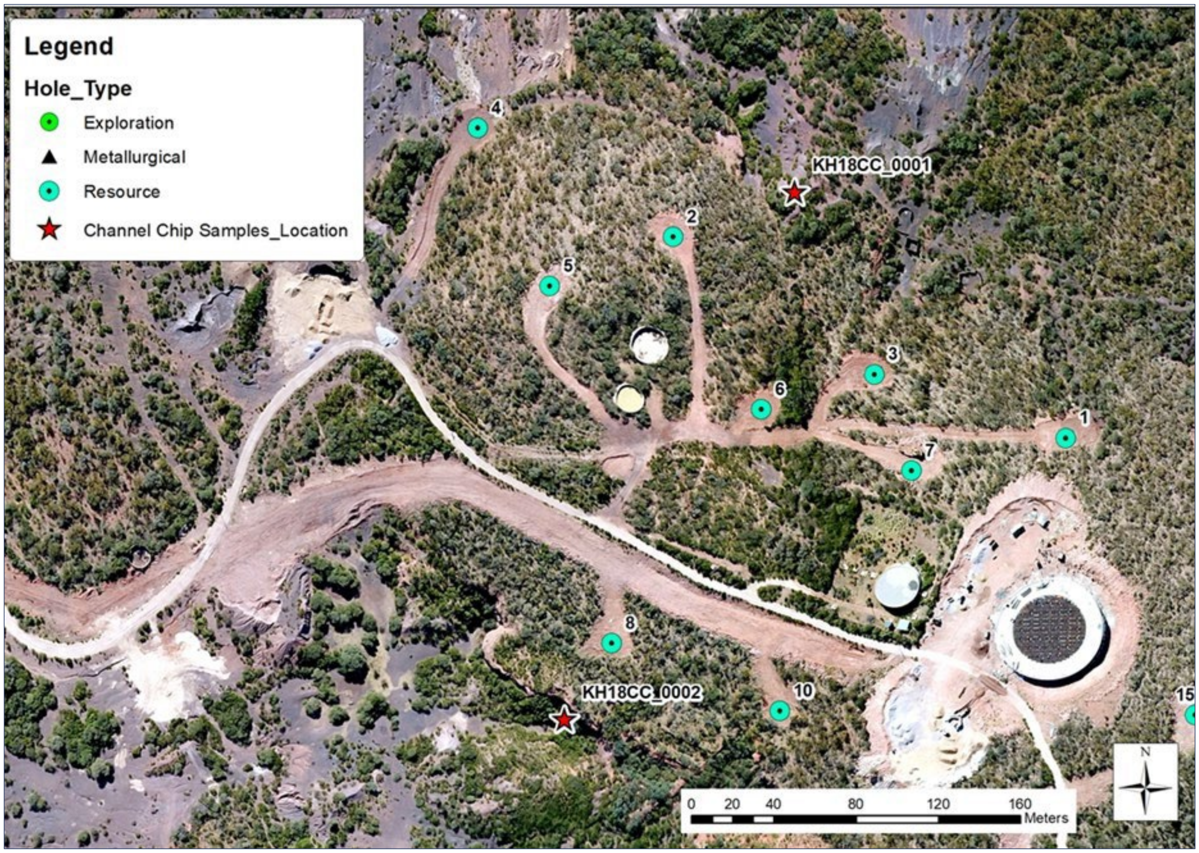
Unit	Number of samples	Maximum MnO (%)	Minimum MnO (%)	Average MnO (%)	Comments
Mn-shale	74	64.10	1.70	44.50	Main mineralised unit
Ferruginous shale	9	0.08	0.04	0.05	
Chert breccia	4	0.04	0.03	0.04	Hanging wall
Silicified shale	5	0.04	0.03	0.03	Hanging wall
Quartzite	1	32.30	32.30	32.30	Footwall; sample probably contaminated

9.2 Channel chip sampling

Channel chip sampling was conducted at the K.Hill Project at two locations. The aim was to collect representative samples from the Mn-shale without any loss in material. The intended use of the samples was for future metallurgical test work.

Two outcrops were identified for sampling: KH18CC_0001 and KH18CC_0002 (Figure 9.3). Site preparation included excavation at the bottom of the face, to a depth of 1 m, to achieve 3 m of exposed face. The exposed face was cleaned by removing the outer 5 cm of weathered material. Sampling was conducted over a 3 m intersection of the Mn-shale unit in intervals of 1 m. A fourth sample consisting of a complete channel chip sample through the entire 3 m interval was also collected at each location.

Figure 9.3 Channel chip sample locations



Source: Lambda Tau (2017)

9.3 Geophysics (gravity, induced polarisation, and magnetics)

Giyani engaged RES to complete high-resolution ground gravity and ground magnetic surveys over the K.Hill Project. RES (2018) described the geophysical surveys as follows:

- The K.Hill Project ground gravity grid comprised 1,987 planned gravity stations. All gravity grids were planned with 50 m × 50 m spaced stations. Ground magnetic data were collected on the same survey lines as the gravity grids along north-northeast to south-southwest oriented lines at the K.Hill Project. All data were collected in continuous surveying (Walkmag) mode, which translates to a reading every 1 m to 2 m on 50 m spaced survey lines. In total, 101 km of magnetic data were collected over the K Hill Project. Three, 1 km IP/DC traverses were undertaken based on the results from the gravity and magnetic data.
- Ground gravity station positions were measured using a Trimble R6 RTK DGPS. Coordinates for the beacons were provided by the Botswana Department of Surveys and Mapping in Cape LO25 format with orthometric heights. These coordinates were transformed to WGS84 UTM 35S. Gravity station positions were marked and measured by walking a pattern that included approximately 5% internal repeats as well as approximately 5% external repeats. Following RTK surveying,

gravity readings were taken over all stations. A Scintrex CG-5 Autograv gravity meter was used to complete the gravity survey.

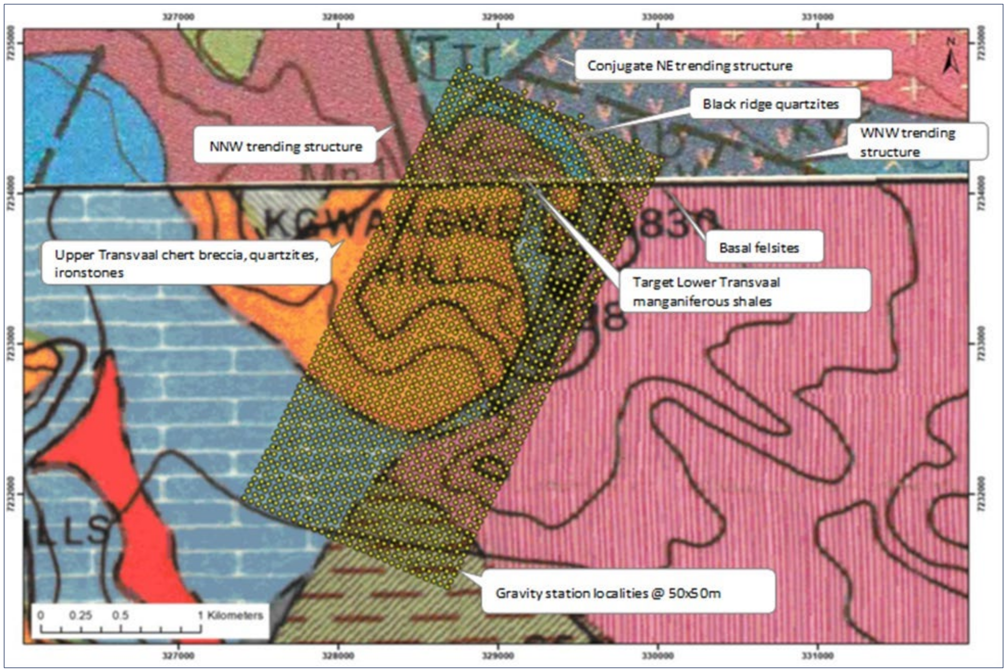
- Magnetic data were collected using a GEM GSM19 Overhauser magnetometer in Walkmag at a 1-second sampling interval. A GEM proton precession magnetometer was used to monitor and correct for diurnal variations. Location data were collected with handheld GPS, which was time synchronised with both Walkmag and base station magnetometers.
- With regards to IP/DC data collection and to evaluate the effectiveness of IP/DC techniques, three (approximately), 1 km lines of IP/DC were collected. IP/DC traverses were designed to extend from felsic volcanic basement (Kanye Volcanic Formation), over the basal unit of the Transvaal quartzites (Black Reef Quartzite Formation), into Lower Transvaal shales (Kgwakgwe Shale Formation), and into Upper Transvaal chert breccia (Kgwakgwe Chert Breccia Formation). The results of the ground gravity data were used to assist with the survey design. IP/DC data were collected in dipole-dipole configuration with a = 50 m and n = 1-7. A Zonge GDP32 receiver, a GGT10 10 kVA transmitter, and a ZMG 7.5 kVA generator were employed.
- Various filters and processes were applied.

9.3.1 Interpretation of geophysical data

Bouguer anomaly and total magnetic intensity images are provided in Figure 9.4 and Figure 9.5. The following broad geophysical characteristics can be ascribed to the geological units:

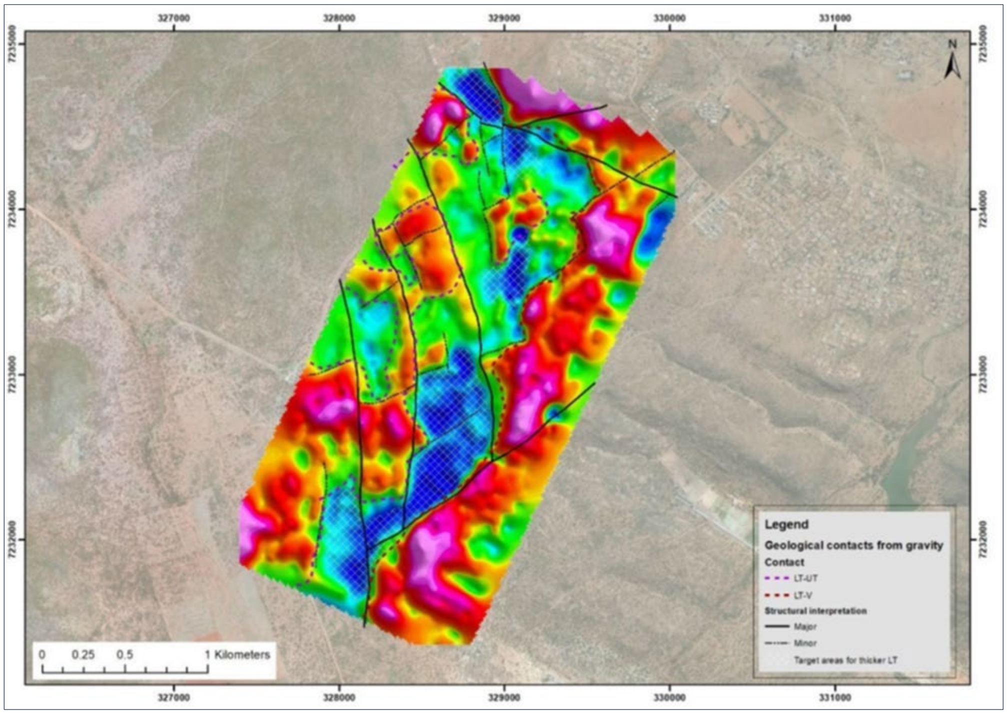
- Both gravity and magnetic datasets are dominated by the response of the felsic volcanics in the northeast and eastern portion of the survey area. These units produce significant gravity and magnetic anomalies.
- The sedimentary units of the overlying Transvaal Supergroup are clearly mapped as distinct, structurally controlled, gravity lows.
- A subtle contrast exists between the Lower and Upper Transvaal sedimentary units, with Upper Transvaal rocks appearing to be denser and producing small gravity highs.
- Thicker portions of the target Lower Transvaal units have been interpreted to correlate with more prominent associated gravity lows as a result.
- No direct gravity response appears to be associated with the manganese oxides.
- No clear magnetic contrasts have been mapped within the Transvaal sedimentary units, although there is some evidence of subtle structure in the magnetic data.

Figure 9.4 Geology with survey stations and features of interest highlighted (1:125,000 scale)



Source: RES (2018)

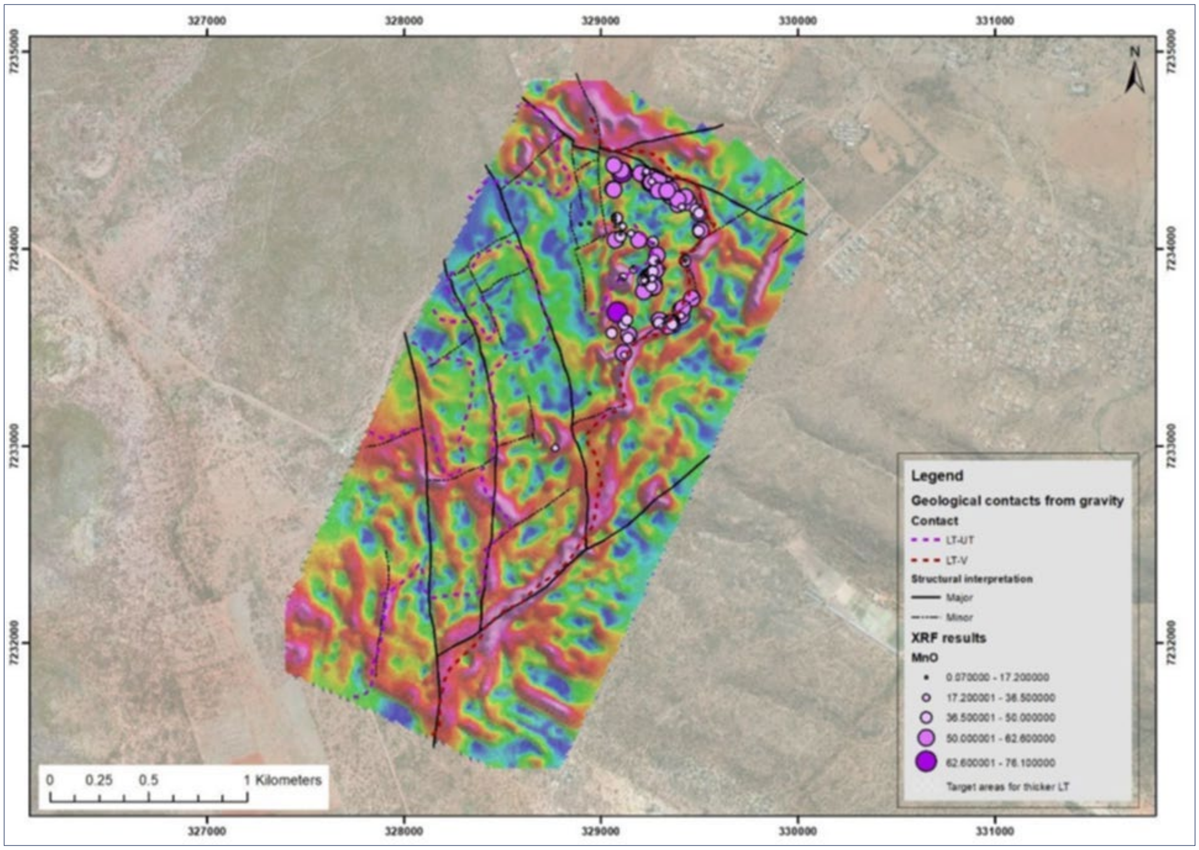
Figure 9.5 Residual filtered Bouguer Anomaly image with structure, Lower Transvaal contacts, and prospective areas for thicker Lower Transvaal units highlighted



Notes: LT - Lower Transvaal, UT - Upper Transvaal, V – Ventersdorp. Hatched areas are the target areas for thicker Lower Transvaal
Source: RES (2018)

Using several different filter and image products, as well as depth slices through the unconstrained density volume, the relationships described above were used to map basin controlling structures, contacts with the target Lower Transvaal and areas where the Lower Transvaal units appear to thicken. The interpreted contacts and target areas for thicker Lower Transvaal correlate well with known geology and sample results. Additional prospective target areas have been identified under recent cover (Figure 9.6).

Figure 9.6 Total horizontal derivative of the Bouguer Anomaly with structure, Lower Transvaal contacts and prospective areas for thicker Lower Transvaal units highlighted



Notes: LT - Lower Transvaal, UT - Upper Transvaal, V – Ventersdorp. XRF manganese oxide grab sample results are illustrated as scaled symbols
Source: RES (2018)

Ground gravity surveying proved to be an effective method for mapping out the extent and possible structural controls for manganese mineralisation in the Lower Transvaal host rocks. Basement felsic volcanics are clearly delineated in both magnetic and gravity data where they manifest as anomalous highs. IP/DC proved to be an effective means of mapping conductive Lower Transvaal host rocks over resistive Upper Transvaal and basement volcanics, as well as potentially higher chargeability manganese mineralisation (RES 2018).

The IP/DC traverse results demonstrate high correlation with the gravity inversion and interpretation. As expected, the Lower Transvaal shale units are more conductive (lower resistivity) than underlying volcanics, and overlying chert breccias correlate well with low-density portions of the inverted density volume. Distinct IP chargeability anomalies coincide with the anticipated position of Mn-shale.

Significant cultural noise (small holder developments, access roads, old workings, new drill pads, etc.) in the northern IP/DC traverse across the known mineralisation has made direct comparison with the known mineralisation somewhat ambiguous (RES 2018).

9.4 Topographic survey

Giyani engaged the services of PhotoSat Information Ltd. in April 2021 to acquire a 1 m stereo satellite survey and 50 cm orthophotograph for the 31 km^2 Project area. The satellite photographs were acquired on 2nd April 2021. The 1 m satellite survey and 50 cm precision orthophotograph were produced using PhotoSat's proprietary geophysical satellite processing system. PhotoSat was supplied with an additional 2,180 ground control points collected by DGPS and were able to use these data to vertically rectify the survey to an accuracy of 15 cm.

9.5 Mineralogical investigation

Dr. Ian Flint at the Dalhousie University Minerals Engineering Centre in Halifax, Nova Scotia, Canada, performed a mineralogical analysis on the K.Hill Project manganese-oxide bearing rocks (Lambda Tau 2018). Four samples were tested to determine the mineralogical composition of the manganese minerals. Haematite, other iron oxides, some pyrite along with silica, and kaolin in one horizon of the shales are evident as well as manganese oxides. This study is described in more detail in Section 13 of this report.

10 DRILLING

10.1 Introduction

Drilling was undertaken in five programmes at the K.Hill Project: an initial DD programme of 18 holes completed in June 2018 totalling 1,109 m, a follow-up infill programme of 96 RC holes (including 6 redrills) that commenced in November 2020 and was completed in June 2021 totalling 3,346 m, and a synchronous DD programme of 11 holes for 346 m. The 2020/2021 DD programme was primarily completed for the purposes of geotechnical analysis, and as such, this core has not been subject to sampling and assaying.

Density determinations completed on the geotechnical holes were used in assigning density to the Mineral Resource model. The fourth phase (Phase 4) of drilling to delineate the southern area was completed in August 2021 and comprised 28 RC holes and 3 DD control holes, for a total of 2,126 m.

The final phase (Phase 5) was completed in August 2022 and included 6,116 m of RC and 216.60 m of DD. In total, 75 RC holes and 7 DD holes were completed. The aim of this drilling campaign was to increase the amount of drilling in the south and decrease the spacing between drillholes on a regular grid across extension of the deposit. A few holes were drilled to extend the drilling in the south. Table 10.1 summarises the drill programmes.

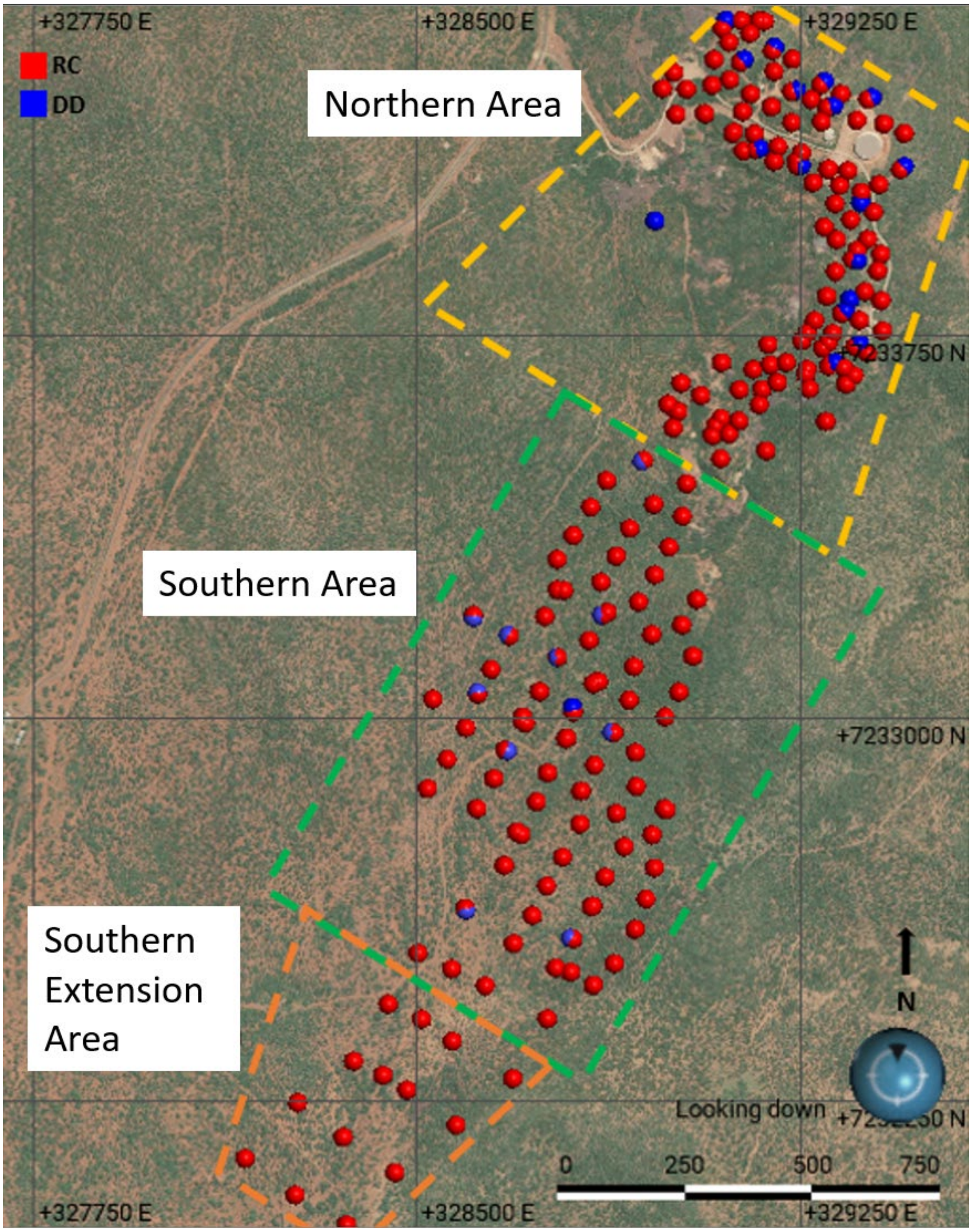
Table 10.1 Summary of drill programmes

Programme	Date	No. of holes	Meterage (m)	Hole type
Phase 1	June 2018	18	1,109	DD
Phase 2	November 2020	96	3,346	RC
Phase 3	2020/2021	11	346	DD
Phase 4	August 2021	28	1,866	RC
		3	260	DD
Phase 5	August 2022	75	6116	RC
		7	217	DD

10.2 Drillhole locations and collar surveying

Drilling was completed on approximately 35 m to 50 m centres in the northern area of the deposit and approximately 75 m to 100 m centres in the southern area and 125 m in the southern extension area (Figure 10.1). All drillholes were oriented vertically.

Figure 10.1 Plan view of the drillhole collars overlain on the K.Hill Project satellite imagery



Source: CSA Global (2023)

On completion of the drill campaigns, collars were surveyed using a DGPS. Collar positions were surveyed using a Trible R8s receiver, a Trimble TSC3 controller, and a Trimble total station. The total station was placed at coordinates 329,411.03 and 7,233,837.55 on an elevation of 1,464 m. This position was marked clearly in the field and has served as the base for all other collar surveys at the K.Hill Project.

10.3 Diamond drilling

The following description of the DD completed on the K.Hill Project to date is adapted and expanded upon from SRK (2022c). The QP notes that the drill contractor and rig and supervision team remained unchanged for both the 2018, 2021, and the 2022 DD programmes.

All DD was undertaken by RotsDrill Botswana (RotsDrill). RotsDrill used an Atlas Copco CS14 diamond drill rig. The drilling programmes were managed by Lambda Tau. Holes were drilled using PQ core diameter and cased to depth with polyvinyl chloride casing.

The standard operating procedure provides for a geologist to be assigned to the drill rig to manage the drilling, stake collars, and align and communicate with the drilling team. The drilling supervisor reported to the geologist on the progress of drilling twice a day at the change of each shift.

The geologist was responsible for safety and environmental matters. The drill rig and drill site preparation and setup were audited at the start and during the drilling programme. The geologist also ensured that all sites were rehabilitated according to the environmental management standards set forward by Giyani prior to the drilling team leaving the site.

Casing was left in the hole once a hole was completed. This was to ensure that the hole remained intact for any future downhole geophysical surveys that might be required. A concrete plinth was constructed around the drillhole collar, such that 50 cm of casing protruded above the concrete block to permanently mark the collar. The drillhole casing was sealed at the collar and a plate inserted with the drillhole name (Figure 10.2).

Figure 10.2 Drillhole collar and site for DDKH18_0005

Source: MSA (2018)

10.3.1 Transport and storage

Immediately after retrieval from the core barrel, core was placed onto a v-rail. While on the v-rail, the drillers under the supervision of the geologist marked the core as per the standard operating procedure. This included measuring the core recovery, core gains, and core losses. Thereafter, core was placed in core trays. The trays were labelled with the drillhole name at the top right corner of the box and on the side.

The drilling contractor was responsible for inserting core blocks to mark the drilled depths at the end of each run. The drill contractor was responsible for all the core handling at the drill site and transportation of the core to the core shed, located in Kanye.

The core was transported in core boxes, stacked to a maximum of five boxes, with a lid on the top core box. The stacked boxes were strapped together using a ratchet strap and further secured to the sides of the vehicle. The delivered core was accepted at the designated core shed by the on-site geologist.

10.3.2 Logging

Geotechnical and geological logging and other data were recorded at the core shed on paper and captured into Microsoft® Excel spreadsheets. After entry, the data were checked by the chief geologist. Standard logging codes were available to ensure that logging was standardised and to reduce errors in rock identification.

10.3.3 Core photography

All holes were photographed using a standard digital camera. Photographs were taken of the core, box-by-box. Drilling depth and borehole IDs were visible in each of the photographs. Once all core boxes for a hole were photographed, the individual photographs were arranged into a photograph collage and saved as a single file (Figure 10.3).

10.3.4 Sampling

The cores were cut longitudinally in half using a rotating diamond saw blade. For samples that were too soft or friable to be cut with a diamond saw, the samples were longitudinally cut in half manually with a knife blade. Half-core samples were collected continually through the mineralised units at a 1 m nominal length, which was adjusted to smaller intervals to honour the lithological contacts. Half-core samples were collected between core blocks where the recovery was less than 50%. In some instances, this resulted in samples with drilled lengths longer than 1 m.

Figure 10.3 An example of completed photograph collage of a diamond drillhole at the K.Hill Project

Source: Giyani (2022)

10.3.5 Density determinations

2018

A total of 732 density determinations were completed on the drill core during the 2018 drill programme, from the K.Hill Project northern area. These samples were not dried nor wax coated prior to undertaking the density determinations. Thus, the recorded densities were considered wet and not dry. The data were not used in previous MREs nor the current MRE presented in Section 14. The current MRE instead utilised the 2020/2021 dataset, which, while comprising only a small number of density determinations, is considered more accurate. Further discussion is provided in Section 14.

2020/2021

A total of 25 density determinations were carried out using the Archimedean water immersion method on pieces of core in both mineralised and waste rock. All determinations were taken from core in the K.Hill Project northern area. This method involves oven drying core samples, then weighing the dry core in air, and then weighing the same core while immersed in water. Giyani established that, when completely dry, the core samples tend to fragment and break down more readily once placed in water, precluding straightforward density determinations. As a consequence, samples were wax coated prior to immersion.

2022

A total of 110 density determinations were carried out using the Archimedean water immersion method on pieces of core in both mineralised and waste rock. All measurements were taken from core in the K.Hill Project southern area. This method involves oven drying core samples, then weighing the dry core in air, and then weighing the same core while immersed in water. Giyani established that, when completely dry, the core samples tend to fragment and break down more readily once placed in water, precluding straightforward density determinations. To avoid disintegration, samples were wax coated prior to immersion.

10.3.6 Recovery

The Mn-shale is friable and weathered, within which competent layers of manganese oxide mineralisation, on a centimetre scale, occur. Recovery was calculated by dividing the drilled length by the recovered length. In Phase 1 (2018), poor core recoveries were observed in most of the intersections of the Mn-shale, averaging approximately 50% in the high-grade mineralisation. Improved recoveries were achieved during Phase 2 of DD (2020-2021), with recoveries averaging 91% in the Mn-shale horizons. For Phases 3, 4, and 5, recoveries were similar, averaging 90%.

10.3.7 Other Data

Downhole surveys were not deemed to be necessary, as both the RC and DD holes were short and vertical, with end of hole depths rarely exceeding 70 m. The following information was recorded in the database:

- hole number, with collar location, length, inclination, and direction
- drilled lengths and recovered lengths (for DD only)
- geological and mineralogical descriptions
- assay results

- QA/QC samples

10.4 Reverse circulation drilling

RC drilling was undertaken by two drill contractors: Stewardship Drilling and Master Drilling. Stewardship Drilling was responsible for the completion of the first 46 RC holes of the infill campaign, each drilled with a diameter of 127 mm using a Hangin drill rig (made up of parts from different drill rigs). Master Drilling completed the remaining RC holes of the 2020-2022 drill campaign. Master Drilling operated two remote-controlled, GPS-enabled Atlas Copco D65 drill machines, drilling holes with a diameter of up to 140 mm. All RC drilling was managed by Lambda Tau.

The standard operating procedure provides for a geologist to be assigned to the drill rig to manage the drilling, stake collars, align the rig, and communicate with the drilling team. The drilling supervisor reports to the geologist on the progress of drilling twice a day at the change of each shift.

The geologist was also responsible for safety and environmental matters and ensured that all sites were rehabilitated according to the environmental management standards set forward by Giyani prior to the drilling team leaving the drilling site.

The completed RC holes were capped with concrete plugs, with the hole names inscribed and including the campaign by year of drilling (e.g., RCKH20), followed by the three-digit hole number (Figure 10.4).

Figure 10.4 Concrete reverse circulation collar plugs used to close and mark completed reverse circulation holes

Source: SRK (2021)

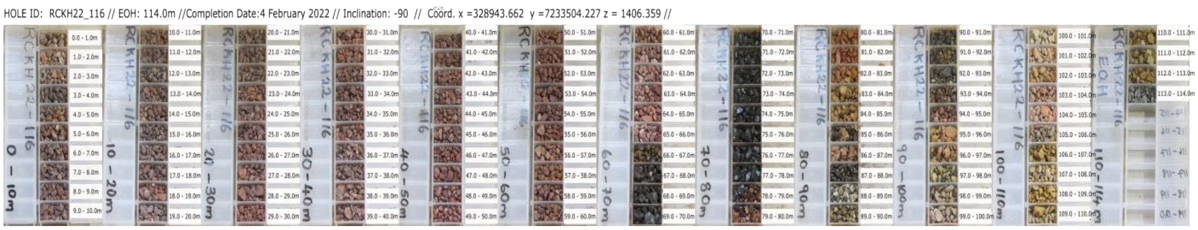
10.4.1 Logging

All RC holes were logged on a run-by-run basis (i.e., at 0.5 m intervals). Logging was completed on paper and captured into Microsoft® Excel spreadsheets. Geological logging concentrated on lithology, mineralogy, weathering, and alteration. Standard logging codes were created to ensure that logging was standardized and to reduce errors in rock, mineral, and alteration identification.

10.4.2 Photography

RC holes were photographed using a standard digital camera. Photographs were taken of each chip tray. Drilling depth and borehole IDs were visible in each of the photographs. Once all chip trays for a hole were photographed, the individual photographs were arranged into a photograph collage and saved as a single image (Figure 10.5).

Figure 10.5 Example of photograph collage made from a typical intersection at the K.Hill Project



Source: Giyani (2022)

10.4.3 Sampling

RC samples were collected at the rig site directly from the cyclone at 1 m intervals and laid out sequentially (Figure 10.6). Sample bags were pre-prepared to ensure correct labelling. The site geologists reported that the samples typically range from between 10 kg to 12 kg. The entire sample was transported to the core shed for sample preparation and analysis.

RC chip samples were gathered from the primary sample. The chips were washed in water and placed into a standard chip tray. The chip trays were clearly labelled with the hole ID and depth intervals. Only those primary samples deemed to be significantly mineralized with manganese were spilt into an A and B sample using a standard 50/50 riffler splitter. This produced two sub-samples, each of approximately 3 kg. Both samples were transported to the core shed, closed using staples, and stored in a secure location with access limited to the on-site laboratory staff.

Figure 10.6 RC samples laid out sequentially at the rig

Source: SRK (2021)

10.4.4 Density determinations

Density determinations were carried out on RC samples using a formula that combined drill sample volume, recovery, and weight of sample. Density was determined from sample weights per 1 m interval. With a known diameter of the RC drill bit (~13.97 cm) and a known drill interval (100 cm), a volume of 15,328 cm^3 was calculated and used for RC bulk densities. This method has its own limitations which were observed on site, such as insufficient sample recovery.

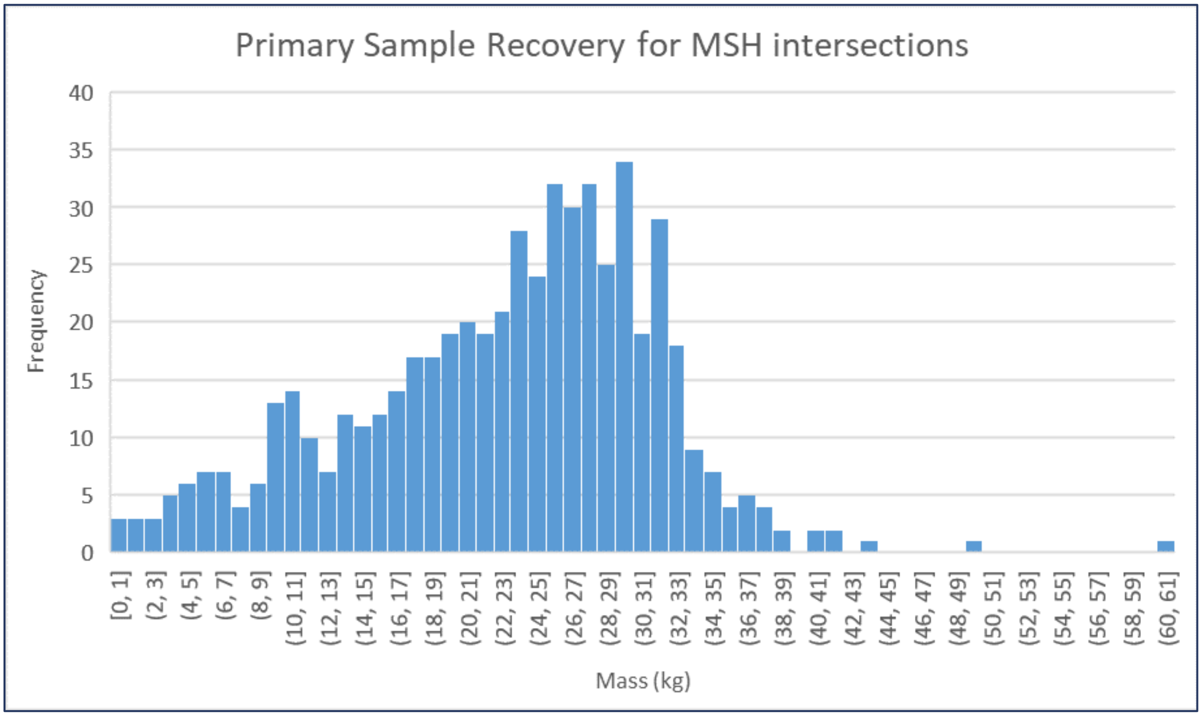
Over 1,552 RC sample densities were calculated from both the northern and southern areas. A comparison of RC sample densities versus those from DD core has been made. A comparison of the southern extension area RC densities with that of densities taken from diamond core using conventional Archimedean method shows comparable results for the Mn-shale.

10.4.5 Recovery

Giyani routinely monitored RC bag weights to track sample recovery during drilling. The average bag weight per each drilled metre, after being composited to 1 m, is plotted against the prevalence of logged intervals of Mn-shale. Analysis of this data shows no systematic effects or unexpected changes in recovery downhole and no significant decrease in recovery in zones of Mn-shale.

For the latest drilling programme (Phase 5) the sample recoveries for those intersections logged as Mn-shales (MSH) are displayed as a frequency plot in Figure 10.7. These data display a tendency toward a normal distribution, with most samples returning a weight of between 25 kg and 30 kg.

Figure 10.7 Frequency plot of primary sample masses during reverse circulation drilling (Phase 5)



Source: CSA Global

10.4.6 Other data

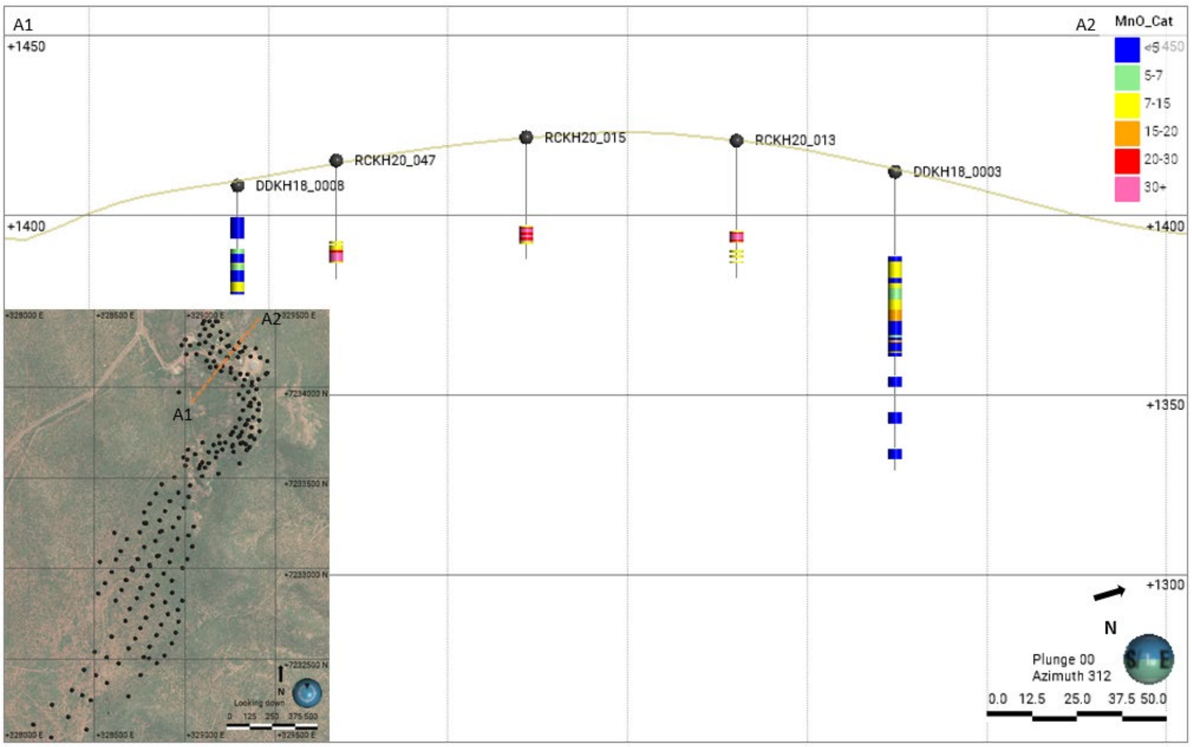
On completion of each hole, collar surveys were completed using a DGPS. Downhole surveying was not necessary as the drilled holes were short and vertical with end-of-hole depths ranging from 10 m to 41 m (typically 30 m). The following information was recorded in the database:

- hole number, with collar location, length, inclination, and direction
- drilled lengths and recovered lengths
- geological and mineralogical descriptions, including weathering and alteration
- assay results
- QA/QC samples

10.5 Significant intercepts

A table of significant intersections above 15% manganese oxide was compiled for a representative cross section in the northern area (Figure 10.8 and Table 10.2).

Figure 10.8 Representative northern area cross section and plan view



Source: CSA Global

Table 10.2 Significant intercepts above 15% manganese oxide for representative northern area cross section

Borehole ID	Type	Year	X Collar	Y Collar	Z Collar	EOH	From	To	MnO (%)
RCKH20_047	RC	2020	329,207	7,234,118	1,416	33.00	25.0	25.5	16.5
							25.5	26.0	18.4
							26.0	26.5	18.5
							26.5	27.0	16.5
							27.0	27.5	16.6
							27.5	28.0	19.5
RCKH20_015	RC	2020	329,234	7,234,165	1,422	34.00	25.0	25.5	24.1
							25.5	26.0	17.8
							26.0	26.5	19.6
							26.5	27	20.1
							27	27.5	15.5
							27.5	28	15.2
							28	28.5	20.9
							28.5	29	26.3
RCKH20_013	RC	2020	329,271	7,234,210	1,421	38.50	25.5	26	20.7
							26	26.5	19.0
							26.5	27	19.0
							27	27.5	19.5
							27.5	28	19.7
							28	28.5	22.9
							33.5	34	28.6
DDKH18_0003	DD	2018	329,298	7,234,245	1,246	83.29	38.5	41.5	24.6
							47	47.5	22.9
DDKH18_0008	DD	2018	329,161	7,234,110	1,406	30.22	No intersection above 15%		

Notes:
EOH - end of hole

A table of significant intersections above 15% manganese oxide was compiled for a representative cross section in the southern area (Figure 10.9 and Table 10.3).

Figure 10.9 Representative southern area cross section and plan view

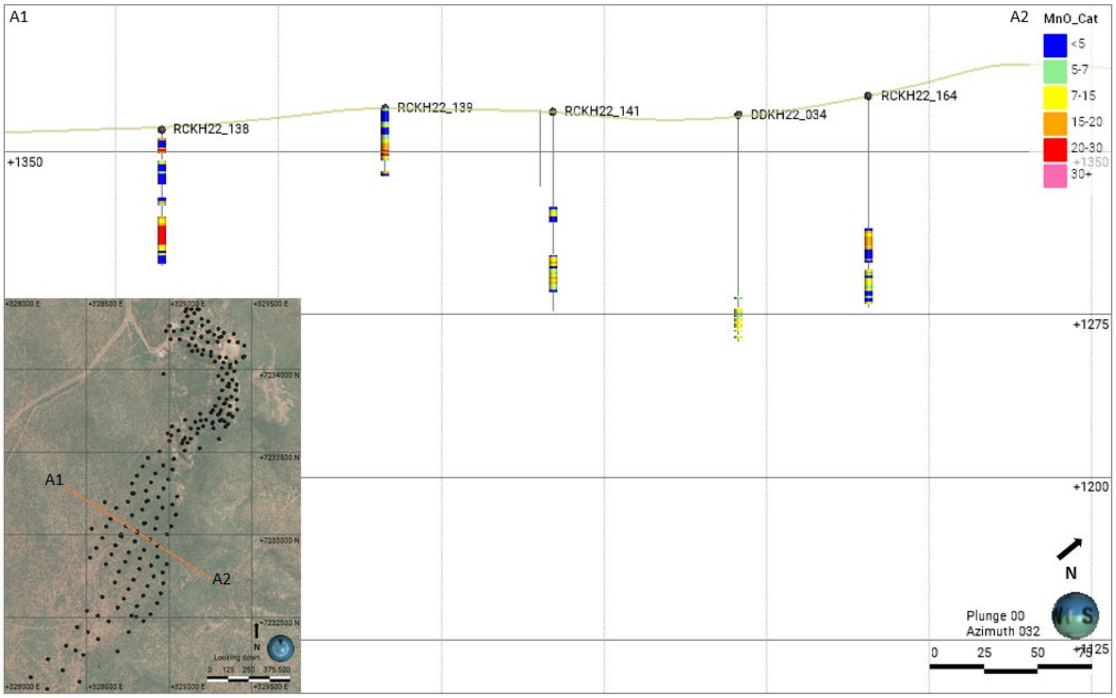


Table 10.3 Significant intercepts above 15% manganese oxide for representative southern area cross section

Borehole ID	Type	Year	X Collar	Y Collar	Z Collar	EOH	From	To	MnO (%)
RCKH22_138	RC	2022	328,645	7,233,095	1,360	63.00	4	5	24.4
							9	10	39.3
							10	11	31.8
							33	34	19.3
							40	41	27.5
							43	44	31.7
							44	45	19.3
							45	46	24.3
							46	47	31.9
							47	48	39.5
							48	49	36.0
							49	50	15.1
							50	51	16.3
							51	52	22.8
							52	53	15.6
RCKH22_139	RC	2022	328,740	7,233,051	1,370	31.00	16	17	16.8
							17	18	16.3
							18	19	44.0
							19	20	25.0
							20	21	16.6
							21	22	34.9
							29	30	28.3
RCKH22_141	RC	2022	328,806	7,233,011	1,369	92.00	69	70	32.8
							76	77	17.2
							78	79	26.3
DDKH22_034	DD	2022	328,881	7,232,971	1,368	105.14	No intersection above 15%		
RCKH22_164	RC	2022	328,928	7,232,932	1,377	98.00	62	63	20.8
							65	66	32.5
							66	67	37.3
							67	68	42.8
							68	69	27.9
							70	71	18.4
							81	82	21.9

Notes:
EOH - end of hole

10.6 Drillhole database

Drilling information was captured into Microsoft® Excel spreadsheets. Data was continuously monitored and checked for errors by Mr. Luhann Theron of Lambda Tau, Giyani's chief geologist. All data was backed up on cloud hosted servers. The Microsoft® Excel sheets were combined to create a master drillhole database in Microsoft® Excel.

10.7 Comments

The QP considers that the quantity and quality of data collected in the K.Hill Project drilling programme is sufficient to support the estimation and reporting of a Mineral Resource. That said, the following concerns are highlighted:

- Core recovery in the Phase 1 DD programme is poor, particularly within the Upper Mn-shale unit that is host to the manganese oxide mineralisation. Specifically, average core recovery within the Upper Mn-shale unit is 41%. Recovery is less than 80% for approximately 85% of samples and less than 50% for approximately 60% of samples. Three holes have no core recovery at all at the anticipated depth of the Upper Mn-shale unit. Recoveries are much improved in the most recent Phases 3, 4, and 5 DD campaigns, averaging 90% in the Mn-shales. At this stage, there is no clear relationship observed between core recovery and grade, although there is only limited data available to assess this.
- Due to the tendency of the more friable portions of drill core to fragment when dry, Giyani did not oven dry the drill core from the Phase 1 DD programme prior to completing density determinations. As no moisture content for the samples was recorded, it was not possible to accurately account for the water content of the samples. Data collected during Phases 3, 4, 5 DD programmes, whereby samples were dried and wax coated prior to density determination, were used for the MRE. Although this data is deemed to be more accurate, it is a significantly smaller dataset.

It is recommended that Giyani continue to undertake density determinations on core using the wax coating method.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Introduction

The descriptions of sample preparation, analysis, and QA/QC checks undertaken on the Phase 1 DD programme in Section 11 have been reproduced from MSA Group (2018), and the QA/QC checks undertaken on the RC drilling campaigns have been reproduced from SRK (2022c). CSA Global documented the Phase 5 drilling campaign.

11.2 Sample security

Samples were dispatched in batches of approximately 100 samples. The sample batches were exported by Aramex (for the DD programme) and Pinnacle Express (for the RC programme) to SGS Randfontein in South Africa for geochemical analysis.

The chain-of-custody was maintained by signature at every point in which the samples changed hands, from the core shed in Kanye, where the samples were stored, to the laboratory. As part of the laboratory procedure, all samples were weighed. All persons involved in the chain-of-custody were required to submit a copy of their receipt of handover of the samples to the project manager for record keeping on site.

11.3 Diamond drill core sample preparation

Samples were prepared and analysed at SGS Randfontein. This is an independent commercial laboratory, which is ISO 17025 accredited by the South African National Accreditation System for chemical analysis. The sample preparation method code is PRP87, which entails the following procedure:

1. Samples are weighed on arrival.
2. The samples are dried and then crushed using a jaw crusher to 80% passing through a 2 mm screen.
3. A 500 g sub-sample is collected from a riffle splitter.
4. The 500 g sub-sample is pulverised using a carbon steel ring and puck to 85% passing through a 75 µm screen.
5. Pulps are logged against sample numbers and submitted for analysis.
6. All pulverised reject samples are stored at SGS Randfontein.

11.4 Reverse circulation sample preparation

Sample preparation for the RC samples follows a procedure detailed in the LT20051R standard operating procedure:

1. Samples received from the drill site are weighed to record the full primary sample.
2. A 3 kg to 4 kg "A-sample" subset is collected from a riffle splitter, with the residual primary sample retained until the completion of the A-sample analysis or for metallurgical test work.

3. Samples are washed to collect a reference chip sample and to visually identify the interval of manganese mineralisation.
4. The mineralised intervals, including four bounding samples either side, are compiled with QA/QC samples to form the sample stream for laboratory analysis.
5. If two or more mineralised intersections are identified, all material internal to these intervals are sampled.
6. The A-sample mineralised stream is separated and marked for p-XRF analysis.
7. Ten to fifteen grams of the A-sample are pulverised for a programmed time and speed (240 seconds) using an Equilab EQM-402 MixerMill, then placed into a sample cup for analysis by p-XRF.

11.4.1 Field geochemical analysis

To select intervals for laboratory assays, the field team followed a simple procedure that involved visual inspection of the A-samples and RC chip trays, followed by p-XRF analysis. Intervals with manganese oxide mineralisation are visually distinct from barren footwall and hanging wall rock types, in that manganese oxide mineralisation appears as a dark grey to black powder/chips within the sample bags and trays. Once the mineralised intersections were identified, the field team proceeded to conduct p-XRF analysis.

The p-XRF analysis was conducted on all visually mineralised intersections. The p-XRF analysis was conducted systematically by first selecting the analysis intervals and removing a small amount (~30 g) of RC dust from the A-sample and placing it into a cylindrical plastic tube. The plastic tube was filled to the brim and the open end sealed with cellophane (cling wrap).

Once the selected sample intervals were placed into labelled cylindrical plastic cups, the laboratory technician commenced p-XRF analysis. Each sample cup was placed into an analysis chamber, with the cellophane side down, on top of the p-XRF detector beam.

Analysis time for each sample was 30 seconds, providing for accurate detection of a large suite of compounds. Compounds of interest included aluminium oxide, barium oxide (BaO), calcium oxide, chromium oxide, iron oxide, potassium oxide, magnesium oxide, manganese oxide, sodium oxide, phosphorus pentoxide, sulphur trioxide (SO_3), silicon dioxide, and titanium dioxide.

InnovX Africa calibrated the p-XRF before the start of the project. During project activities, routine calibration checks were conducted by analysing certified reference materials (CRMs) with known manganese oxide and ferrous oxide (FeO) contents and comparing them with the results obtained by the p-XRF machine.

Following the field p-XRF analysis, the technicians shared the geochemical results with the project geologist. The project geologist then compiled a sample list by selecting all sample intervals >5% manganese oxide for submission to Intertek Genalysis. A shoulder sample of 4 m (or four samples of 1 m each) on either side of the mineralised interval was included in the sample list. Each of the selected intervals were assigned a unique sample ID within the sample list. The project geologist also inserted the appropriate amount of QA/QC samples into the sample stream.

11.5 Diamond drilling and reverse circulation sample analysis

All samples were assayed at SGS Randfontein using method XRF76V, which assays major element oxides by XRF using borate fusion. The oxides assayed included aluminium oxide, calcium oxide, chromium oxide, iron oxide, magnesium oxide, manganese oxide, sodium oxide, phosphorus pentoxide, potassium oxide, silicon dioxide, titanium dioxide, and vanadium pentoxide (V_2O_5) reported in percent. LOI was also determined.

The analytical procedure involves the following:

1. A pulverised sample between 0.2 g and 0.7 g is required for analysis.
2. The samples are mixed with 10 g of flux, which is made up of equal amounts of lithium tetraborate-metaborate and a non-wetting agent.
3. The sample is fused to create a bead.
4. XRF is carried out on the fused bead.
5. LOI is determined separately by roasting approximately 1 g of the pulverised sample at 1,000°C for 1 hour in a furnace.

11.6 Quality assurance/quality control

11.6.1 Introduction

The QA/QC section is summarised in three parts:

- QA/QC for Phases 1 and 2 drilling (pre-September 2021)
- QA/QC for Phase 4 southern area drilling (post-September 2021)
- QA/QC for Phase 5 southern area drilling (post-September 2022)

11.6.2 Phases 1 and 2 diamond drilling

This section describes the procedures undertaken for QC samples submitted as part of the Phase 1 DD programme. No comment is provided on the Phase 3 DD programme, as the drilling was undertaken for geotechnical purposes, and no analysis of core samples was undertaken.

QC samples were inserted to test analytical accuracy, laboratory contamination, and repeatability on a hole-by-hole basis. CRMs and blank samples were inserted into the sample stream with the core samples. Coarse duplicate samples were inserted at the analytical laboratory. Empty bags with sample labels were submitted to the sample preparation laboratory with an instruction to make a duplicate of a specified sample and insert it into the sample sequence. One CRM, one blank, and one duplicate sample were inserted into the sample stream for every 20 core samples.

Blanks

A total of 19 blank samples were inserted within the field sample stream to detect contamination, especially in the preparation stage. The number of blank samples inserted equalled a 5% insertion rate, which is in line with industry practice. Giyani used blank silica chips from African Mineral Standards (AMIS), as shown in Table 11.1. The graphs for all blank analyses can be seen in Table 11.1.

Table 11.1 AMIS0439 blank silica chips certified mean grades and 2 standard deviations

Variable	Certified value (%)	Two standard deviations (%)
Al_2O_3	0.99	0.13
CaO	0.02	0.01
Cr_2O_3	0.01	0.002
Fe_2O_3	1.53	0.23
K_2O	0.21	0.04
MgO	0.03	0.01
MnO	0.01	0.01
Na_2O	0.02	0.01
P_2O_5	0.01	0.003
SiO_2	96.9	0.4
TiO_2	0.06	0.01

Figure 11.1 Results of blank sample assays

Source: MSA Group (2018)

Apart from two blank samples, all manganese oxide assays for blank samples were below the upper limit. The upper limit is the threshold below where the blank assays are expected to be when there is no contamination and is generally taken at ten times the lower detection limit for the method used for each analyte. The blank assays that returned values beyond the upper limit had assays of between 0.19% and 42.5% manganese oxide.

An assay of 42.5% manganese oxide is considered too high to result from contamination. Therefore, it is concluded that this sample was swapped with a field sample. The other blank failure, with an assayed grade of 0.19%, could have been due to contamination, as it follows a CRM, which has an assayed grade of 60.1% manganese oxide. Contamination should not have taken place during the sample preparation phase, as CRMs are inserted as pulverised material, so they do not require any crushing, the point at which most contamination takes place. Overall, the blank sample assay grades for manganese oxide show that there may have been some contamination at the laboratory, but this was low and would not have had any significant impact on the manganese oxide grade of the core samples.

Analyses of blank samples were also undertaken for aluminium oxide, calcium oxide, iron oxide, magnesium oxide, and phosphorus pentoxide. The aluminium oxide and iron oxide grades were judged in reference to 3 standard deviations of the certified values. The average assayed grade of the blank sample for aluminium oxide and iron oxide was 1.4% and 2.6%, respectively. Out of 19 samples, 1 was interpreted to be a sample swap, 11 iron oxide assays were outside 3 standard deviations of the certified value, and 5 aluminium oxide assays were outside 3 standard deviations of the certified value.

It is considered that the level of possible contamination within the Phase 1 DD samples is low. The reason for the elevated iron oxide values is uncertain, as the sample cannot be contaminated for only one analyte. This is more likely either an assay accuracy issue or inherent variability in the blank material. It is recommended that Giyani monitor the accuracy of the SGS Randfontein results for iron oxide.

Certified reference material

Samples of a single CRM (AMIS0403) were inserted within the K.Hill Project drillhole core sample stream. This CRM was made from manganese mineralisation from the Wessels Mine in the Kalahari Manganese Field in South Africa (Table 11.2).

Table 11.2 Certified reference material AMIS0403 certified values

Variable	Certified value (%)	Two standard deviations (%)
MnO	60.42	0.64
Al_2O_3	0.37	0.02
CaO	5.12	0.14
Fe_2O_3	18.52	0.36
MgO	0.66	0.06
P_2O_5	0.08	0.008
SiO_2	5.25	0.18
LOI	4.27	0.48

The number of inserted CRMs was 19, equating to a 5% insertion rate for the K.Hill Project samples, which is in line with acceptable industry practice. The manganese oxide grade of the CRM was high relative to the K.Hill Project samples, and more than one type of CRM is generally recommended.

Out of 19 samples, only 1 sample returned a value outside 3 standard deviations of the certified manganese oxide value. This sample had a grade of 0.02% manganese oxide and is considered to be a sample swap. The average grade of the CRM assays, excluding the sample swap, is 60.51% manganese oxide, which compares favourably with the CRM certified value of 60.42% manganese oxide (Figure 11.2).

The CRM samples were also assayed for aluminium oxide, calcium oxide, iron oxide, magnesium oxide, phosphorus pentoxide, and silicon dioxide. The CRM assays for these variables was within 3 standard deviations of the certified value of the respective variables, except for one aluminium oxide assay that returned a value above the limit. None of the variables showed significant bias, and none exceeded 4% relative difference from the certified values (Figure 11.2). In general, the assayed grades of the Phase 1 DD CRM samples indicate acceptable analytical accuracy at the grade of the CRM. That said, the single CRM used does not fully confirm the accuracy of assays at the ranges of manganese oxide grades at the K.Hill project, since the K.Hill Project drillhole core samples have lower manganese oxide grades than the certified value of the CRM.

Coarse duplicates

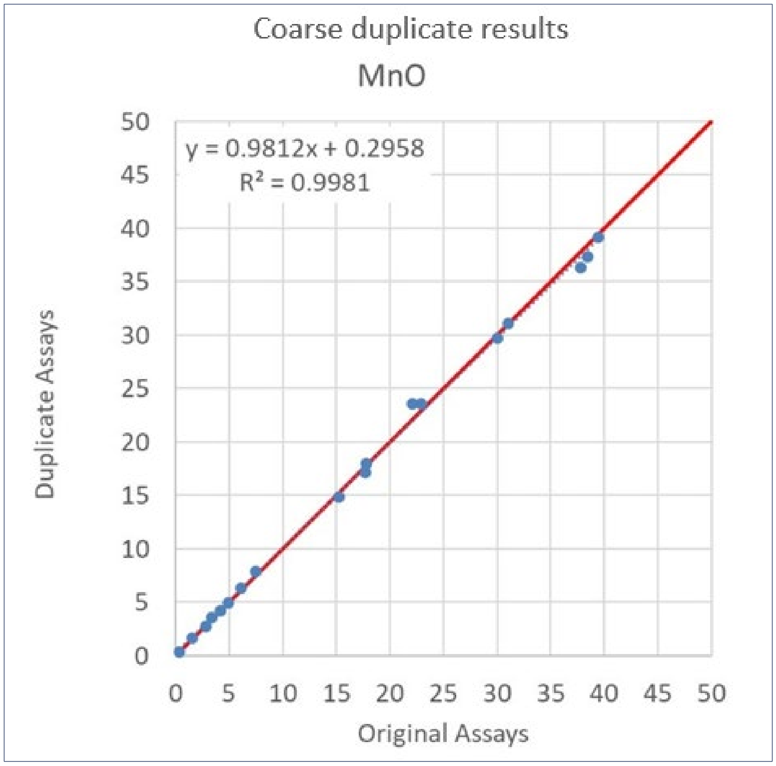
Coarse duplicates were inserted to assess the adequacy of the sub-sampling process after crushing and the repeatability of the analytical process. As per the laboratory sample preparation standard operating procedure, a sub-sample of 500 g was collected using a riffle splitter after crushing. At this point, a second sub-sample was collected as a coarse duplicate and assigned a different sample number.

The duplicate assays of manganese oxide, aluminium oxide, calcium oxide, iron oxide, magnesium oxide, phosphorus pentoxide, and silicon dioxide showed good precision (or repeatability) with linear correlation coefficients of greater than 0.9 (Figure 11.3 and Figure 11.4). This suggests that the sub-sampling process and the analytical processes are repeatable, and that the results are appropriate for Mineral Resource estimation and reporting.

Figure 11.2 Results of AMIS0403 certified reference material sample assays

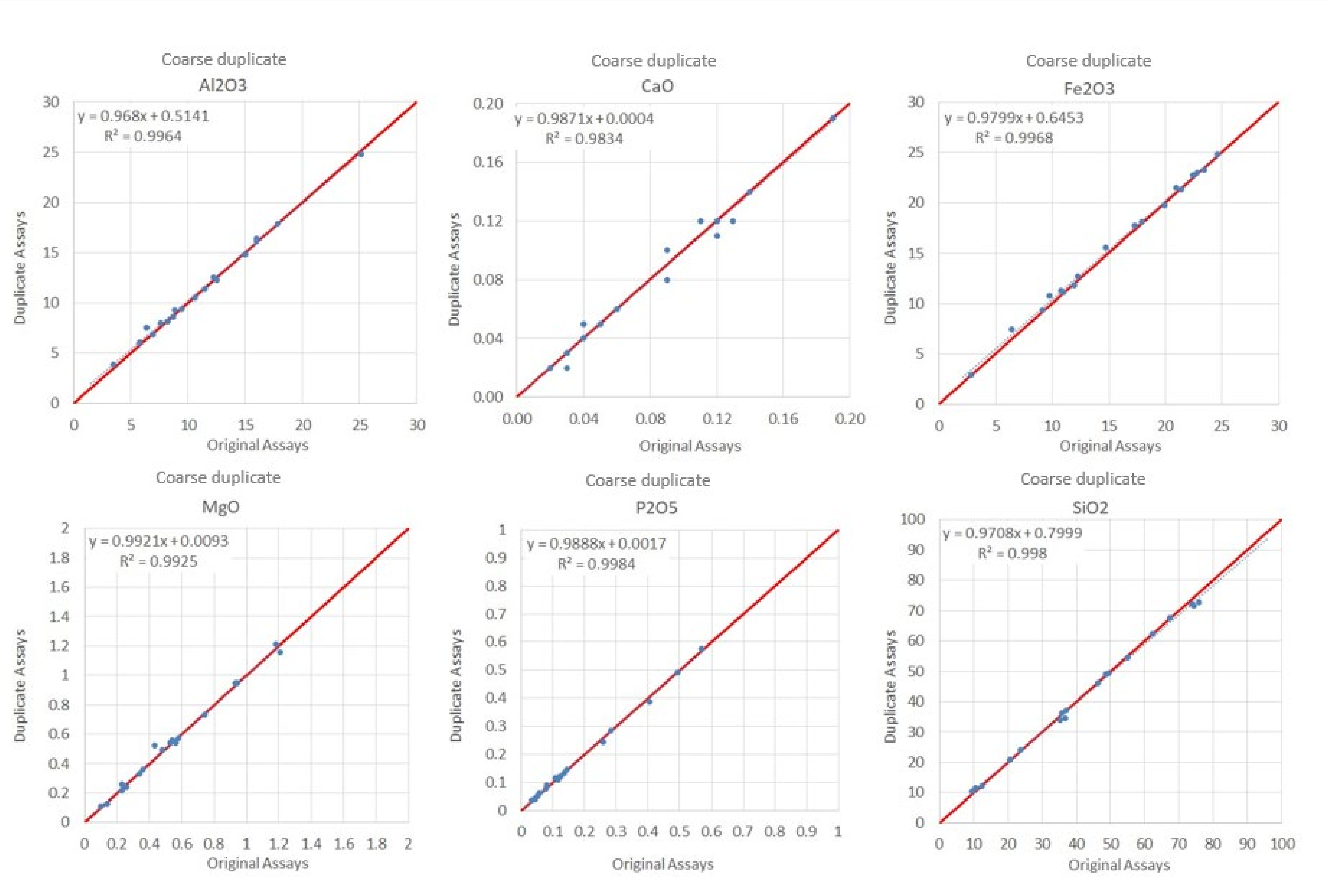
Source: MSA Group (2018)

Figure 11.3 Scatterplot of manganese oxide assay pairs (in percent) of duplicate samples



Source: MSA Group (2018)

Figure 11.4 Scatterplots of duplicate sample assay pairs (in percent)



Source: MSA Group (2018)

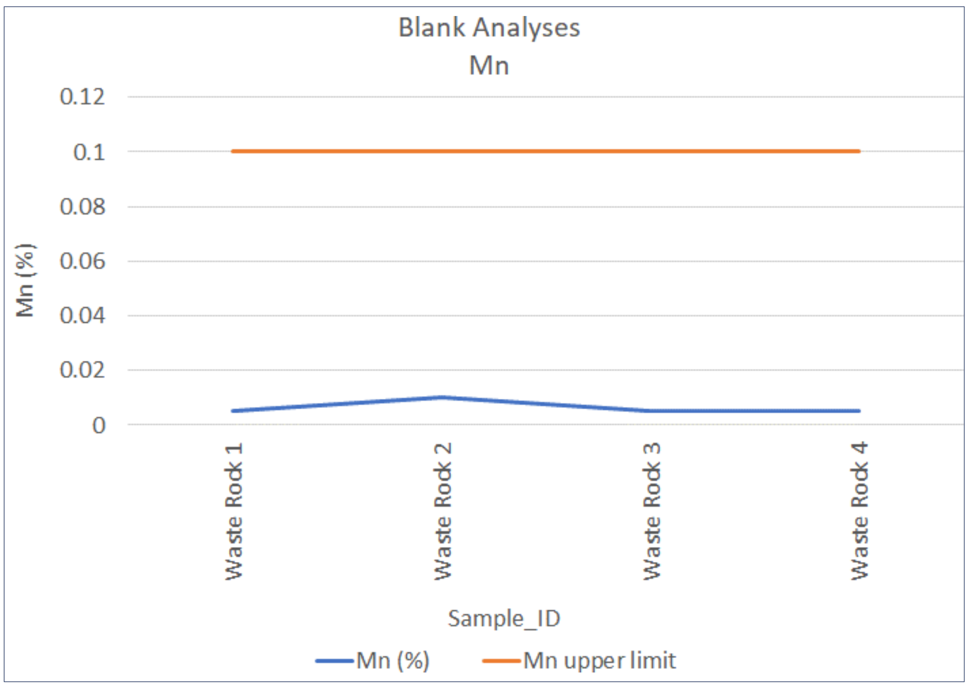
Check assays

Reject material from 40 samples that SGS Randfontein previously had assayed was submitted to Intertek Genalysis laboratory in Maddington, Western Australia. Intertek Genalysis is an independent commercial laboratory, which is ISO 17025 accredited by the National Association of Testing Authorities, Australia for chemical testing. The duplicate samples were accompanied by four CRM and four blank samples. The assay method undertaken on these samples was XRF, similar to the primary laboratory (SGS Randfontein) assay method. The insertion rate was 10%.

The blank samples accompanying the check assays returned grades close to the detection limit for all variables, including manganese oxide (Figure 11.5). This indicates that there was limited, if any, contamination at the laboratory during the assaying of these samples.

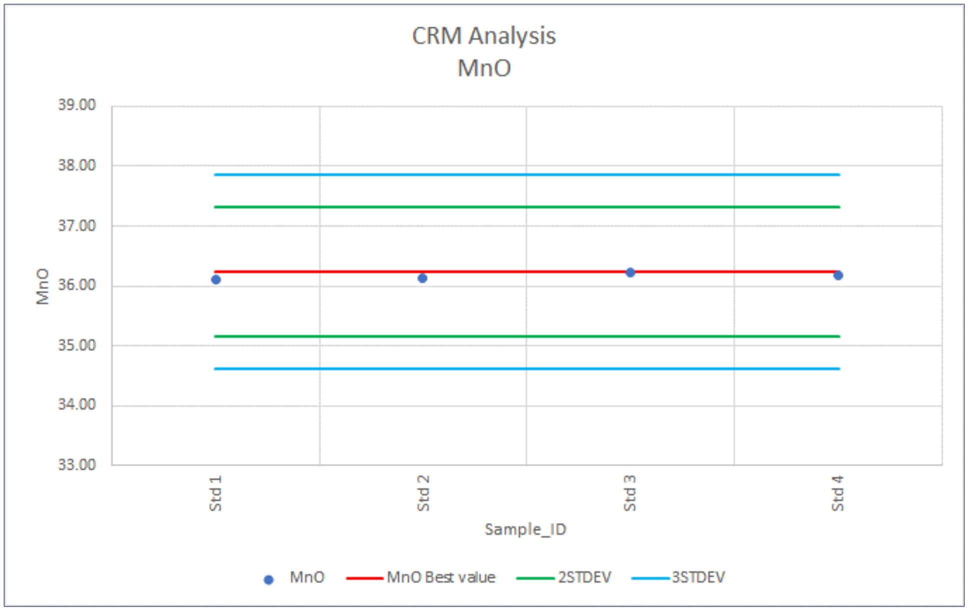
The CRM samples inserted with the check assay batch were sourced from AMIS0407. The assays that were returned were within 2 standard deviations of the certified manganese oxide value for this method (Figure 11.6).

Figure 11.5 Assays of manganese oxide in blank samples accompanying the check assays



Source: MSA Group (2018)

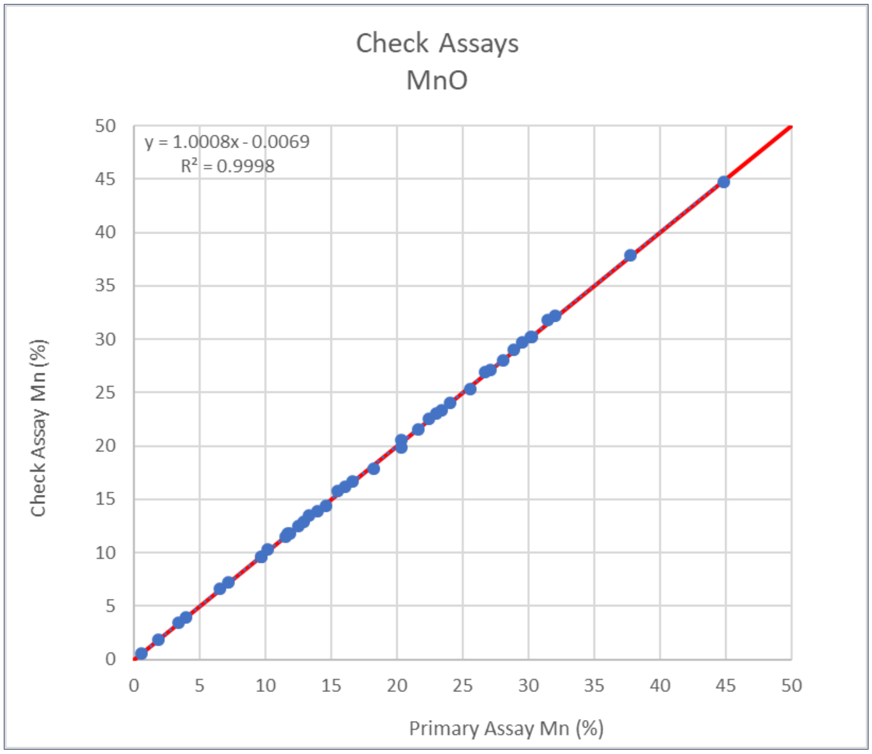
Figure 11.6 Manganese oxide assays (in percent) of AMIS0407 samples by the check laboratory



Source: MSA Group (2018)

A scatterplot comparing the manganese oxide assays from the primary and the secondary laboratory shows very good correlation, with a linear correlation coefficient of approximately 1 (Figure 11.7).

Figure 11.7 Scatterplot of manganese oxide assays (in percent) of the primary laboratory versus the secondary laboratory



Source: MSA Group (2018)

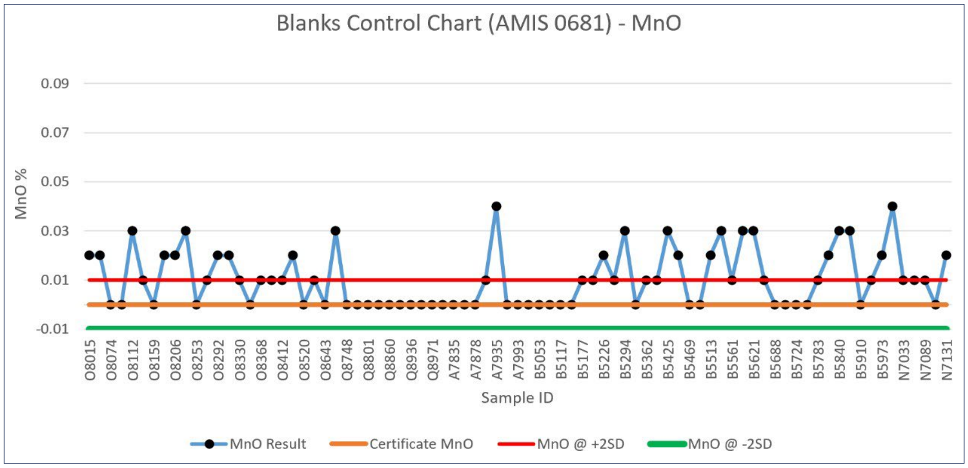
11.6.3 Reverse circulation Phases 1 and 2

During the RC drilling campaign (96 drillholes) conducted between November 2020 and June 2021, a total of 427 QA/QC samples were inserted into the sample stream. The QA/QC samples included 121 blanks, 224 CRMs, 42 field duplicates, and 40 pulp duplicates, representing an overall QA/QC sample insertion rate of approximately 13%.

Blanks

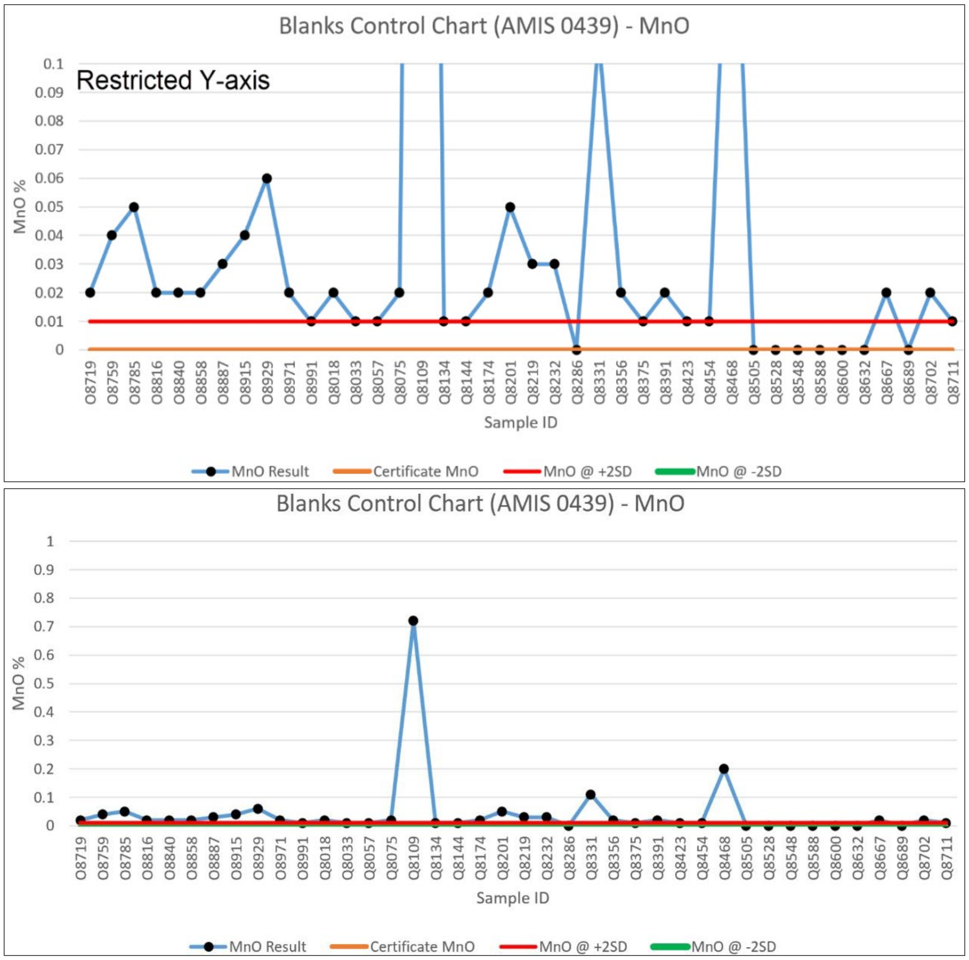
Giyani used AMIS0681 and AMIS0439 blank silica chips for blank material submission to the laboratory to monitor sample contamination (Figure 11.8 and Figure 11.9). The results show a significant proportion of samples around or exceeding the specified failure threshold (0.01% manganese oxide). Three samples returned manganese oxide values greater than 0.1%, but all less than 1%. While noting that the results suggest minor contamination, the degree of contamination (typically less than 0.05% manganese oxide) is not considered material in the context of the average manganese oxide grades within the shale horizons.

Figure 11.8 Blank sample (AMIS0681) control plot, 2020/21 reverse circulation drilling



Source: SRK (2022c)

Figure 11.9 Blank sample (AMIS0439) control plot, 2020/21 reverse circulation drilling



Source: SRK (2022c)

Certified reference material

Samples of three CRMs (AMIS0407/533/535), covering a grade range of approximately 24% to 47% manganese oxide, were inserted into the K.Hill Project drillhole RC sample stream. The CRMs are produced by AMIS and derived from manganese mineralisation from the Sakura Ferro-Alloy (Malaysia) and the Mamatwan Mine in the Kalahari Manganese Field, South Africa. CRM grades for the main analysed elements are provided in Table 11.3. The manganese oxide grades of the CRMs are appropriate for the average grade of the mineralised K.Hill Project samples, although inclusion of a lower-grade CRM at or around the Mineral Resource reporting grade (8% to 12% manganese oxide) would be a beneficial addition.

Table 11.3 Certified reference material values for manganese oxide

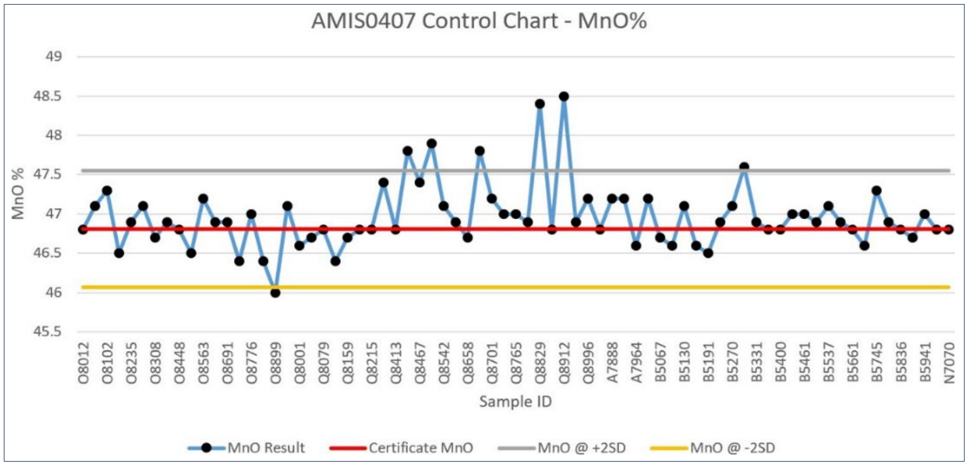
CRM	Certified value (%)	Two standard deviations (%)
AMIS0407	46.81	0.740
AMIS0533	23.97	0.555
AMIS0535	26.70	0.830

The submission and failure rates (above or below 2 standard deviations) of the three CRMs are summarised in Table 11.4, with performance illustrated graphically in Figure 11.10 to Figure 11.12.

Table 11.4 Summary of certified reference material performance for the Phase 2 reverse circulation drilling

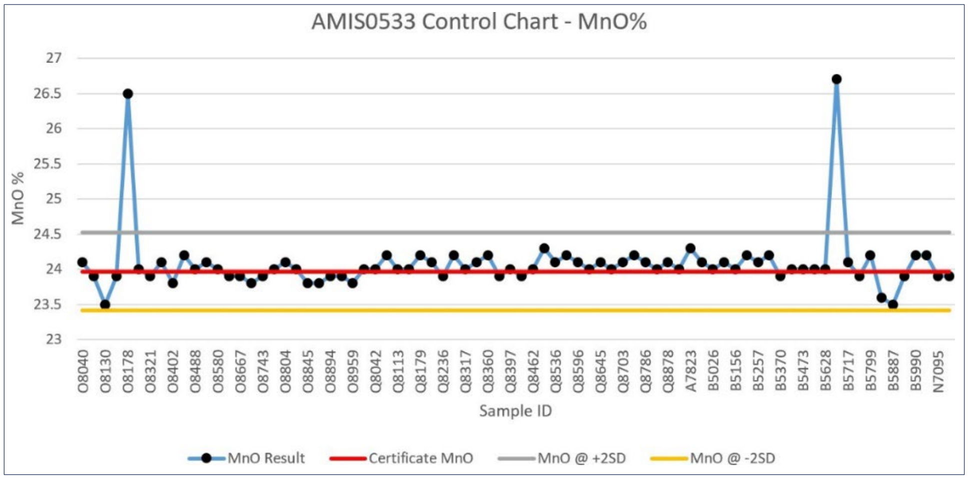
CRM	Number of submissions	Insertion rate (%)	Number of failures	Failure rate (%)
AMIS0407	73	2.2	7	10
AMIS0533	78	2.3	2	3
AMIS0535	73	2.2	2	3
Total	**224**	**2.22**	**11**	**5**

Figure 11.10 Certified reference material control plot for AMIS0407, 2020/21 reverse circulation drilling



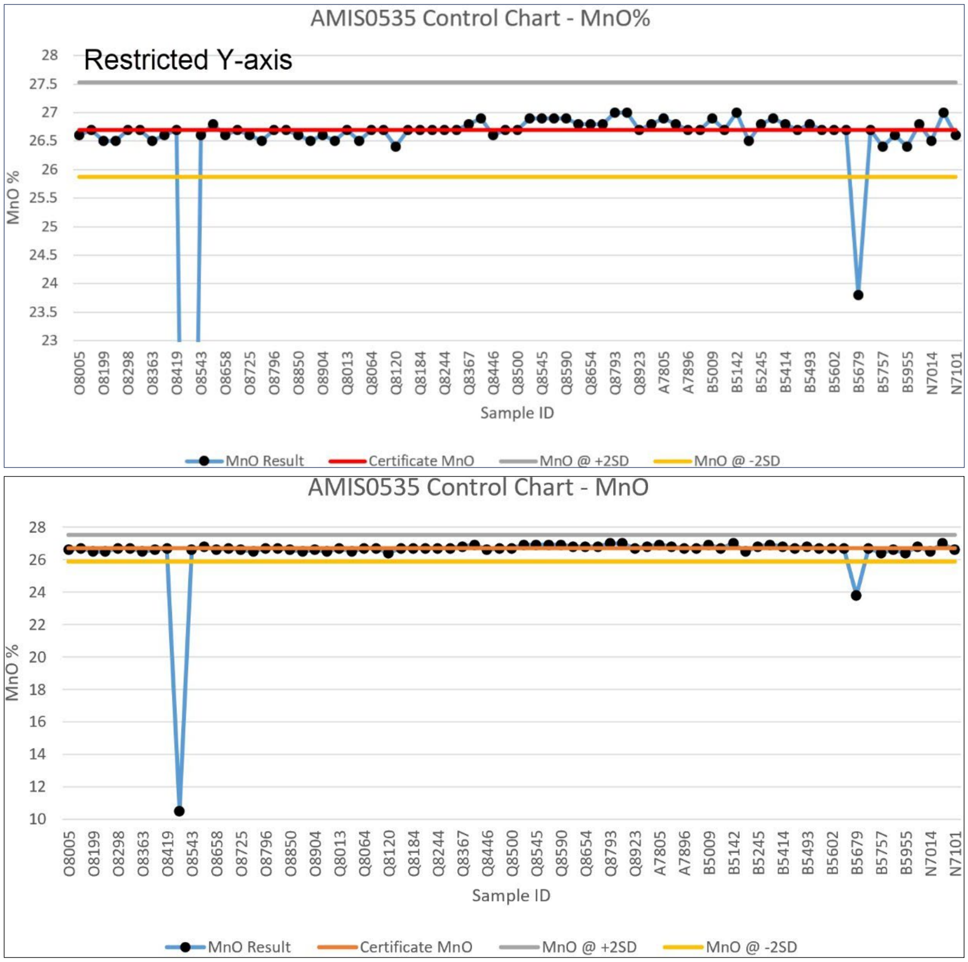
Source: SRK (2022c)

Figure 11.11 Certified reference material control plot for AMIS0533, 2020/21 reverse circulation drilling



Source: SRK (2022c)

Figure 11.12 Certified reference material control plot for AMIS0535, 2020/21 reverse circulation drilling



Source: SRK (2022c)

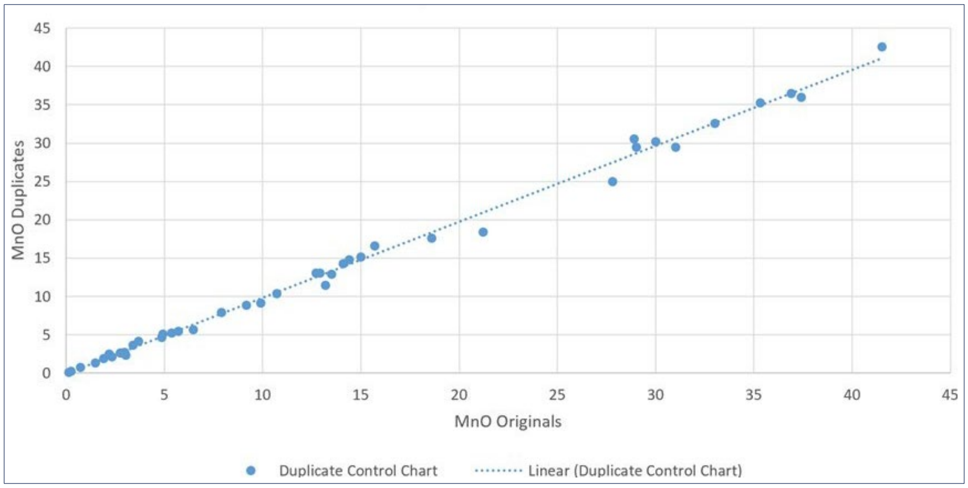
Overall, the two lower grade CRMs (AMIS0533 and AMIS0535) performed well, with the two AMIS0533 failures were likely due to sample swaps with AMIS0535 samples, and the same being true for one of the two AMIS0535 failures. The higher-grade AMIS0407 CRM shows some periods of systematic over-reporting of true grades by approximately 0.5% to 1.0% manganese oxide. However, given that most samples are within 2 standard deviations of the certified value for this CRM, it is not considered to be a material risk to the accuracy of the sample analyses during the period.

CRMs were assayed for aluminium oxide, calcium oxide, iron oxide, magnesium oxide, phosphorus pentoxide, and silicon dioxide. The CRM assays for these variables were all within 4 standard deviations of the certified value of the respective variables, except for samples previously identified as likely CRM swaps. None of the variables showed any significant, consistent bias.

Duplicates

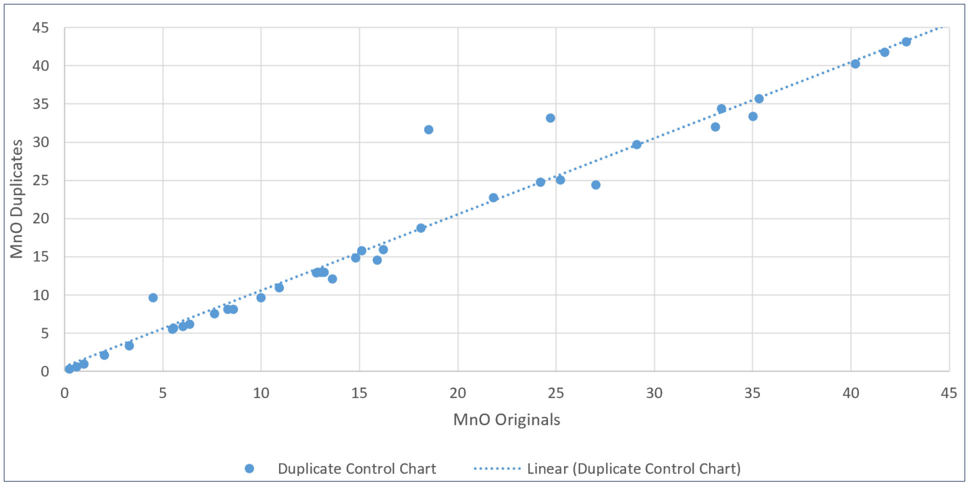
A total of 42 field and 40 pulp duplicates were inserted into the regular sample stream during the 2020/21 RC drilling campaign, representing insertion rates of approximately 1.3% and 1.2%, respectively. Overall, both duplicate sample types showed good precision (Figure 11.13 and Figure 11.14). As such, it is considered that the sub-sampling and analytical processes showed an acceptable degree of repeatability, and the precision of the assay results determined during this drilling campaign was appropriate for Mineral Resource estimation and reporting.

Figure 11.13 Field duplicate control plot, 2020/21 reverse circulation drilling



Source: SRK (2022c)

Figure 11.14 Pulp duplicate control plot, 2020/21 reverse circulation drilling

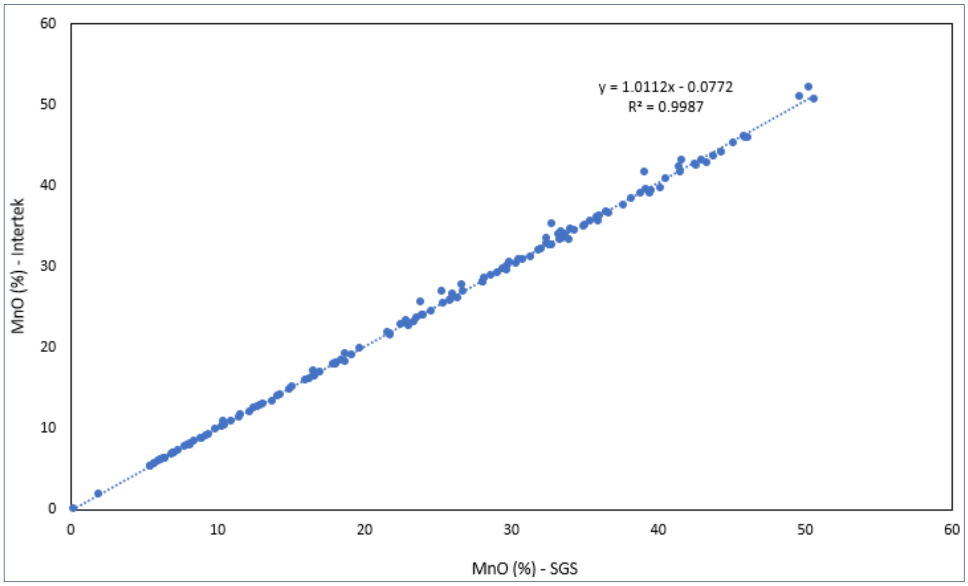


Source: SRK (2022c)

Umpire analyses

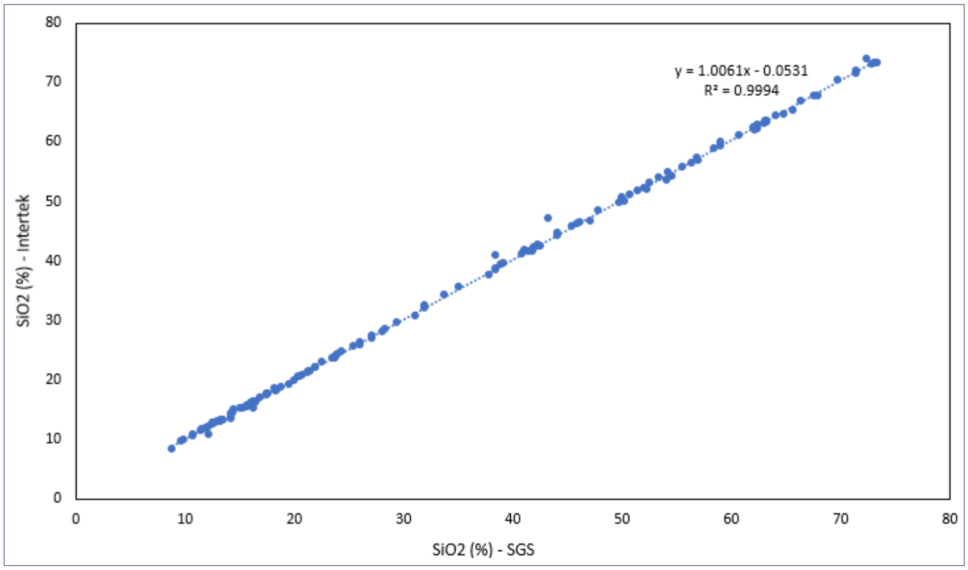
As an external control, a total of 139 duplicate samples from the 2020/21 RC drilling programme were sent to an umpire laboratory (Intertek Genalysis) for analysis. Overall, there is an excellent correlation (R^2 > 0.99 for the major oxides of interest) between the original (SGS Randfontein) and the umpire (Intertek Genalysis) analyses for each of the elements analysed (Figure 11.15 and Figure 11.16). Based on these results, there was no material issue with the accuracy, precision, or contamination of RC sample analyses conducted at SGS Randfontein during this period.

Figure 11.15 Umpire sample control plot for manganese oxide (%) analyses at the primary (SGS Randfontein) and umpire (Intertek Genalysis) laboratories



Source: SRK (2022c)

Figure 11.16 Umpire sample control plot for silicon dioxide (%) analyses at the primary (SGS Randfontein) and Umpire (Intertek Genalysis) laboratories



Source: SRK (2022c)

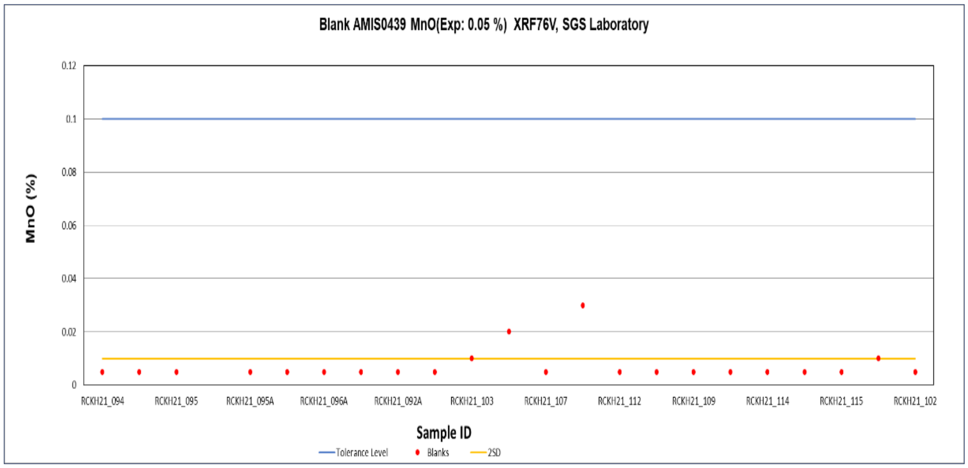
11.6.4 Reverse circulation Phase 4 southern extension area

During the southern extension area drilling campaign (28 RC holes), 118 QA/QC samples were inserted into the sample stream. The QA/QC samples included 21 blanks, 37 CRMs, 30 field duplicates, and 30 pulp duplicates, representing an overall QA/QC sample insertion rate of 13%.

Blanks

Giyani used AMIS0439 blank silica chips for blank material submission to the primary laboratory (SGS Randfontein) to monitor sample contamination. The results are shown in Figure 11.17 and show no blank samples exceeding the expected upper limit of 0.1% manganese oxide.

Figure 11.17 Blank sample (AMIS0439) control plot



Source: SRK (2022c)

Certified reference material

Three CRMs (AMIS0407/533/535), covering a grade range of approximately 24% to 47% manganese oxide, were inserted into the K.Hill Project southern extension area RC sample stream. CRM grades for the main analysed elements are provided in Table 11.5. The manganese oxide grades of the CRM are appropriate to the average grade of the mineralised K.Hill Project samples. As mentioned previously, inclusion of a lower-grade CRM at or around the Mineral Resource reporting grade (8% to 12% manganese oxide) would be a beneficial addition.

Table 11.5 Certified reference material values for manganese oxide

CRM	Certified value (%)	Two standard deviations (%)
AMIS0407	46.81	0.740
AMIS0533	23.97	0.555
AMIS0535	26.70	0.830

The submission and failure rates (above and below 3 standard deviations) of the three CRMs are summarised in Table 11.6, with performance illustrated in

Overall, the three CRMs performed moderately well, with a combined failure rate of 8% for manganese oxide, given that the majority of samples are within 2 standard deviations of the certified value for this CRM.

The CRMs were also assayed for aluminium oxide, calcium oxide, iron oxide, magnesium oxide, phosphorus pentoxide, and silicon dioxide. The assays for these variables are mostly within 4 standard deviations of the certified value of the respective variables; only a less than 2% failure rate for all elements was noted. None of the additional elements showed any significant, consistent bias.

Figure 11.18.

Table 11.6 Certified reference material performance for the Phase 4 southern extension area reverse circulation programme

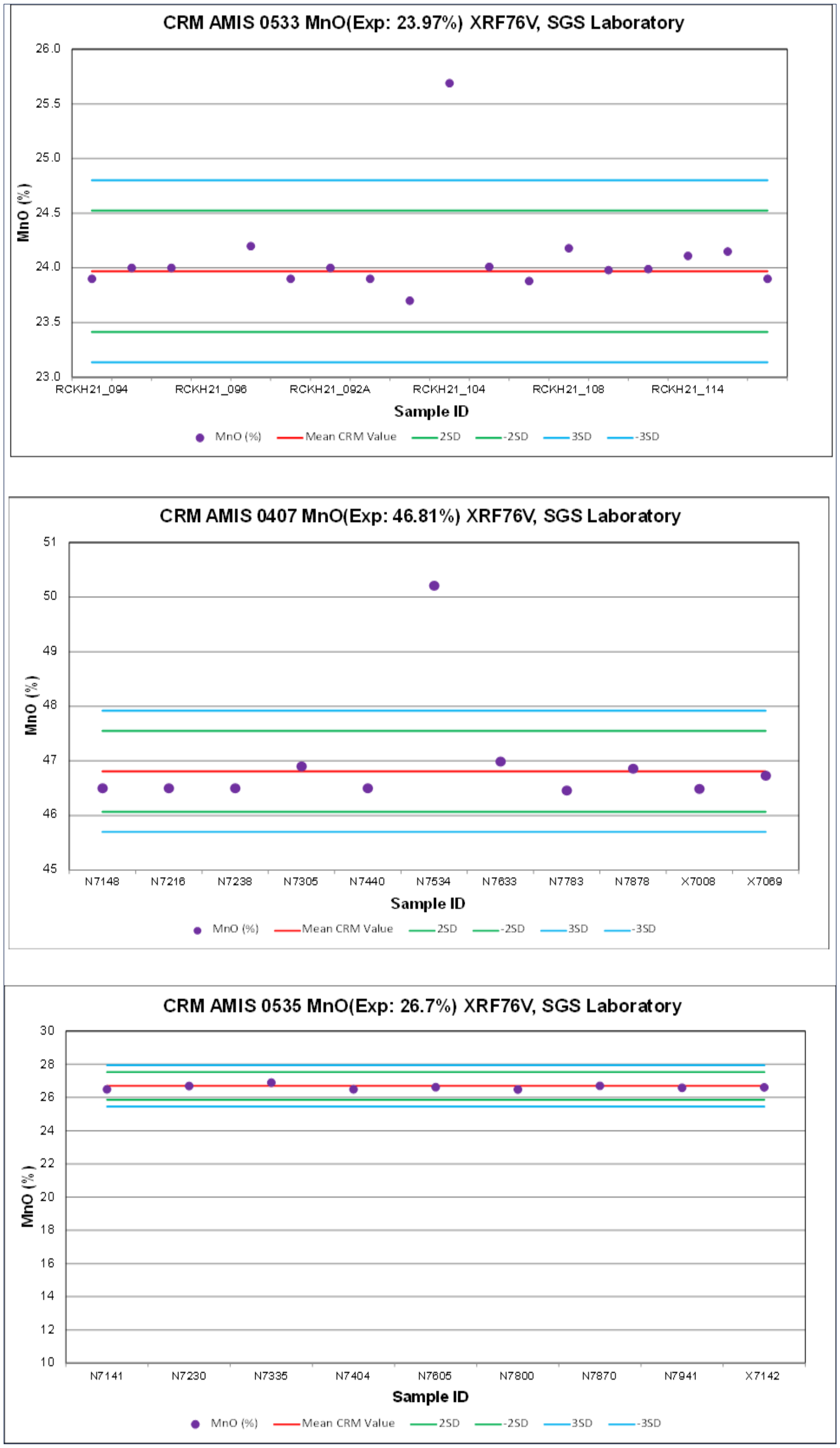
CRM	Number of submissions	Insertion rate (%)	Number of failures	Failure rate (%)
AMIS0407	11	1.2	1	9
AMIS0533	17	1.8	2	11
AMIS0535	9	1.0	0	0
Total	**37**	**4.0**	**3**	**8**

Overall, the three CRMs performed moderately well, with a combined failure rate of 8% for manganese oxide, given that the majority of samples are within 2 standard deviations of the certified value for this CRM.

The CRMs were also assayed for aluminium oxide, calcium oxide, iron oxide, magnesium oxide, phosphorus pentoxide, and silicon dioxide. The assays for these variables are mostly within 4 standard

deviations of the certified value of the respective variables; only a less than 2% failure rate for all elements was noted. None of the additional elements showed any significant, consistent bias.

Figure 11.18 Control plots for AMIS0535, AMIS0407 and AMIS0533, SGS Randfontein



Source: SRK (2022c)

Duplicates

A total of 30 field and 30 pulp duplicates were inserted into the regular sample stream during the southern extension area RC drilling campaign, representing insertion rates of approximately 3.3%. Overall, both duplicate sample types show good precision

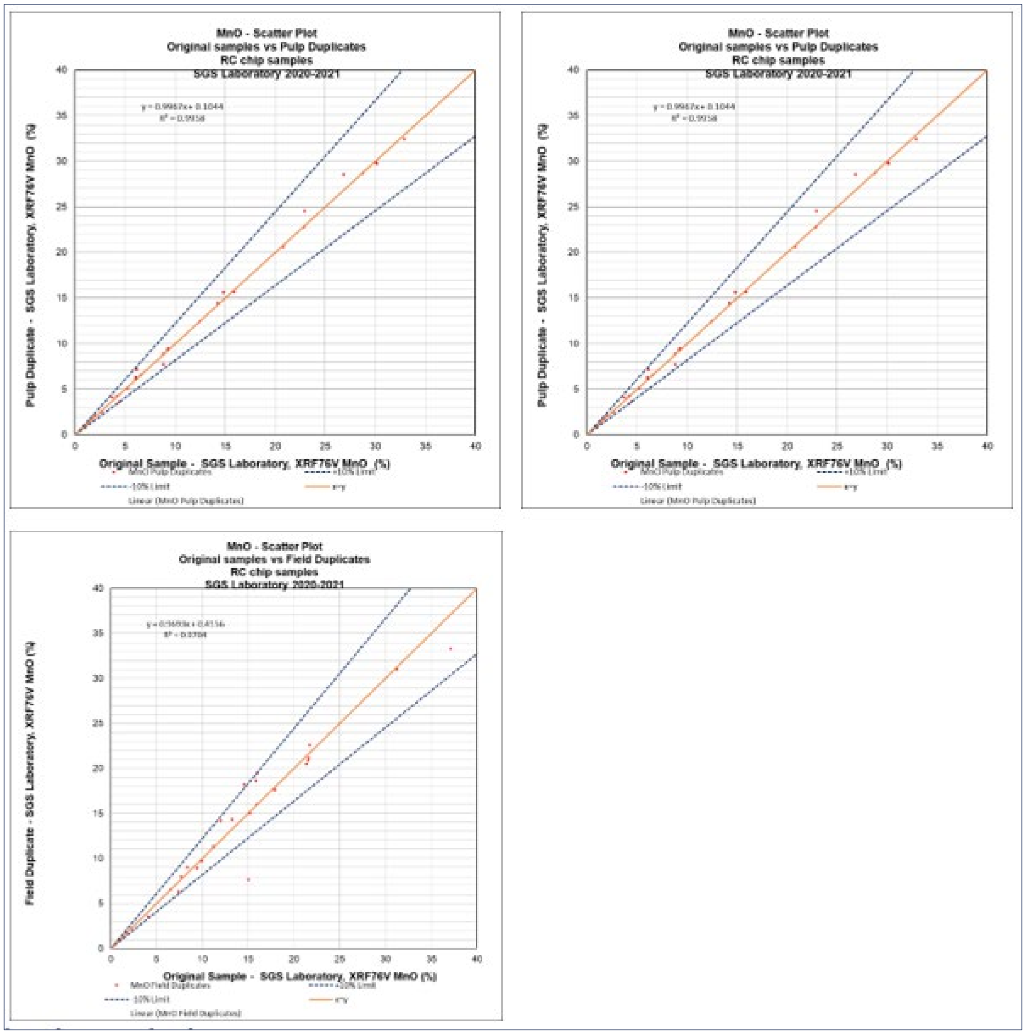
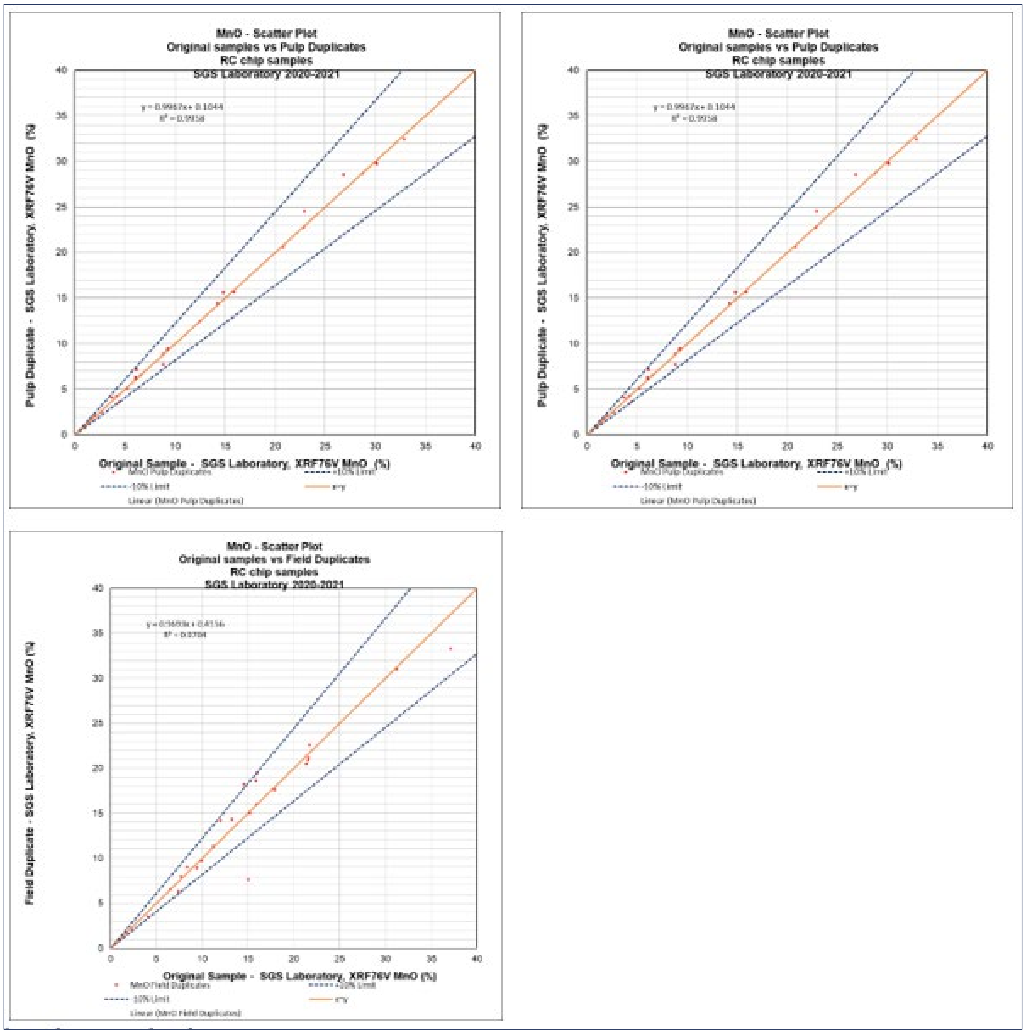


Figure 11.19). As such, it is considered that the sub-sampling and analytical processes show an acceptable degree of repeatability, and the precision of the assay results determined during this drilling campaign is appropriate for Mineral Resource estimation and reporting.

Figure 11.19 Field and pulp duplicate control charts, Phase 4 reverse circulation drilling

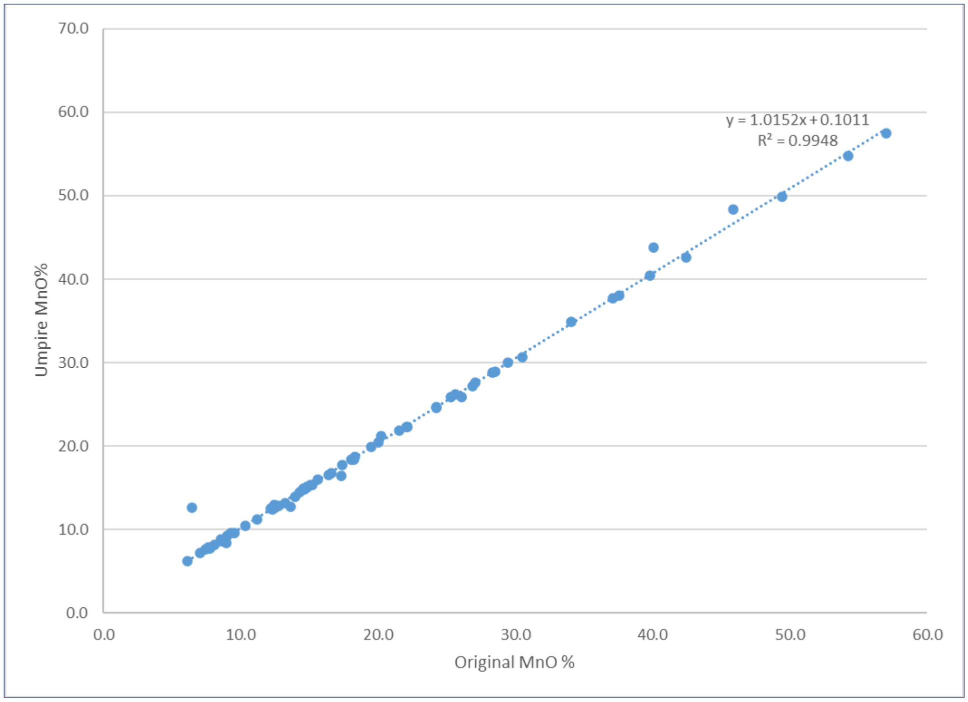


Source: SRK (2022c)

Umpire analysis

As an external control, a total of 68 duplicate samples from the drilling programme were sent to an umpire laboratory (SGS South Africa) for analysis. Overall, there is an excellent correlation ($R^2 > 0.99$ for manganese oxide) between original (SGS Randfontein) and umpire analyses (Figure 11.20).

Figure 11.20 Umpire results for manganese oxide, Phase 4



Source: CSA Global (2023)

11.6.5 Diamond drilling and reverse circulation drilling Phase 5 southern extension area

Blanks

Giyani used AMIS0681 blank silica chips for blank material submission to the primary laboratory (SGS Randfontein) to monitor sample contamination.

The blank sample analysis did not highlight any issues with sample contamination at the laboratory. Of the 89 blanks submitted, only 2 returned manganese oxide assays higher than three times the laboratory's lower detection limit. The certified values for AMIS0681 are shown in Table 11.7. The lower detection limit of 0.01292% was used to test manganese oxide performance in the blank. The upper limit was set at six times the standard deviation.

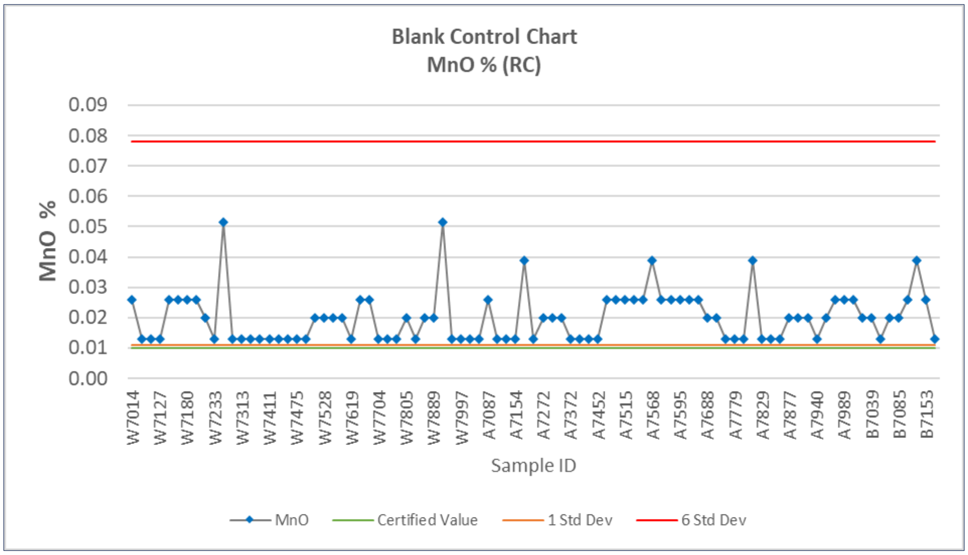
Table 11.7 Certified AMIS0681 values

Analyte (%)	Certified value (%)	Standard deviations (%)
MnO	0.01	0.00
Al_2O_3	1.06	0.12
SiO_2	97.04	0.44
Fe_2O_3	1.17	0.22

The other elements were included in the blank analyses. Overall, the performance is good. Figure 11.21 and Figure 11.28 show the results for the RC and DD holes, respectively.

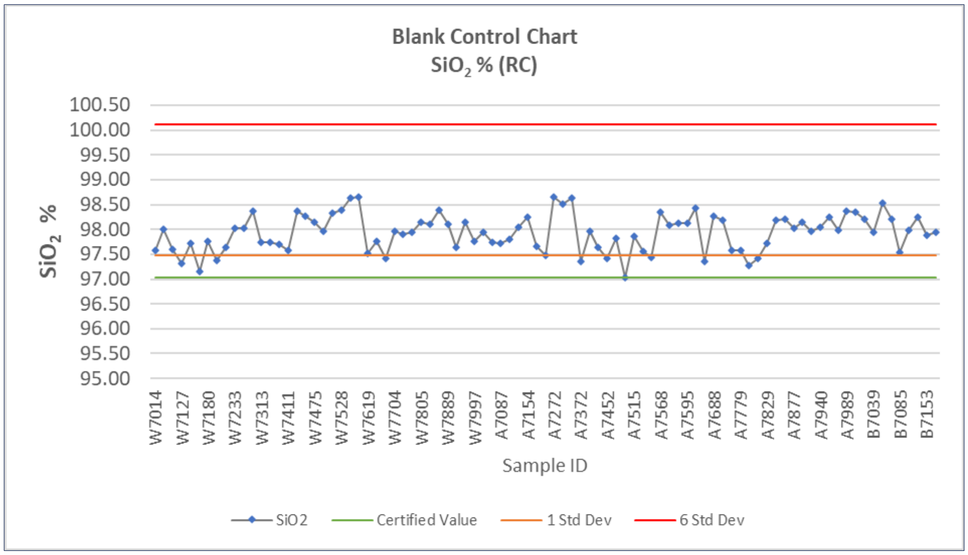
Based on this threshold, the performance of the blank sample is acceptable, and cross-contamination at the preparation or analytical laboratory facilities is not considered a material issue.

Figure 11.21 Blank control graph for manganese oxide (%) for reverse circulation holes



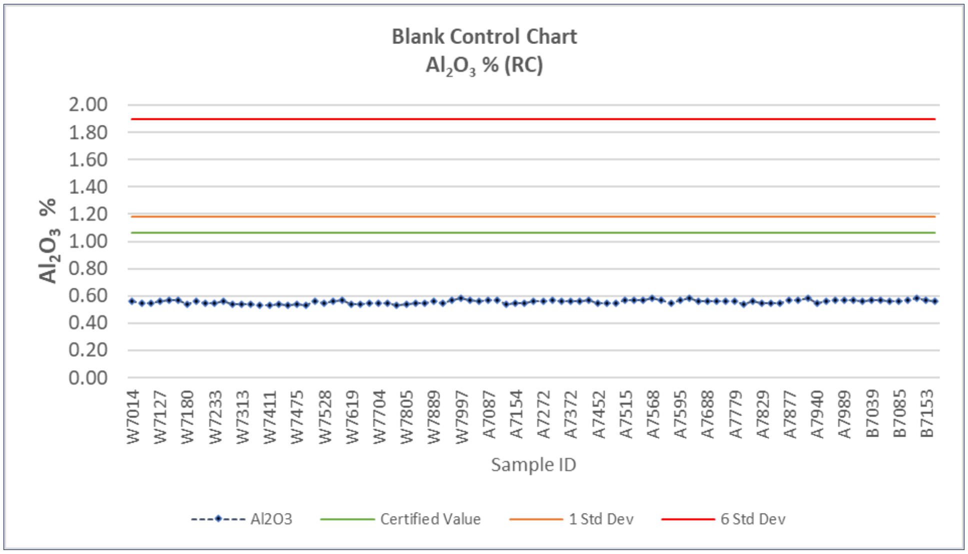
Source: CSA Global (2023)

Figure 11.22 Blank control graph for silicon dioxide (%) for reverse circulation holes



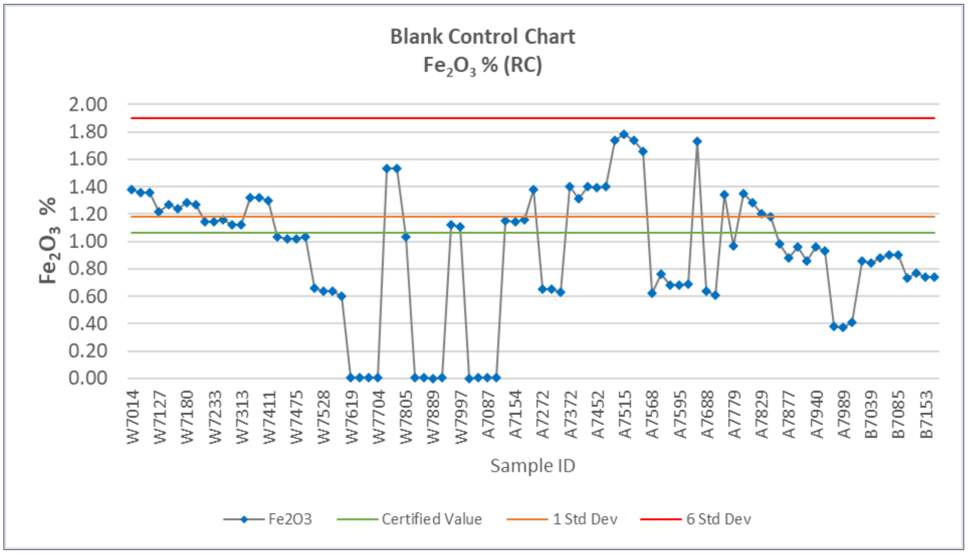
Source: CSA Global (2023)

Figure 11.23 Blank control graph for aluminium dioxide (%) for reverse circulation holes



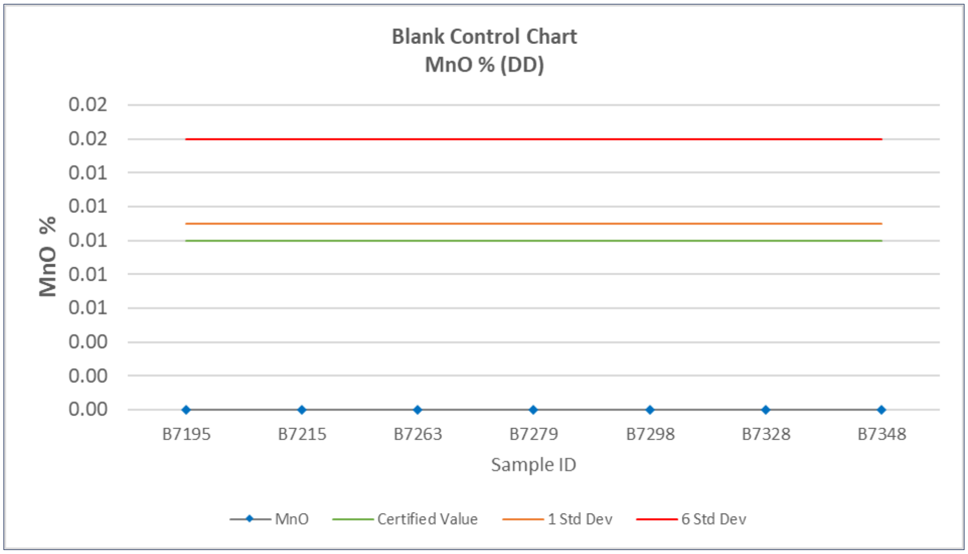
Source: CSA Global (2023)

Figure 11.24 Blank control graph for iron oxide (%) for reverse circulation holes



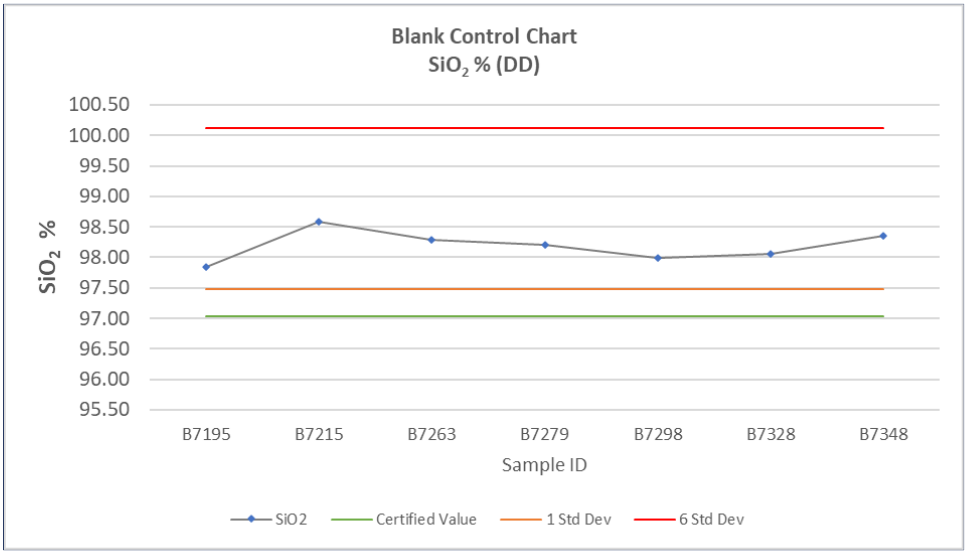
Source: CSA Global (2023)

Figure 11.25 Blank control graph for manganese oxide (%) for diamond drilling holes



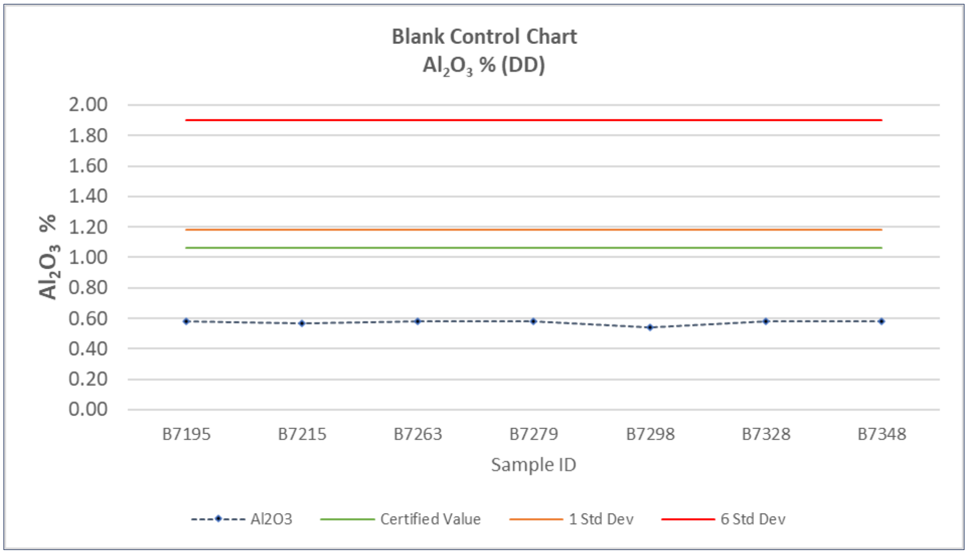
Source: CSA Global (2023)

Figure 11.26 Blank control graph for silicon dioxide (%) for diamond drilling holes



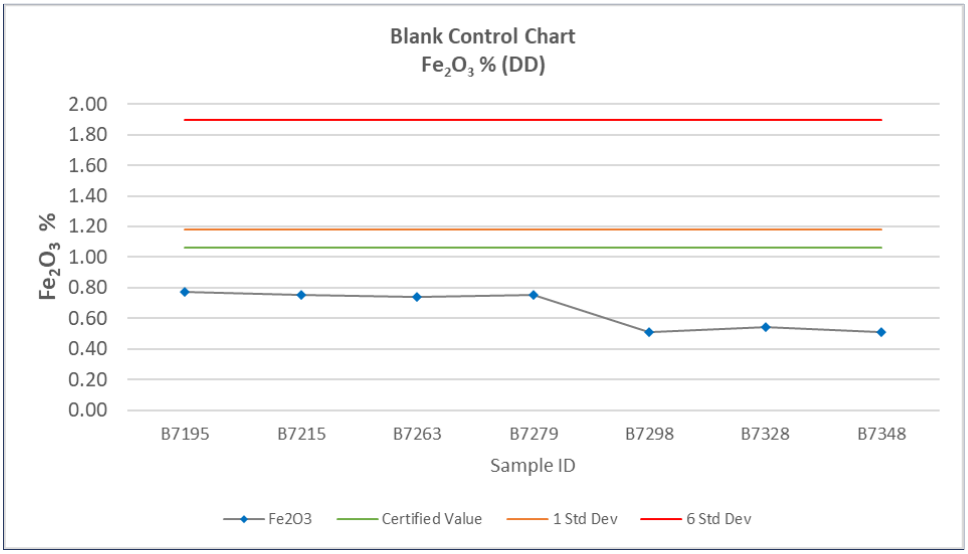
Source: CSA Global (2023)

Figure 11.27 Blank control graph for aluminium dioxide (%) for diamond drilling holes



Source: CSA Global (2023)

Figure 11.28 Blank control graph for iron oxide (%) for diamond drilling holes



Source: CSA Global (2023)

Certified reference material

To monitor assay accuracy, CRMs were inserted randomly into the sampling stream. Commercially available CRMs were purchased from AMIS in South Africa. These were delivered as homogeneous pulp material with certified concentrations and expected standard deviations of the elements of interest. CRMs were supplied in heat-sealed, airtight, pulp packets, not requiring any further sample preparation. CRM pulps utilized in the K.Hill Project were not matrix-matched to the host rock lithologies.

Samples of three separate CRMs (AMIS0407/533/535), covering a grade range of approximately 24% to 47% manganese oxide, were inserted into the K.Hill Project's drillhole sample stream. The manganese oxide grades of the CRM are appropriate to the average grade of the mineralised K.Hill Project samples. As mentioned previously, inclusion of a lower-grade CRM at or around the Mineral Resource reporting grade (8% to 12% manganese oxide) would be a beneficial addition.

The submission and failure rates of the three CRMs are summarised in Table 11.8 and Table 11.9. Failure rate is above or below 3 standard deviations.

Table 11.8 Summary of certified reference material insertion

CRM	Number of submissions	Insertion rate (%)
AMIS0407	26	1.4
AMIS0533	47	2.5
AMIS0535	34	1.8
Total	**107**	**5.7**

Table 11.9 Summary of certified reference material performance

CRM	Element	No. of samples	Mean Value	CV	Mean Bias (%)	Failure (%)
AMIS0407	MnO	26	46.99	0.002	0.4	0
	Al_2O_3	26	5.55	0.007	0.6	0
	SiO_2	26	0.29	0.034	1.4	0
	Fe_2O_3	26	5.32	0.211	-11.8	4
AMIS0533	MnO	47	24.39	0.002	1.8	0
	Al_2O_3	47	6.98	0.006	0.1	0
	SiO_2	47	29.70	0.002	-0.5	0
	Fe_2O_3	47	0.76	0.429	-16.4	19
AMIS0535	MnO	34	27.06	0.004	1.3	0
	Al_2O_3	34	7.47	0.006	-0.2	0
	SiO_2	34	26.53	0.003	0.7	0
	Fe_2O_3	34	0.65	0.365	-15.7	9

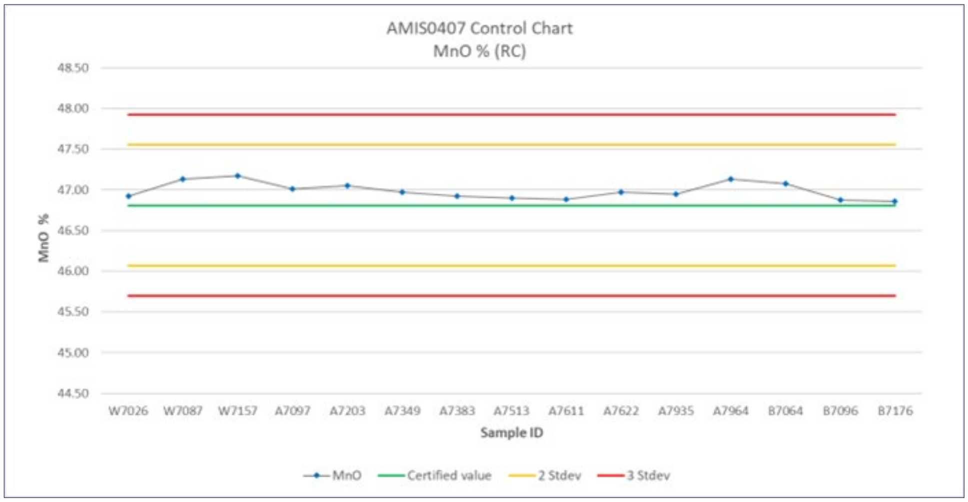
Notes:
CV - coefficient of variation

The analyses of CRMs indicate good analytical accuracy and precision, with all but two assay results plotting within the three times upper and lower standard deviation thresholds. The majority of reported values for manganese oxide (%) for CRMs AMIS0407/0535/0533 are slightly higher than the expected reference value, indicating a slightly high assay bias. Silicon dioxide shows a slightly low bias; the other elements show satisfactory results. Iron oxide shows the most failures within the CRMs and should be monitored with ongoing exploration programs.

Given the majority samples are within 2 standard deviations of the certified value, the QP does not consider there to be a material risk to the accuracy of the sample analyses for this drill programme.

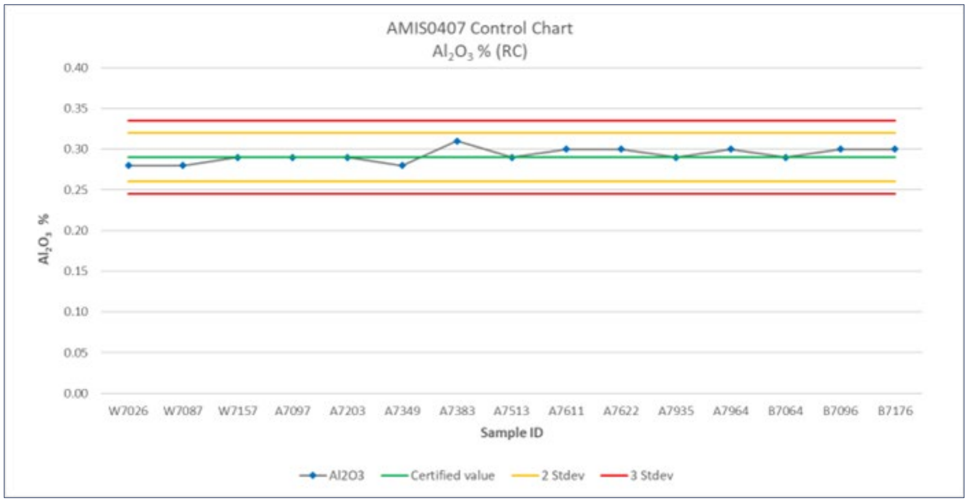
The CRM assays for the main variables are mostly within 3 standard deviations of the certified value for the respective variables. None of the elements showed any significant, consistent bias. The performance charts for CRMs are split between the RC and DD holes shown in Figure 11.29 to Figure 11.48 (no DD holes for AMIS0407).

Figure 11.29 Control chart for manganese oxide (%) for AMIS0407 reverse circulation holes



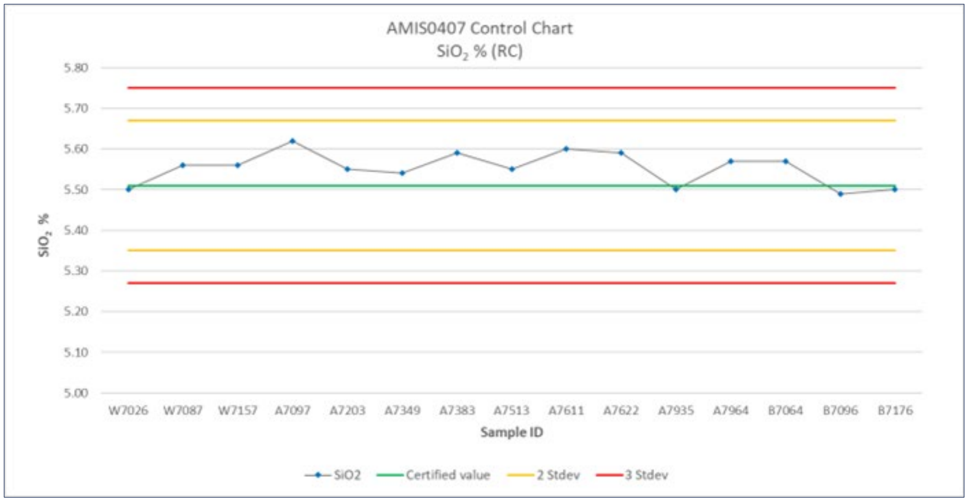
Source: CSA Global (2023)

Figure 11.30 Control chart for aluminium dioxide (%) for AMIS0407 reverse circulation holes



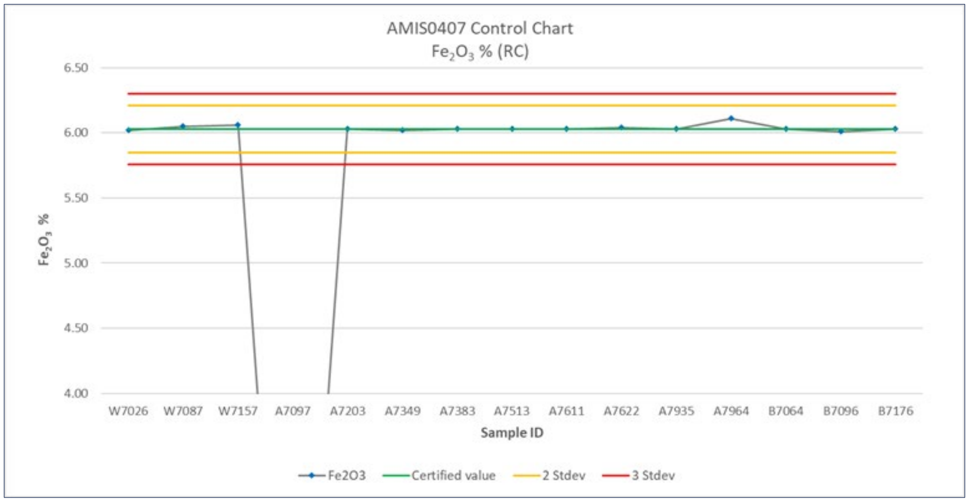
Source: CSA Global (2023)

Figure 11.31 Control chart for silicon dioxide (%) for AMIS0407 reverse circulation holes



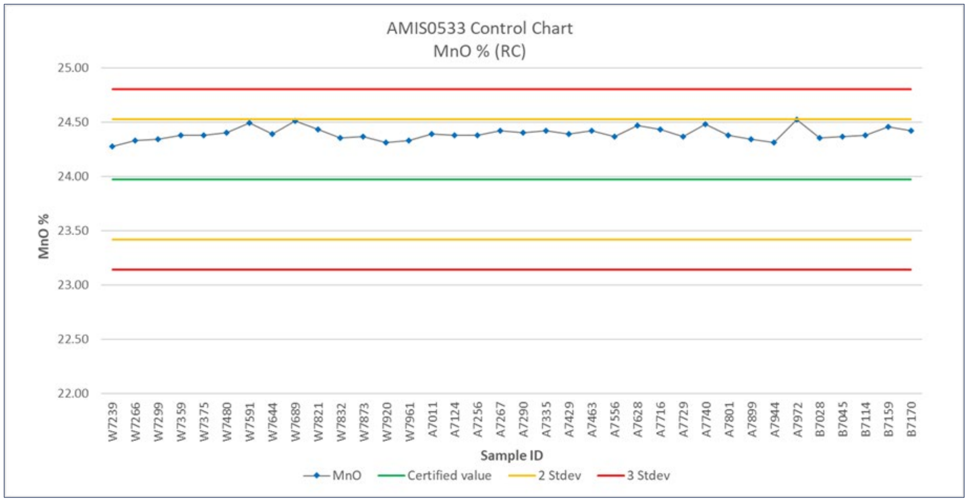
Source: CSA Global (2023)

Figure 11.32 Control chart for iron oxide (%) for AMIS0407 reverse circulation holes



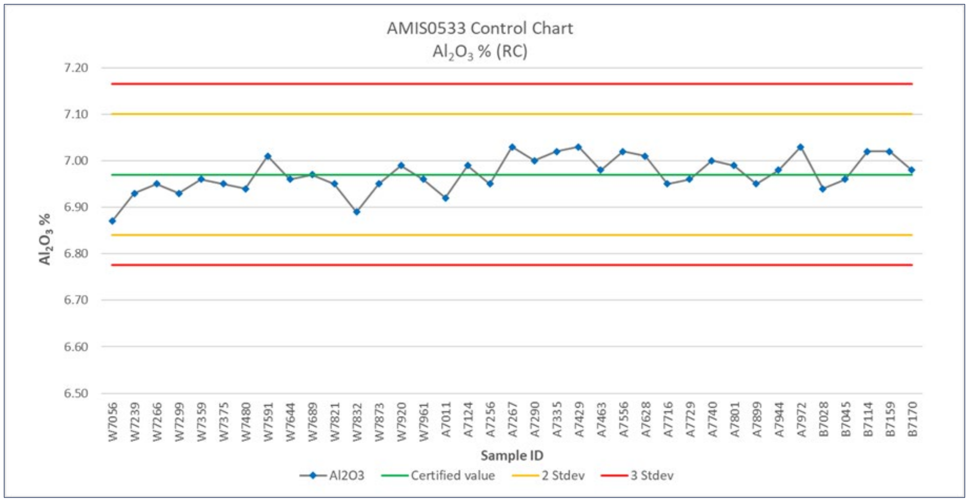
Source: CSA Global (2023)

Figure 11.33 Control chart for manganese oxide (%) for AMIS0533 reverse circulation holes



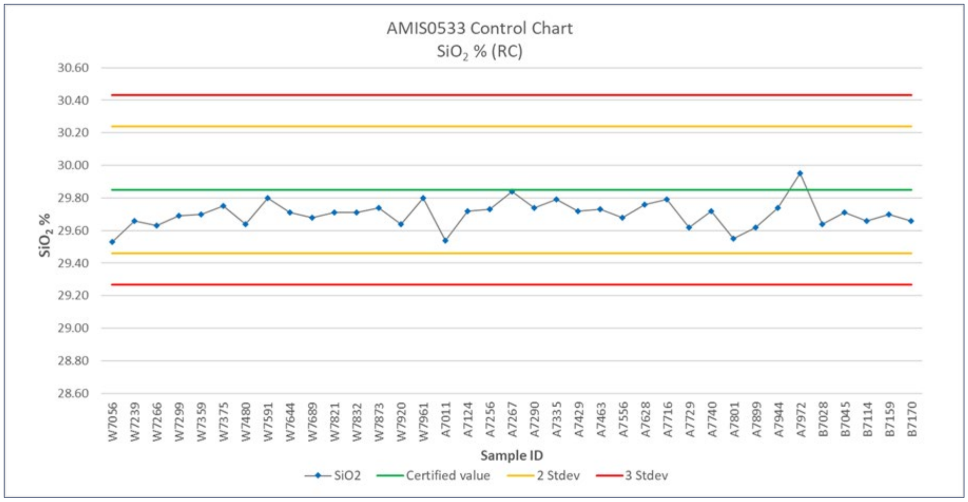
Source: CSA Global (2023)

Figure 11.34 Control chart for aluminium oxide (%) for AMIS0533 reverse circulation holes



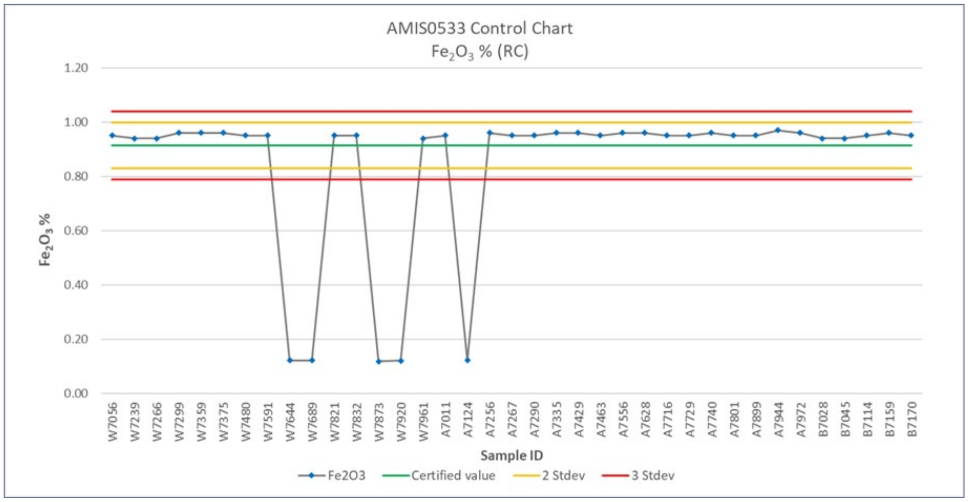
Source: CSA Global (2023)

Figure 11.35 Control chart for silicon dioxide (%) for AMIS0533 reverse circulation holes



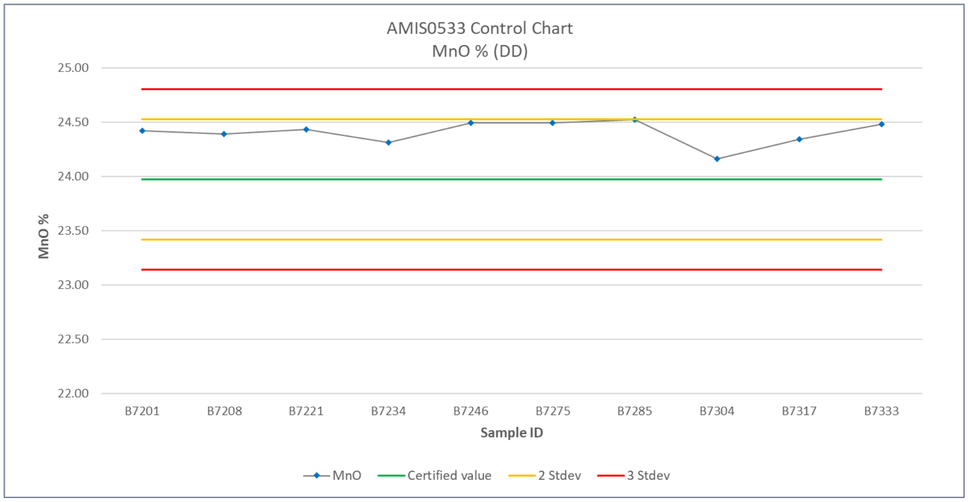
Source: CSA Global (2023)

Figure 11.36 Control chart for iron oxide (%) for AMIS0533 reverse circulation holes



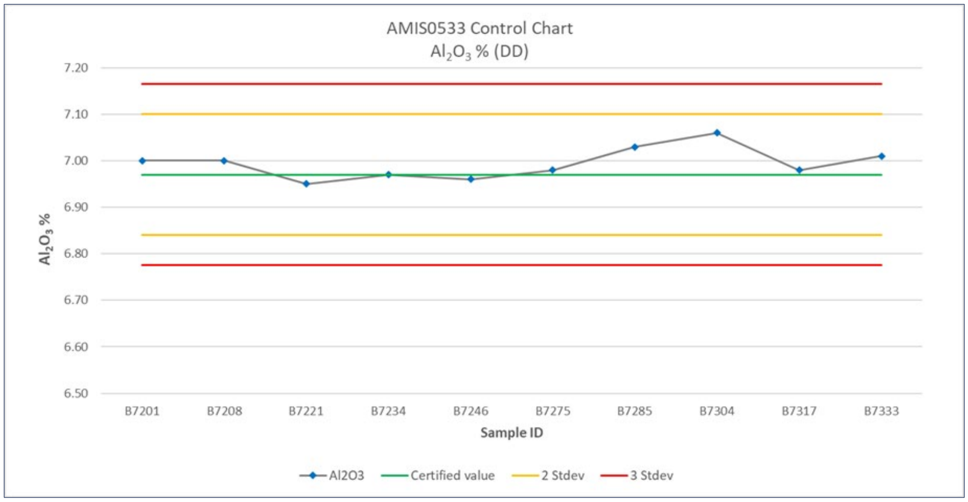
Source: CSA Global (2023)

Figure 11.37 Control chart for manganese oxide (%) for AMIS0533 diamond drilling holes



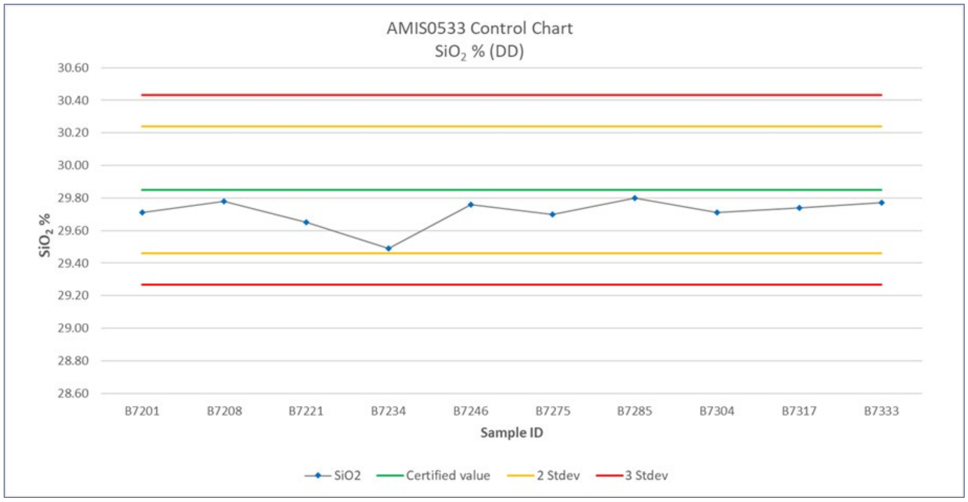
Source: CSA Global (2023)

Figure 11.38 Control chart for aluminium oxide (%) for AMIS0533 diamond drilling holes



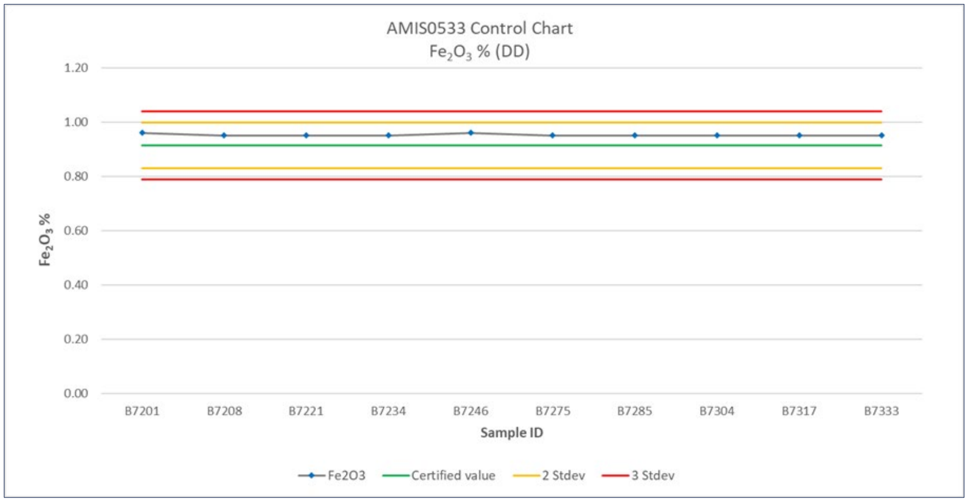
Source: CSA Global (2023)

Figure 11.39 Control chart for silicon dioxide (%) for AMIS0533 diamond drilling holes



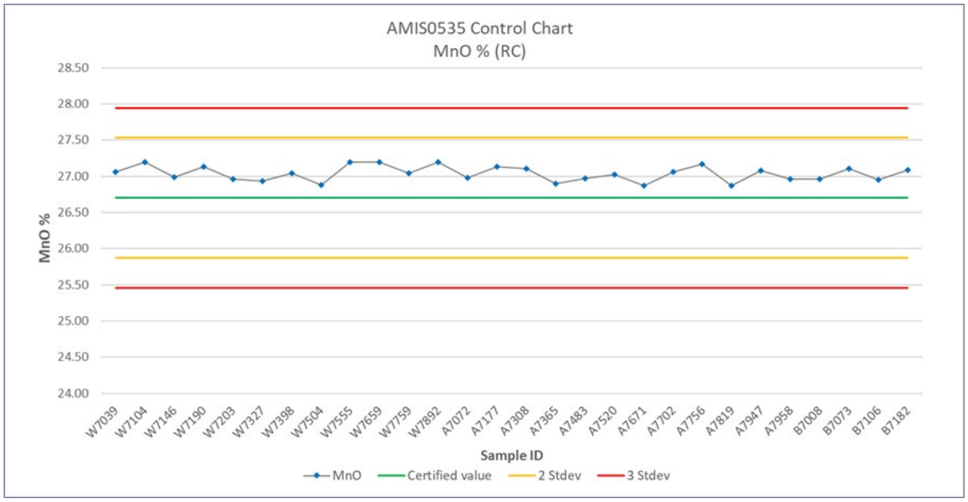
Source: CSA Global (2023)

Figure 11.40 Control chart for iron oxide (%) for AMIS0533 diamond drilling holes



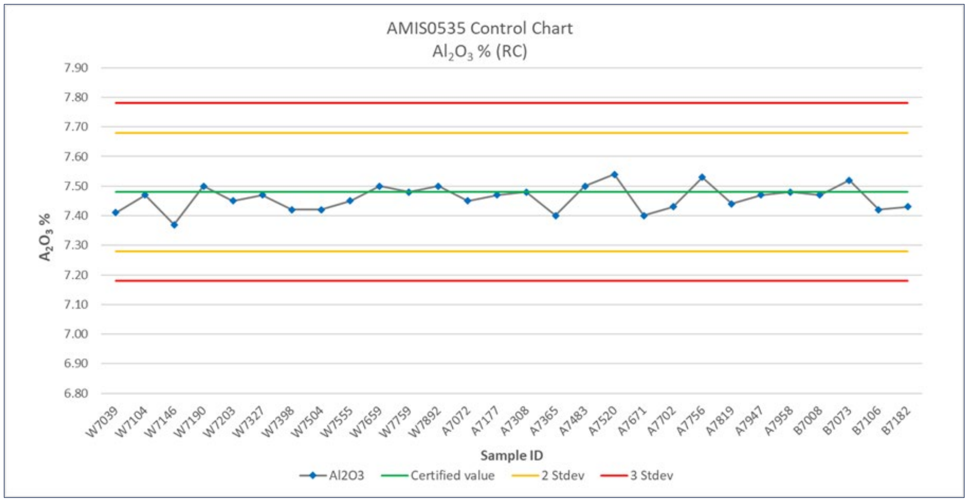
Source: CSA Global (2023)

Figure 11.41 Control chart for manganese oxide (%) for AMIS0535 reverse circulation holes



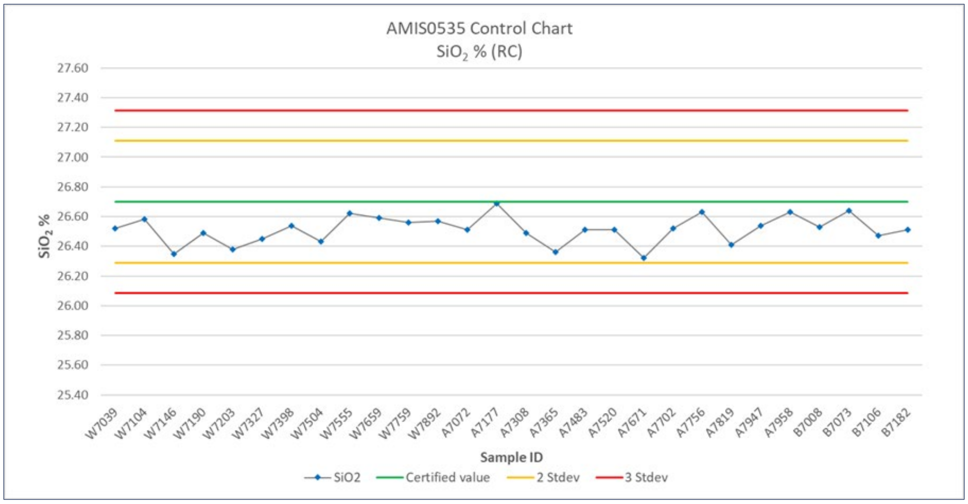
Source: CSA Global (2023)

Figure 11.42 Control chart for aluminium dioxide (%) for AMIS0535 reverse circulation holes



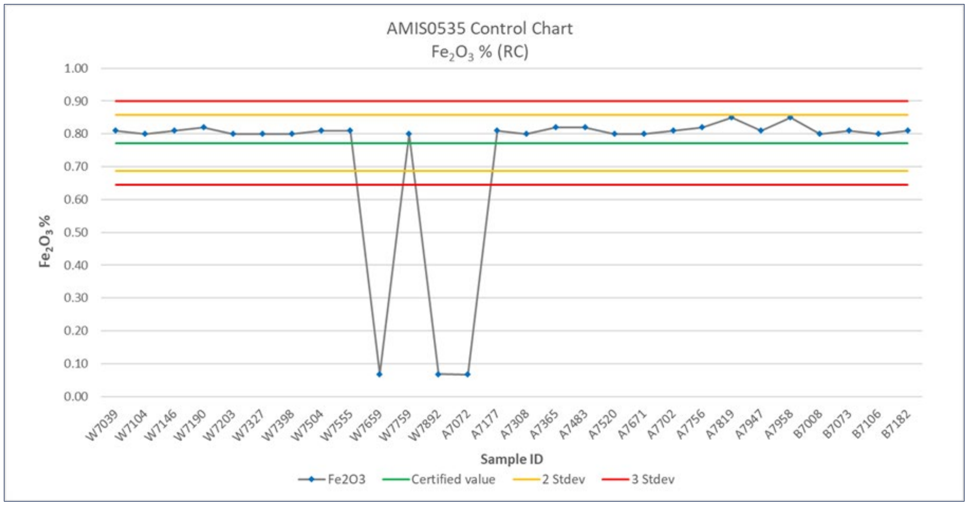
Source: CSA Global (2023)

Figure 11.43 Control chart for silicon dioxide (%) for AMIS0535 reverse circulation holes



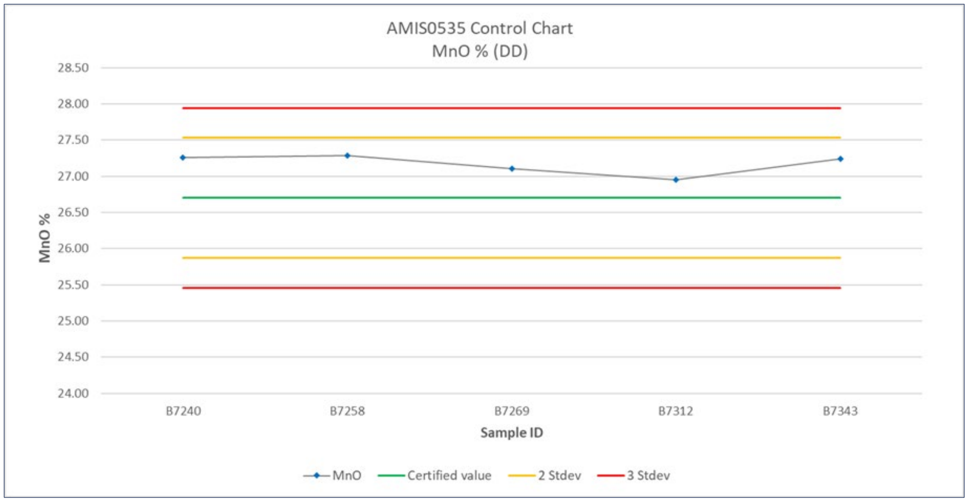
Source: CSA Global (2023)

Figure 11.44 Control chart for iron oxide (%) for AMIS0535 reverse circulation holes



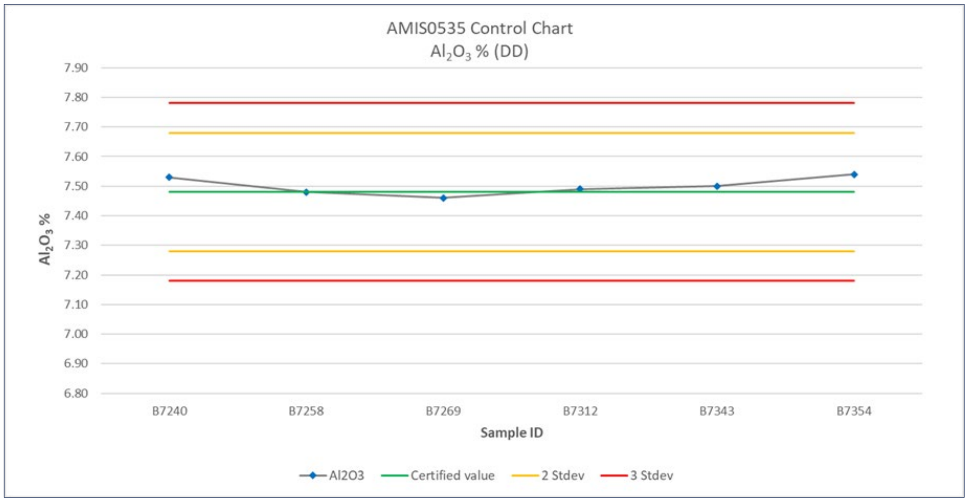
Source: CSA Global (2023)

Figure 11.45 Control chart for manganese oxide (%) for AMIS0535 diamond drilling holes



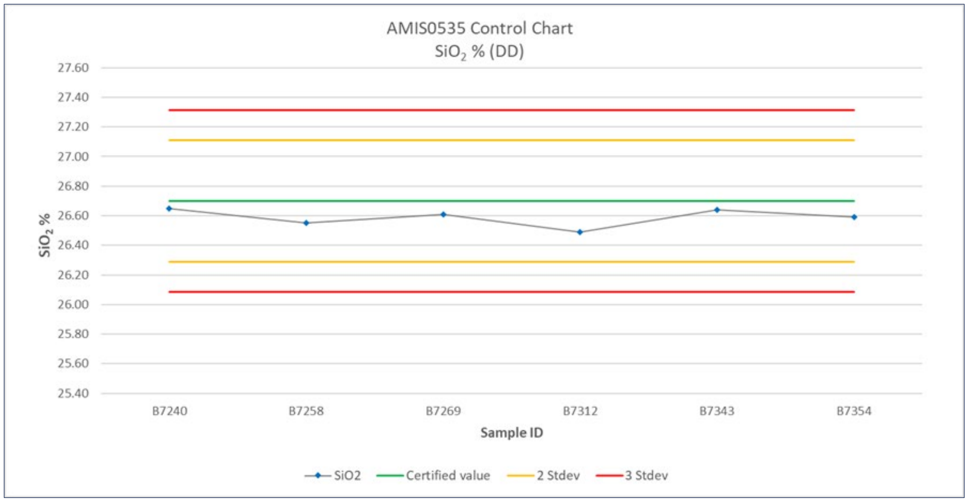
Source: CSA Global (2023)

Figure 11.46 Control chart for aluminium dioxide (%) for AMIS0535 diamond drilling holes



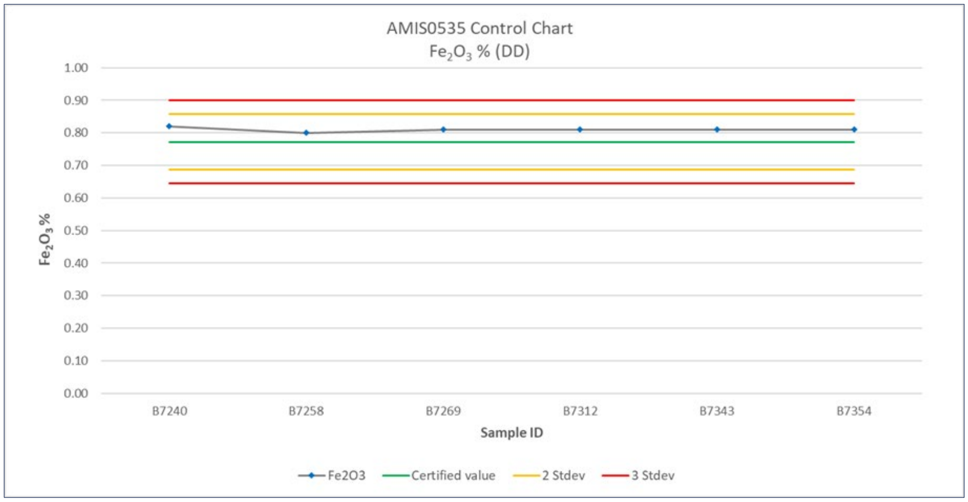
Source: CSA Global (2023)

Figure 11.47 Control chart for silicon dioxide (%) for AMIS0535 diamond drilling holes



Source: CSA Global (2023)

Figure 11.48 Control chart for iron oxide (%) for AMIS0535 diamond drilling holes

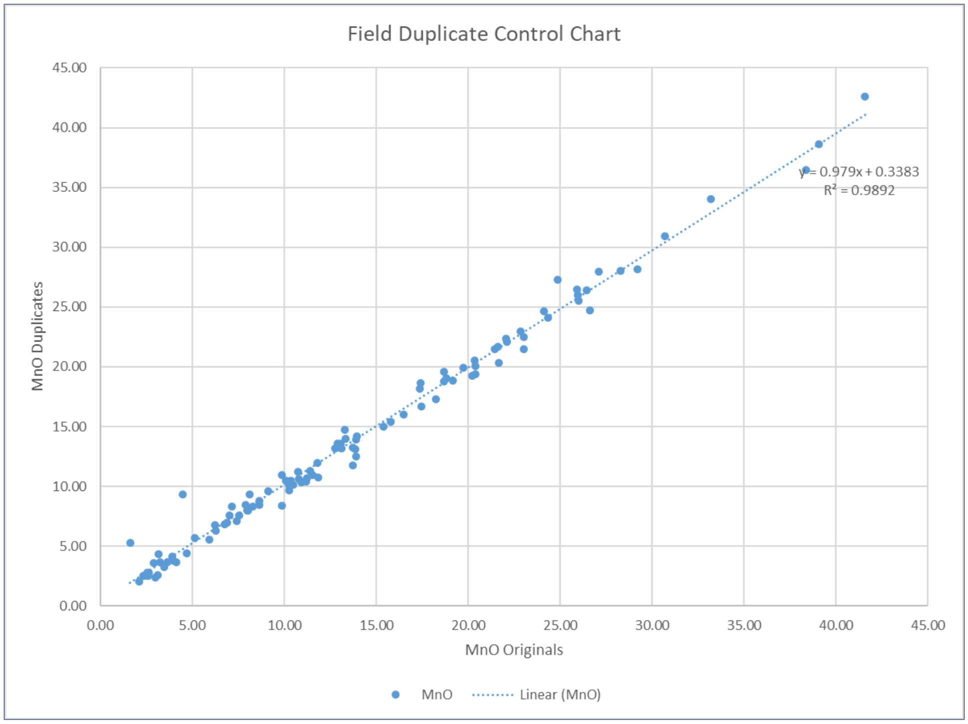


Source: CSA Global (2023)

Duplicates

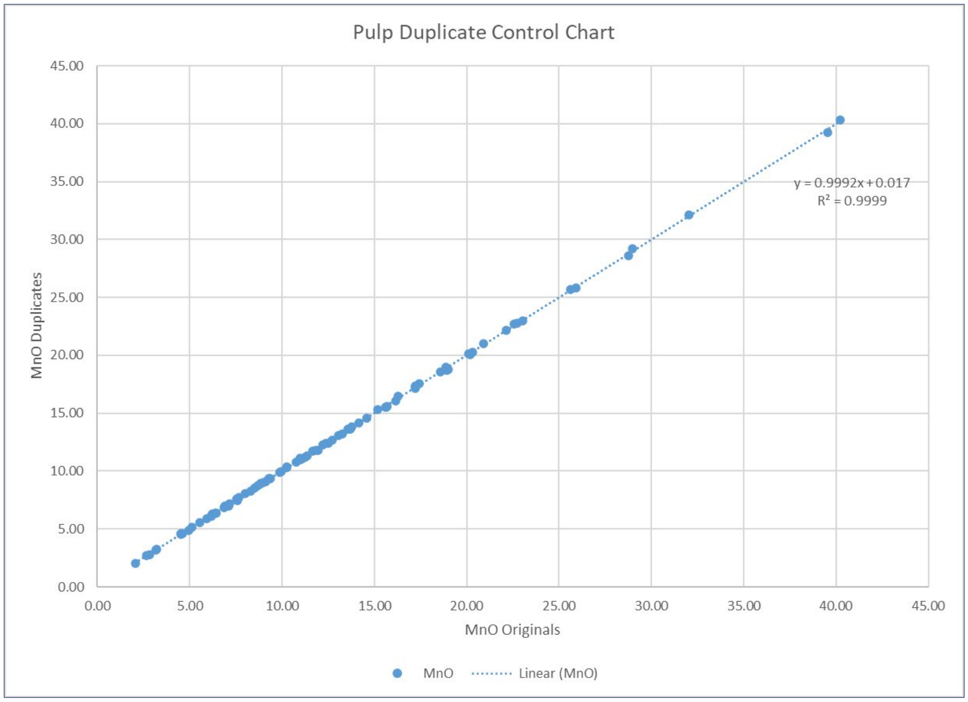
A total of 108 field and 96 pulp duplicates were inserted into the regular sample stream during the drilling campaign, representing insertion rates of approximately 5.8% and 5.1%, respectively. For both the pulp and coarse reject duplicates, a reasonable correlation is observed, indicating acceptable sample homogeneity and analytical precision. (Figure 11.49 and Figure 11.50).

Figure 11.49 Field duplicate control plot



Source: CSA Global (2023)

Figure 11.50 Pulp duplicate control plot

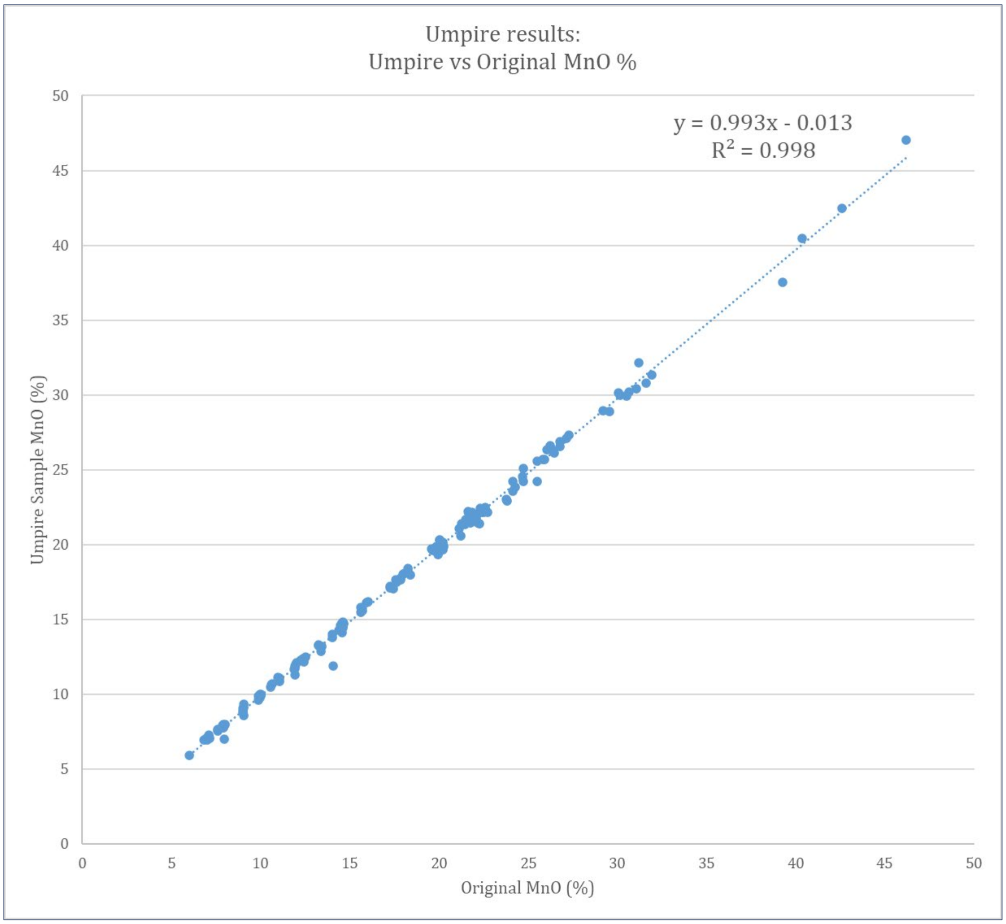


Source: CSA Global (2023)

Umpire analyses

As an external control, a total of 155 duplicate samples from the drilling programme were sent to an umpire laboratory (SGS South Africa) for analysis. Overall, there is an excellent correlation ($R^2 > 0.99$ for the major oxides of interest) between original (SGS Randfontein) and umpire analyses (Figure 11.51). Based on these results, the QP does not consider there to have been a material issue with the accuracy, precision, nor contamination of sample analyses conducted at SGS Randfontein during this period.

Figure 11.51 Umpire results for manganese oxide (Phase 5)



Source: CSA Global (2023)

11.6.6 Comments

The results of the QA/QC measures applied for the K.Hill Project do not indicate significant contamination and demonstrate a high degree of accuracy and precision. The check laboratory assays confirm the primary laboratory assays within close limits.

Sampling and sample preparation methodologies are aligned with standard industry practice. Assay methodology is appropriate for the type of mineralisation.

The QA/QC protocols employed by Giyani are aligned with industry standard practice:

- Blank performance is satisfactory.
- CRM performance is satisfactory and spans a grade range appropriate for the deposit. Some evidence of negative bias is present with two of the CRMs for iron oxide, but this is not considered material.
- The umpire laboratory results are acceptable.

The QP has not completed an independent check or visit to observe the sample preparation and analysis at SGS Randfontein. That said, the results of the QA/QC process do not highlight any significant concerns in the quality of the assay data used for the K.Hill Project MRE.

12 DATA VERIFICATION

12.1 Mineral Resource estimate site visit and data evaluation

The MRE QP, Mr. Anton Geldenhuys, MEng, MGSSA, PrSciNat, conducted a site visit for Mineral Resources from 20^{th} to 21^{st} April 2023 . During the trip, the MRE QP visited the following sites:

- K.Hill Project area and historical workings
- core processing facility
- Otse and Lobatse deposits
- Giyani office (Gaborone)

12.1.1 Site visit

K.Hill Project area and historical workings

The MRE QP accessed the K.Hill Project area by vehicle and explored on foot. Several outcrops and historic mining excavations, both surface and underground, were observed (Figure 12.1 and Figure 12.2).

Figure 12.1 Terrain and vegetation at the K.Hill Project; note the higher elevation of the hill relative to the dwellings in the distance

Source: CSA Global

Figure 12.2 One of the historical open pit and underground access locations at the K.Hill Project

Source: CSA Global

Collar locations of four drillholes were verified in the field (Figure 12.3):

- RCKH20_028
- DDKH18_011
- RCKH22_119
- RCKH22_132

Figure 12.3 Verified drillhole collar locations at the K.Hill Project

Source: CSA Global

Core processing facility

The core processing facility was inspected with regards to core storage and quality. Drilling was not taking place at the time of the visit; therefore, physical sampling of core was not observed.

Drill core and RC drilling chips from four holes were inspected relative to the original geological logs and digital geological logs contained within the supplied database for the following holes:

- DDKH22_038
- DDKH21_029
- RCKH21_115
- RCKH21_110

All logging data in the original and digital logs were verified by means of cross checks against the two log types and the core.

During the inspection of the four holes by the MRE QP, assays contained in the database were checked relative to geological logging and the core for signs of mineralisation. Mineralisation was evident in the core, and in all instances, mineralisation occurring in the core was evident in the recorded logging and assays.

Four geological logs in the database were verified relative to the original logs and drill core. The assays in the database were verified relative to geological logs and visual inspection of the physical core.

Core trays are labelled and stacked in an orderly fashion at the facility (Figure 12.4). Plastic core trays were used in earlier drilling phases; however, metal core trays are currently used.

Figure 12.4 Core processing facility showing stacking and storage of core trays; core trays in foreground laid out for Qualified Person inspection

Source: CSA Global

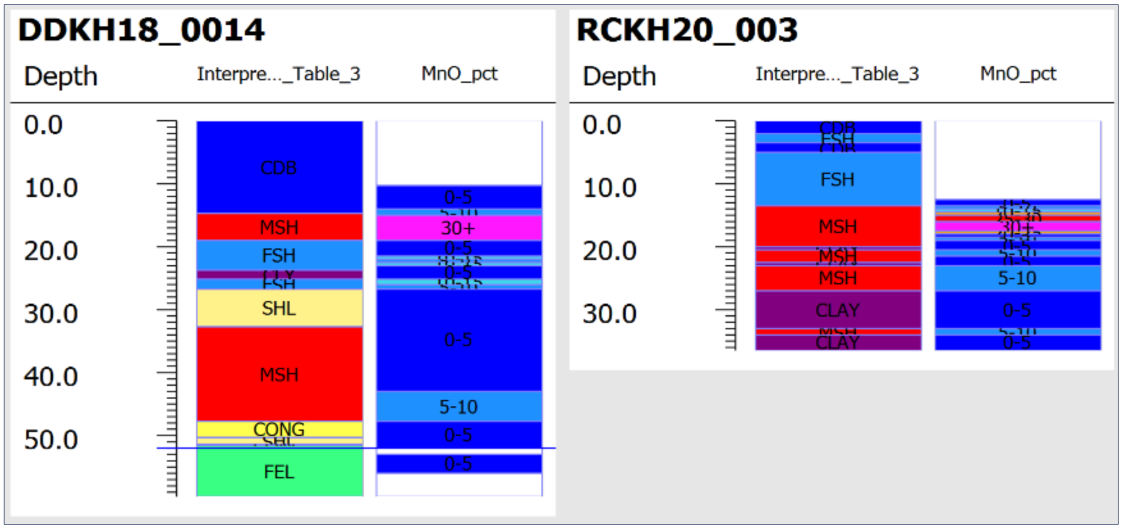
12.1.2 Database verification and validation

Twin drilling

In 2022, SRK undertook an analysis of the twin drilling completed by Giyani, comparing Phase 1 DD holes against the Phase 2 RC holes. CSA Global updated the study with newly acquired twin data. Twin holes allow for an analysis of the conformity of grade and logging between the DD and RC drilling.

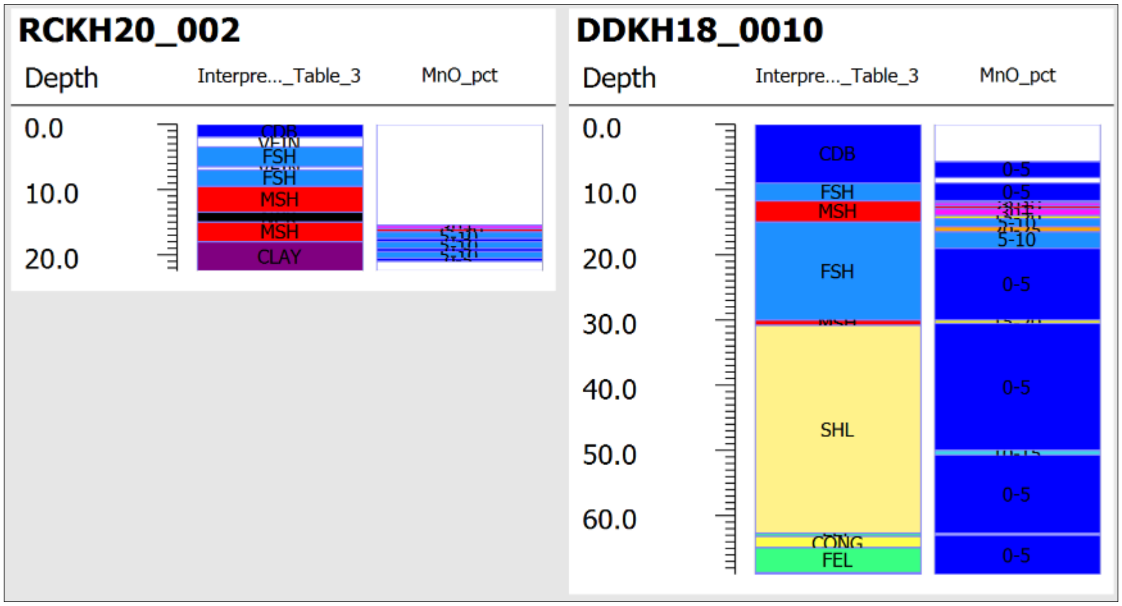
Twin plots showing analytical grade for manganese oxide as well as logged geology are presented in Figure 12.5 to Figure 12.8. The analysis suggests that the assays derived from RC drilling and DD show similar location, thickness, and grade across the mineralised zones, but that geological logging differs between the RC drilling and DD. Consequently, when using the RC drilling to construct mineralisation wireframes, CSA Global relied more on the assay data (where available) to guide the boundary locations of the wireframes, over the geological logging. Figure 12.9 shows a QQ plot to compare the RC and DD holes. There is a slight bias in the higher grades between the RC and DD holes. This is not significant and can be explained by various factors, including sampling methodology differences.

Figure 12.5 Twin plot showing logged geology and assayed manganese oxide grade for twinned drillholes DDKH18_0014 and RCKH20_003



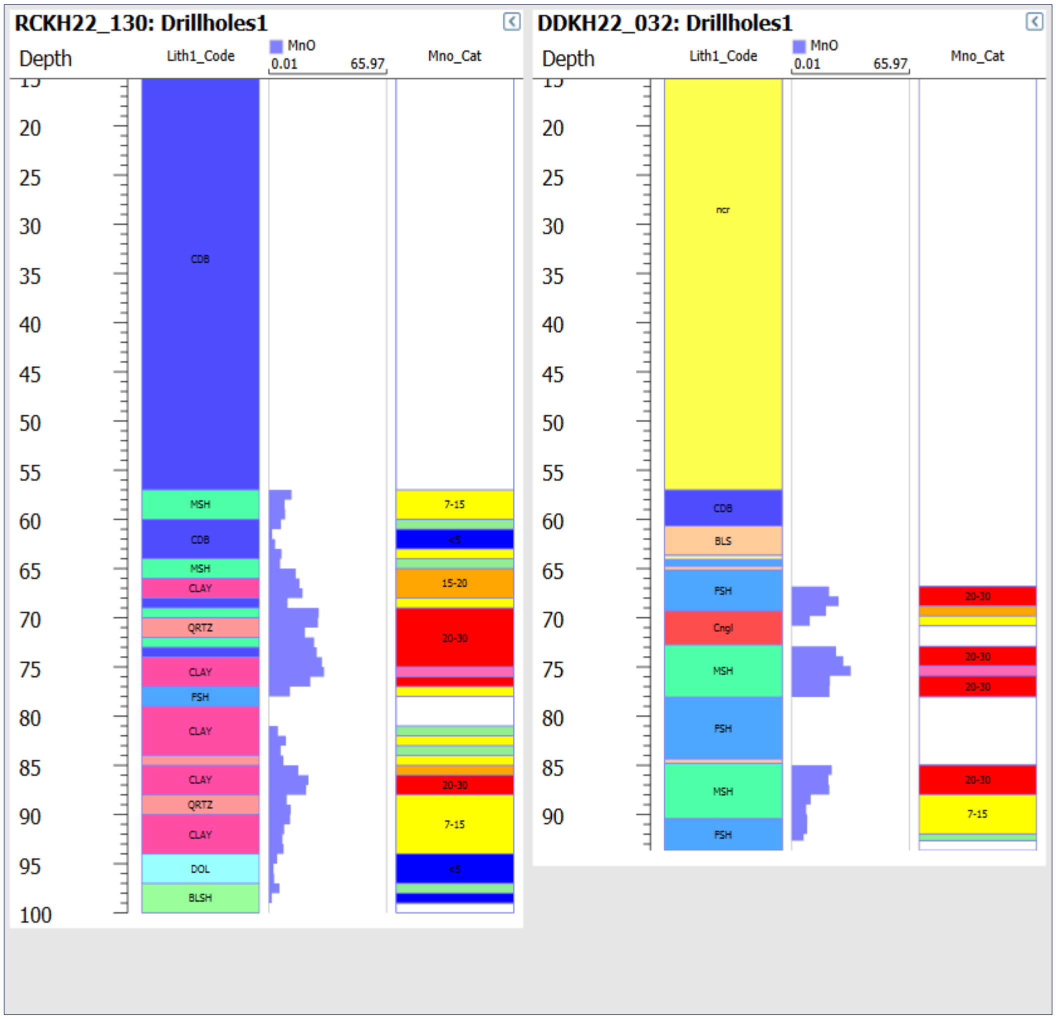
Source: SRK (2022)

Figure 12.6 Twin plot showing logged geology and assayed manganese oxide grade for twinned drillholes DDKH18_0010 and RCKH20_002



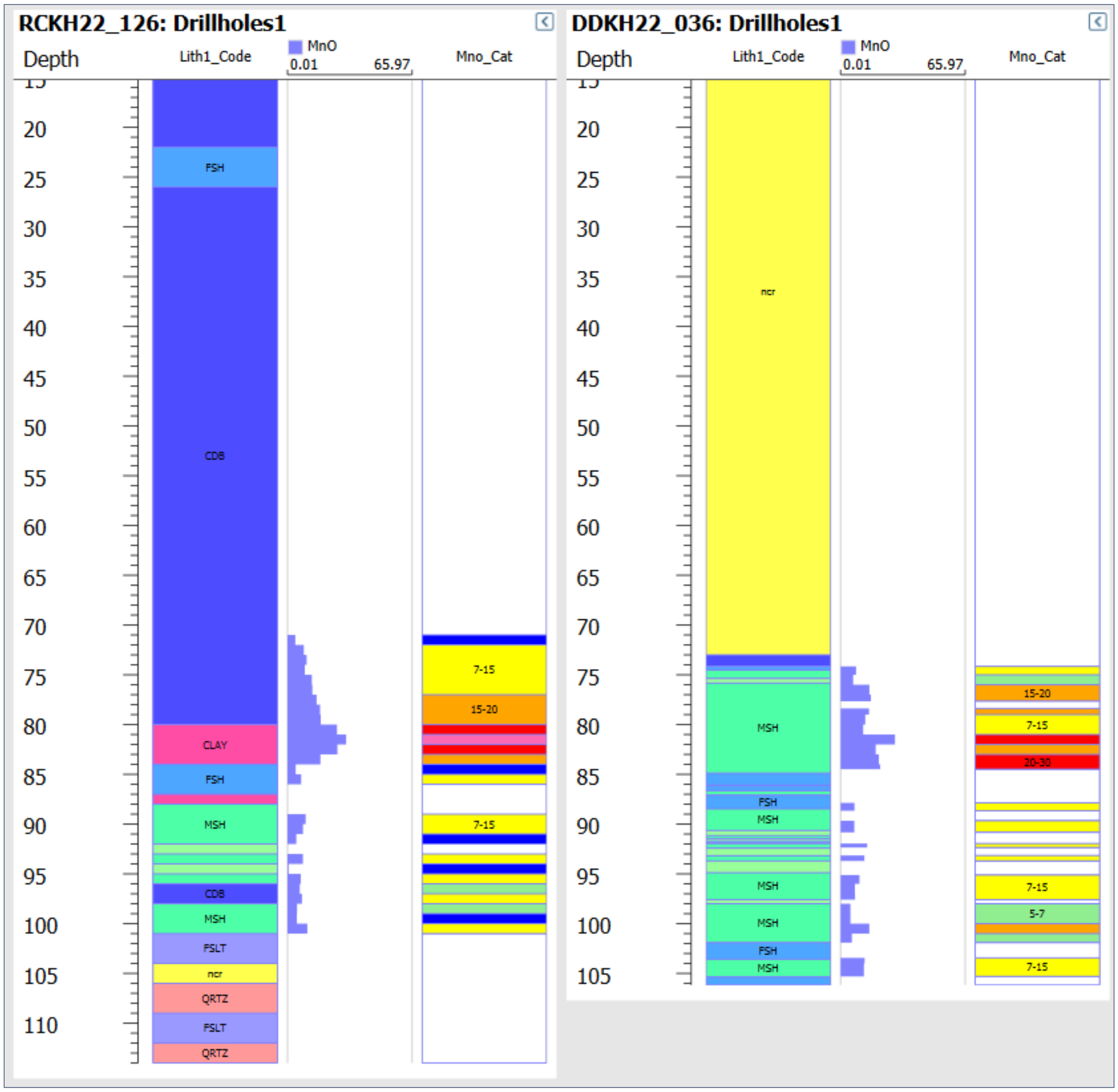
Source: SRK (2022)

Figure 12.7 Twin plot showing logged geology and assayed manganese oxide grade for twinned drillholes DDKH22_0032 and RCKH22_130



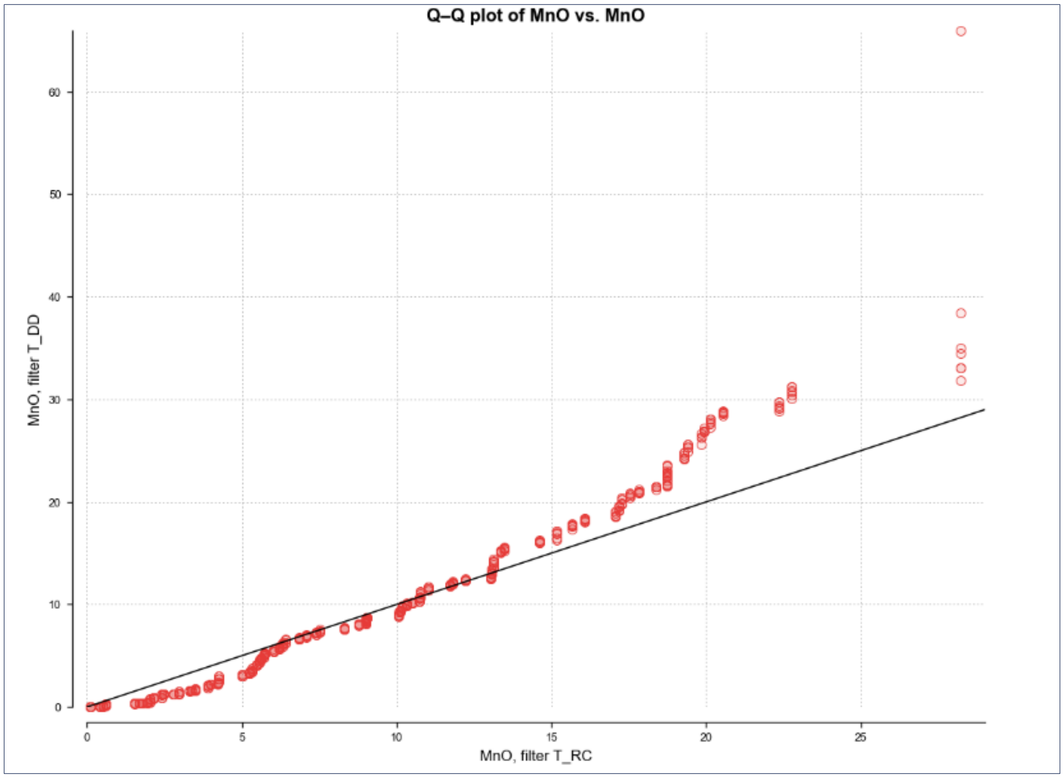
Source: CSA Global

Figure 12.8 Twin plot showing logged geology and assayed manganese oxide grade for twinned drillholes DDKH22_0036 and RCKH22_126



Source: CSA Global

Figure 12.9 QQ plot showing the manganese oxide grade between the twinned reverse circulation and diamond drilling holes

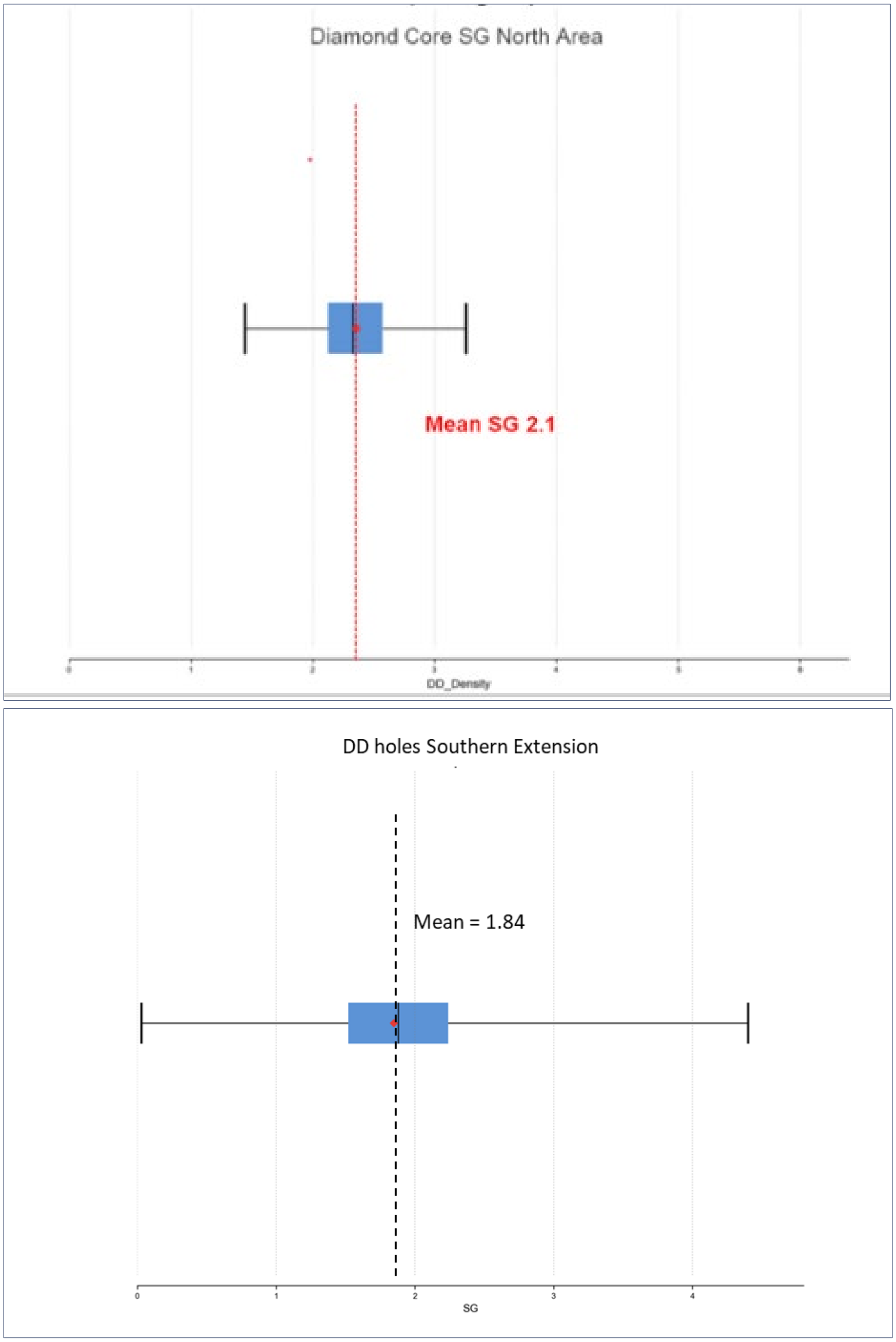


Source: CSA Global

Density comparison between reverse circulation and diamond drilling

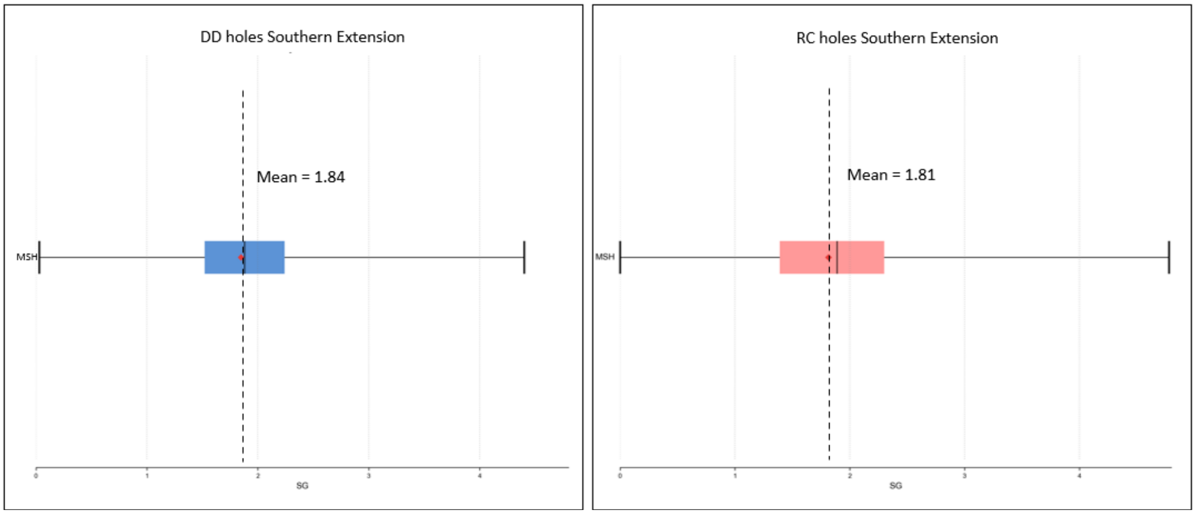
On average, the specific gravity (SG) measurements for the north have a typical density of 2.1 g/cm^3 compared to 1.8 g/cm^3 for the southern extension area (Figure 12.10). Statistically, a comparison of the RC densities with that of the diamond core densities for the MSH horizon in the southern area shows very similar average values (Figure 12.11). This provides a level of confidence for both the RC and DD SG values. Additionally, a comparison of RC- and DD-derived densities for the northern part of the deposit shows average values of around 2.0 gm/cc for both the DD and RC samples (1.91 RC vs. 2.3 DD).

Figure 12.10 Comparison of diamond drilling core specific gravities for the northern and southern extension areas manganiferous shale



Source: CSA Global

Figure 12.11 Comparison of reverse circulation vs. diamond drilling specific gravity determinations for the southern extension area manganiferous shale



Source: CSA Global

12.1.3 Database validation/adjustments

A high-level validation was undertaken and included the following checks:

- examining the sample assay, collar information, and geology data to ensure that the data were complete for all drillholes
- examining the de-surveyed data in 3D to check for spatial errors and their position relative to mineralisation
- checking for “FROM-TO” errors to ensure that the sample data do not overlap
- checking collars relative to the topography; minor immaterial differences were observed

The MRE QP considers the drilling, sampling, assaying, and QA/QC procedures utilised by Giyani have resulted in data that is of sufficient quality to support a subsequent MRE. The MRE QP confirms verification the data to the extent described above and confirms its suitability and adequacy as such.

12.2 Mining data evaluation

The mining QP, Mr. Matthew Randall, BSc (Hons), PhD, MIMMM, CEng, is satisfied that the necessary steps were taken to verify the data used for the mining methods assessment of the K.Hill Project.

12.3 Metallurgical data evaluation

The metallurgical test work and process design criteria used in this Technical Report are based on analytical data reported during the laboratory test work program. The metallurgical test work program was conducted by Mintek, 200 Malibongwe Drive, Randburg, 2125, South Africa, a fully accredited laboratory under the South African National Accreditation System (SANAS). All analyses were undertaken using fully accredited analytical procedures and the results of tests were issued as analytical chemistry test reports under signature of a duly authorised representative of Mintek.

The metallurgical QP, EUR ING, Andrew Carter, BSc, CEng, MIMMM, MSAIM, SME, is satisfied that QA/QC procedures as required by SANAS have been followed by Mintek, ensuring the integrity of the data used in compilation of this report.

12.4 Tailings management facility data evaluation

The TMF QP, Mr. Richard Elmer, BSc, MSc, CEng, MIMMM, MCSM, conducted a data review of all information, relevant to the tailings, made available via the Project's data sharing platform (Microsoft® SharePoint). The following documents were the focus of the review and were used to develop the design basis for the TMF, presented in Table 18.3:

- *Kgwakgwe Hill Manganese Feasibility Study - Tailings Management Facility Design* prepared for Giyani by SRK, dated July 2022 (SRK KZ0647), including appendices (SRK 2022b)
- *Kgwakgwe Hill Manganese Feasibility Study - Geotechnical Interpretative Report* prepared for Giyani by SRK, dated March 2022 (SRK KZ0647), including appendices (SRK 2022a)
- *Giyani LOM 200Ktpa Schedule Summary*, Rev 1.3.0
- *K.Hill Battery-Grade Manganese Project Feasibility Study, National Instrument 43-101 Technical Report* by SRK and Tetra Tech, dated November 2022 (SRK and Tetra Tech 2022)

The TMF QP has reviewed the above-noted data, which includes soil test work results, tailings test work results, and stability assessments as appendices.

The ground conditions are summarised as topsoil varying in thickness between 0.3 m and 1.0 m comprising sand and gravel with up to 41% fines content. Topsoil is underlain by sand and gravel to depths of 0.6 mbgl to 7.5 mbgl, with permeabilities between 1×10-6 m/s and 1×10-7 m/s based on percolation tests. SRK recommended an internal friction angle Φ (phi) = 34°.

Bedrock was encountered below the sand and gravel comprising highly weathered Kgwakgwe CDB in the northern TMF area and soft ferruginous shale in the southern TMF area.

Groundwater was encountered to be at a depth between 4.6 mbgl and 11.3 mbgl and between 3 mbgl and 12 mbgl when measured 4 to 5 months later (10^{th}December 2021).

Southern Botswana is reported as an area of very low seismicity; therefore, seismic considerations are not deemed relevant to the Project.

Tailings properties were previously assessed based on undertaking a moisture content/dry density relationship (Proctor) test, classification tests, and remoulding samples at a target moisture content or dry density to undertake consolidation and shear strength testing. Table 12.1 presents the tailings classification test results.

Table 12.1 Tailings classification test results

Material description	Plastic limit	Liquid limit	Plastic index	SG	Maximum dry density	Optimum moisture content
Silt	27%	35%	8	2.92 t/m^3	1.597 t/m^3	27%

Tetra Tech has since advised that filtered tailings will contain 63% solids by mass based on laboratory scale tests. This relates to a geotechnical moisture content of 59% ($100 \times 37/63$). While industrial scale filter presses may increase the solids content, it cannot be relied upon to reduce the geotechnical moisture content below the liquid limit of 35% or to the optimum moisture content of 27% for placement and compaction as a "dry stack."

The TMF QP has recommended that further testing is required to understand the tailings condition following the filter process and the deposited tailings properties. For the purposes of the PEA, it is assumed that the tailings will be in the form of a filtered material as produced by the process filter press.

As no new test work has been completed since the completion of the above-noted documentation, the TMF QP considers the soil and tailings test work procedures described as being appropriate and the results are in line with the quality expected to support a PEA-level study.

12.5 Mine geotechnical data evaluation

The geotechnical QP, Mr. Rob Thomas, MSci ARSM, FAusIMM(CP), CEng, CSi, RenvP, MIMMM, is satisfied that the necessary steps were taken to verify the data used for the mine geotechnical assessment of the K.Hill Project.

12.6 Hydrological data evaluation

Sections of the PEA with regard to hydrology considerations are based on the information contained in the previously completed Technical Report (SRK and Tetra Tech 2022), which includes a discussion on the hydrography and hydrogeology of the K.Hill Project area.

A high-level validation of the information contained within the previously completed Technical Report (SRK and Tetra Tech 2022) was conducted by evaluating the following:

- The discussion on surface runoff compared to expected drainage patterns at the hand of topographical elevations, satellite imagery, and the Project site layout
- The aquifer description compared to the expected aquifer characteristics based on experience in a similar environment (geology, climate)
- The groundwater flow patterns compared to expected flow patterns, taking into account no large-scale groundwater abstraction within the mining area. It should be noted that the groundwater level data originates from the 1980s and 1990s and is considered to be out of date.
- The calculated open pit inflow volumes from surface runoff cannot be directly verified due to no discussion on the input parameters, assumptions and simplifications made during the calculations contained within the previously completed Technical Report (SRK and Tetra Tech 2022).

Regarding the hydrological assessment, the hydrological QP considers that the baseline characterisation of both the surface water and the groundwater baseline conditions are of sufficient quality to support an MRE, with the exception of the groundwater level information, which is at least 30 years out of date and should be updated with current data.

The hydrological QP cannot confirm the accuracy of the surface water runoff calculations due to the lack of detail on the calculation methodology. It should be noted that the inflow calculations were updated by CSA Global as part of this study using the latest pit layout. The calculated inflows are an extrapolation of the inflow calculations completed as part of the SRK and Tetra Tech (2022) Technical Report, and there could be an error in this depending on the accuracy of the original calculations; therefore, a detailed update of the surface water runoff study is recommended as part of this PEA (Section 26).

12.7 Conclusions

The QPs are satisfied that the necessary steps were taken to verify the data used for the various sections of this Technical Report.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Summary

Tetra Tech, working closely with Giyani and guided by historical test work reports, implemented new test work in order to characterise, understand, and optimise the metallurgical extraction of manganese for the production of HPMSM from the K.Hill Project.

Historical documentation, developed by Giyani and its consultants between 2018 and present, was available from SRK, MSA Group, Vietti Slurrytec, Lab 4 Inc., and Dalhousie University. The available documentation covered mineralogy, chemical assays, leach testing, and solvent extraction testing.

The test work initiated by Tetra Tech and implemented at Mintek in South Africa included assays, specific gravity, mineralogy, comminution, solid/liquid separation, leach optimisation, jarosite precipitation, iron and aluminium precipitation, other base metal precipitation, calcium and magnesium precipitation, fluoride removal, crystallisation, and manganese hydroxide precipitation.

In addition to the K.Hill North orebody comprising manganese and iron shales, samples were taken of the K.Hill B Horizon material, a separate local orebody, and were subjected to mineralogical tests, assaying, and leach extraction work. The intention was to understand the differences, if any, between the B Horizon material and the OLOM sample.

13.1.1 Comminution development

Comminution test work was consistent and indicated that the K.Hill Project material, from both the OLOM material and the B Horizon material, can be regarded as soft for crushing, low for abrasion, and variably soft to hard for the grinding.

13.1.2 Solid/liquid separation development

The solid/liquid testing consistently indicated that, post leach, the OLOM leach residue material settles poorly and produces low solids content thickener underflows. On this basis, the decision was made to use filtration for separation, rather than settling.

13.1.3 Leach development

The leach performance was investigated and demonstrated that a high temperature (90°C) reductive leach produced excellent extraction results of 95% to 99% extraction. After evaluation, sucrose, as the reductant in early test work, was replaced sulphur dioxide, which provided benefits of cost, practicality, and reduced acid consumption. The B Horizon material showed very similar results to the OLOM material. The manganese extraction was found to be insensitive to head grade and mineralogical composition.

13.1.4 Precipitation and purification development

Test work indicated that the stage-wise precipitation of impurities was effective for their removal. Manipulation of pH, the addition of chemical reagents, and the use of activated alumina allowed production of a solution that was suitable for crystallisation. The purification is sensitive to impurities in the feed, and this affects the amounts of reagents used, precipitates produced, and associated treatment costs.

13.1.5 Crystallisation development

Two routes were pursued for the crystallisation design development:

- use of synthetic solution produced from analytical grade chemical reagents
- work at Mintek to produce a solution from representative material that had been processed step-by-step through the intended recovery methods

The synthetic solution provided a means for rapid and inexpensive production of crystalliser feed, allowing the development of initial process feed purity requirements and expected operational criteria. The Mintek work yielded a fully representative stock solution that provided the opportunity to complete the crystalliser design. The HPMSM produced by the crystalliser met the product specification and provided a significant amount of process-accurate HPMSM for acceptance testing and trials by potential battery metal clients.

13.1.6 Overall recovery

From the test work and process development, 88.5% overall recovery of manganese as battery-grade HPMSM is anticipated.

13.2 Source documents

Tetra Tech had access to the original laboratory test work reports and the following documents were reviewed:

- *Kgwakgwe Hill Manganese Project Independent Technical Report* (SRK 2020)
- *Mineral Resource Estimate for the K.Hill Manganese Project, Botswana, NI 43-101 Technical report* (MSA Group 2018)
- *Metallurgical Sample Specification Memo* (Tetra Tech 2022)
- *Giyani K.Hill Process Verification* (Mintek 2022)
- *K.Hill Project Solid Liquid Separation Test Work* (Vietti Slurrytecc 2021)
- MSM Crystallisation Test Work (Confidential)

All test work reported in Section 13 is taken from the information presented in these documents.

13.3 Historical test work

Historical test work was completed under the guidance of Dr. Ian Flint of Lab 4 Inc., in Dartmouth, Nova Scotia, Canada, in two stages:

- Stage 1 included optical and electron probe work that identified both valuable and waste minerals, mineral particle size distributions, and the approximate grind sizes required for mineral phase liberation or exposure.
- Stage 2 included tests on the leaching of three samples to determine the dissolution characteristics and residence times required to extract the manganese in the primary leach stage of a hydrometallurgical process.

13.3.1 Mineralogical test work

Dalhousie University, in Nova Scotia, Canada, undertook a mineralogical analysis of the K.Hill Project material using optical and electron microscopy probes. The analysis was performed on a selection of manganese oxide bearing rocks taken as grab samples from the K.Hill Project mineralisation. Four samples were tested to determine the mineralogical composition of the manganese minerals. These samples and the resulting determinations are shown in Table 13.1.

Table 13.1 Identified and possible mineral species in K.Hill Project samples

Sample No.	Description	Mineral	Formula
KH17 MT01	Dump material	Cryptomelane	$K(Mn^{4+}{}_7Mn^{3+})O_{16}$
		Hausmannite	$Mn^{2+}Mn^{3+}{}_2O_4$
		Hollandite	$Ba(Mn^{4+}{}_6Mn^{3+}{}_2)O_{16}$
		Psilomelane	$Ba(Mn^{2+}Mn^{4+}{}_3)O_{16}(OH)_4$
		Pyrolusite	MnO_2
KH17 MT02	Altered shale	Jacobsite	$Mn^{2+}Fe^{3+}{}_2O_4$
KH17 MT03 KH17 MT05	Shale with kaolin	Coronadite	$PbMn^{4+}{}_6Mn^{3+}{}_2O_{16}$
		Cryptomelane	$K(Mn^{4+}{}_7Mn^{3+})O_{16}$
		Hausmannite	$Mn^{2+}Mn^{3+}{}_2O_4$
		Hollandite	$Ba(Mn^{4+}{}_6Mn^{3+}{}_2)O_{16}$
		Psilomelane	$Ba(Mn^{2+}Mn^{4+}{}_3)O_{16}(OH)_4$
KH17 MT04	Silicified shale	Hausmannite	$Mn^{2+}Mn^{3+}{}_2O_4$
		Psilomelane Group	$Ba(Mn^{2+}Mn^{4+}{}_3)O_{16}(OH)_4$

Dalhousie University reported that, in terms of presentation, the manganese minerals occurred in three forms:

- as staining on the silicates, iron oxides, and as themselves
- as small veins where manganese oxides have been deposited on each other, particularly within well-fissured zones
- as nodules where the manganese has built up into botryoidal masses that may contain other minerals within them
    - ✦ It was also reported that the mineralogical analysis did not specifically investigate the nodules owing to insufficient samples.

Dalhousie University also reported that the head assays of the samples were reported after converting them to the standard oxide form. The assays are shown in Table 13.2.

Table 13.2 Head assays

Sample	Description	FeO (%)	MnO (%)
KH17 MT01	Dump material	39.7	48.7
KH17 MT02	Altered shale	27.3	48.8
KH17 MT04	Silicified shale	4.5	4.3
KH17 MT03 KH17 MT05	Shale with kaolin	23.2	60.8

13.3.2 Metallurgical test work

Test work sample

Tetra Tech was not involved with the sample selection for the metallurgical test work completed by Dalhousie University. It is understood that three different intervals from a single drillhole were used for the metallurgical test work. Sample descriptions and related pictures are shown in Table 13.3.

Table 13.3 Metallurgical test work samples

Sample	Description	From (m)	To (m)	Photograph
KH18 MT010	Mn-shale	23.73	27.00	
KH18 MT011	Fe-shale	27.00	30.85	
KH18 MT012	Manganese oxide	48.00	50.73	

Head assays

Dalhousie University reported the manganese and iron head assays for the test samples (Table 13.4). The analytical methods were not specified in the available reports.

Table 13.4 Manganese and iron head assay results for test samples

Sample	Mn (%)	Fe (%)
KTH18 MT010	24.70	13.80
KTH18 MT011	30.70	12.10
KTH18 MT012	6.85	1.99

Comminution test work

Tetra Tech understands that comminution test work was not completed due to limited sample availability.

Leach test work

Dalhousie University conducted ROM acid leach tests with sucrose added as a reductant. In addition, baseline leach tests were also conducted under the same conditions but without any reductant addition. The tests were conducted with 125 mL of 3.64 Molar sulphuric acid (H_2SO_4) solution. A summary of test conditions is shown in Table 13.5.

Table 13.5 Leach test work conditions

Description	Value
Grind size - P_{80} (µm)	200
Sample mass (g)	25
Acid concentration (g/L)	260
Temperature (°C)	90
Retention time (hour)	3

A summary of the leach results is shown in Table 13.6.

Table 13.6 Summary of leach test results

Sample	Redundant	Head assay (%)		Leach extraction (%)									
				15 minute		30 minute		60 minute		120 minute		180 minute	
		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 shown in Table 13.6 indicate that the reductant significantly improved the manganese extraction as well as the leaching kinetics. However, the test work programme did not evaluate other possible reductants, such as sulphur dioxide, which is commonly used in acidic sulphate reduction processes. Significant savings can be obtained by adopting sulphur dioxide as a reductant, as it is used as a reductant in the process, and sulphurous acid (H_2SO_3) is produced during solubilisation as an intermediate reactant. This sulphurous acid is further oxidised in the process to sulphuric acid, which is used as the lixiviant in the leaching process. This can lead to savings due to reduced acid consumption.

The solution assays for various elements at the end of the leach test are shown in Table 13.7.

Table 13.7 Solution assays

Element	Assay (mg/L)	Element	Assay (mg/L)	Element	Assay (mg/L)	Element	Assay (mg/L)
Ag	1.0	Cr	1.5	Mn	26,470.0	Si	217.0
Al	938.0	Cu	29.0	Mo	0.1	Sn	1.0
As	0.1	Fe	6,589.0	Na	111.0	Sr	2.4
Ba	0.1	Ga	0.5	Nb	1.0	Ta	28.0
Be	0.3	Ge	5.0	Ni	38.0	Te	1.0
Bi	0.5	In	1.0	P	19.0	Ti	21.0
Ca	132.0	K	641.0	Pb	2.2	Tl	10.0
Cd	0.1	La	2.0	S	78,634.0	V	21.2
Ce	20.0	Li	3.1	Sb	1.0	Zn	18.0
Co	138.0	Mg	206.0	Se	7.5	Zr	1.1

Notes:

Ag - silver
As - arsenic
Ba - barium
Be - beryllium
Bi - bismuth
Cd - cadmium
Ce - cerium
Co - cobalt
Cr - chromium
Ga - gallium
Ge - germanium
In - indium
K - potassium
La - lanthanum
Li - lithium
Mo - molybdenum
Na - sodium
Nb - niobium
P - phosphorus
Pb - lead
S - sulphur
Sb - antimony
Se - selenium
Si - silicon
Sn - tin
Sr - strontium
Ta - tantalum
Te - tellurium
Ti - titanium
Tl - thallium
V - vanadium
Zn - zinc
Zr - zirconium

The assays indicate that several metals (iron, aluminium, copper, zinc, nickel, and cobalt) are co-extracted as sulphates during leaching. The solution can be purified and the iron and aluminium sulphate removed in the form of hydroxides by precipitation with lime, as indicated in the chemical reactions shown in Equations 13.1 and 13.2.

$$Fe_2(SO_4)_3 + 3Ca(OH)_2 + 6H_2O = 2Fe(OH)_3 + 3CaSO_4.2H_2O \quad \text{Equation 13.1}$$

$$Al_2(SO_4)_3 + 3Ca(OH)_2 + 6H_2O = 2Al(OH)_3 + 3CaSO_4.2H_2O \quad \text{Equation 13.2}$$

The solution can be further purified by removal of the base metals in the form of sulphides via precipitation with sodium hydrosulphide, as shown in Equations 13.3 to 13.6.

$$CuSO_4 + NaHS = CuS + NaHSO_4 \qquad \text{Equation 13.3}$$

$$ZnSO_4 + NaHS = ZnS + NaHSO_4 \qquad \text{Equation 13.4}$$

$$NiSO_4 + NaHS = NiS + NaHSO_4 \qquad \text{Equation 13.5}$$

$$CoSO_4 + NaHS = CoS + NaHSO_4 \qquad \text{Equation 13.6}$$

Solvent extraction test work

Dalhousie University conducted solvent extraction tests on the pregnant leach solution produced in the leach tests. The solutions were tested with the organic extractant di(2-ethylhexyl)phosphoric acid (D2EPHA). The pH conditions and results of each test are summarised in Table 13.8.

Table 13.8 Solvent extraction test results

Sample	Mn (ppm)	Mn extraction (%)	pH
KTH18 MT010	16,908	95.8	3.0
KTH18 MT011	12,222	96.7	3.7
KTH18 MT012	7,047	92.2	3.6

The results indicate that approximately 96% of manganese could be extracted by D2EPHA into the organic phase. The loaded organic solvent could then be stripped with concentrated sulphuric acid to produce a concentrated manganese sulphate solution. The sulphate could then be recovered by vacuum crystallisation to produce an HPMSM product.

The solvent extraction tests reported in Table 13.8 were not optimised but conducted as a standard scoping test. Thus, the 96% extraction is regarded as indicative only.

13.4 Current test work

13.4.1 Mineralogical test work

Sample

Selection criteria

Tetra Tech specified the sample requirements and selection criteria appropriate to a Feasibility Study-level metallurgical test work programme. Tetra Tech's sample selection was based on the following criteria:

- there are two distinct lithological domains: Mn-shale (MSH) and Fe-shale (FSH)
- sample sources:
  - comminution: whole PQ3 drill core
  - hydrometallurgy: RC rejects from the geological drilling programme

Sample selection

A total mass of approximately 890 kg was selected for the test work.

Samples were packaged in waterproof sample bags with appropriate labelling, sealed, and then batched. The batches were packed into 30 L high-density polyethylene (HDPE) lock ring closure drums that were labelled, sealed, and shipped to the laboratory.

A summary of the different samples, expected sample mass, appropriate labels, and drums required for transport is shown in Table 13.9. Drums were labelled with the sample label and for variability samples either a COM VAR seq no. or an HM VAR seq no.

Table 13.9 Summary of sample selection

Test	Type	No of Samples	Mass (kg)	Sample Label	No of Drums
Comminution	Variability	11	169	-	4
	Composite - MSH	1	61	COM-COMP-MSH	2
	Composite - FSH	1	61	COM-COMP-FSH	2
	Composite - LOM	1	61	COM-COMP-LOM	2
Hydrometallurgy	Variability	6	60	-	2
	Composite - MSH	1	168	HM-COMP-MSH	4
	Composite - FSH	1	126	HM-COMP-FSH	4
	Composite - LOM	1	184	HM-COMP-LOM	5
Expected total		23	890	-	25

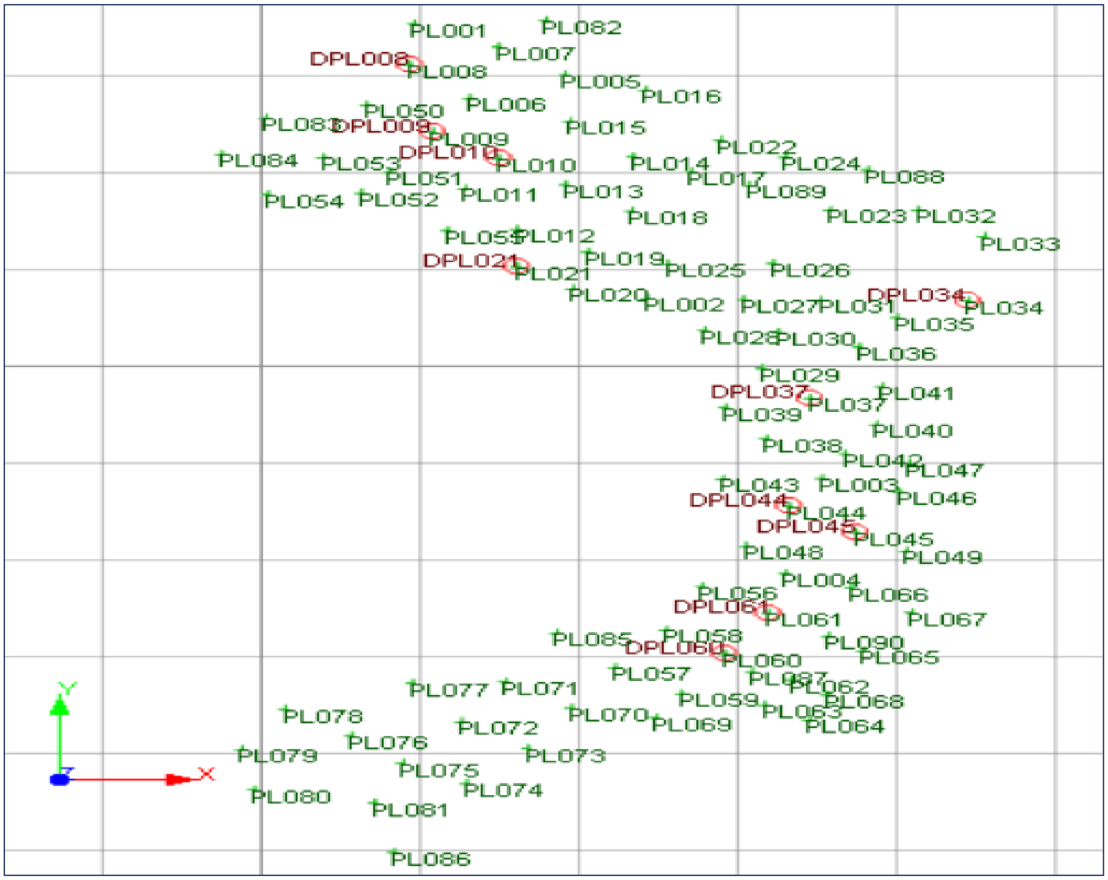
Comminution Sample Selection Method

The comminution samples were selected from the planned PQ3 drill core, assuming the whole of the core of the selected intervals would be available for the test work.

Variability samples were selected to cover the spatial distribution of drillhole samples, as shown on the drillhole map on Figure 13.1. Nine samples (five from FSH and four from MSH) were selected, and the tenth sample was taken from within the orebody where the lithology type was not known. In addition to the ten variability samples of material, an additional variability sample was selected from the host rock, since this material is likely to enter the mill as dilution material. The approximate length of the core and mass for each of the comminution variability samples was 1.5 m and 15 kg, respectively.

Upon completion of the variability sample selection, suitable intervals were identified within the leftover core to select three composite samples, each with an approximate mass of 60 kg.

Figure 13.1 K.Hill North Orebody drillhole location map indicating location of variability samples (in red)



Hydrometallurgical Sample Selection Method

The hydrometallurgical variability samples were taken from RC drilling completed in December 2020. The primary aim of the hydrometallurgical composite sample selection was to match the sample grades of the samples that made up the OLOM composite sample, which had a composite manganese oxide grade of 34%.

The aim of the variability sample selection was to compile a low-, medium-, and high-grade sample for each of the two mineralogical domains (MSH and FSH).

The lithology composite samples were selected from the completed RC drill holes. Intervals that represented the two lithologies were selected and then combined to make a composite sample.

The residual sample from the FSH domain was insufficient to achieve an equal proportion of both domains in the LOM composite that is comparable to the LOM manganese oxide grade. Thus 32% (by mass) of FSH samples and 68% of MSH samples were combined to make the LOM composite.

Material characterisation

Material characterisation tests were performed including comprehensive chemical and mineralogy analyses on the three composite samples.

Sample preparation

Mintek received RC drill samples classified into three material types. The material types were delivered in separate 220 L plastic drums, with specific labelling for the three composites (i.e., COMP LOM, COMP MSH, and COMP FSH). The samples in the drums were subdivided into multiple bags. Bags containing similar ore types were blended to obtain a representative sample of each type.

Each blend of the three material types was crushed to 1.7 mm. Initially, only the COMP LOM sample was milled to three different grind sizes (80% 212 µm, 80% 150 µm and 80% 106 µm) until the optimum grind size was determined from the subsequent test work. Two, 200 g subsamples of each crushed sample (1.7 mm) were taken and submitted for chemical and mineralogy analyses.

Head grade analysis

A semi-quantitative XRF scan was conducted on each composite sample. Head grade analysis was conducted on each sample (pulverised) using inductively coupled plasma-optical emission spectroscopy (ICP-OES) for cobalt, copper, iron, magnesium, aluminium, silicon, calcium, titanium, vanadium, chromium, manganese, nickel, and zinc in which the detection limit was 0.05% (metre by metre [m/m]). The results of the XRF scan on the three composites are listed in Table 13.10; the results of the ICP-OES analysis are listed in Table 13.11.

Table 13.10 Semi-quantitative x-ray fluorescence scan on LOM, MSH, and FSH composite samples

Elements	Units	LOM	MSH	FSH
Mn	%	32.23	30.85	28.87
Al	%	5.48	5.44	6.40
Ba	%	1.22	1.32	1.31
Ce	%	nd	0.13	nd
Fe	%	19.88	22.33	19.86
K	%	1.29	1.13	1.65
Mg	%	0.07	0.11	0.19
P	%	0.07	0.11	0.07
Pb	%	1.26	2.15	1.36
Si	%	6.51	5.19	7.34
Sr	%	0.16	0.15	0.17
Ti	%	0.31	0.34	0.28
V	%	0.09	0.08	0.08
Ca	ppm	342.71	269.75	368.61
Cl	ppm	197.20	nd	425.41
Co	ppm	408.89	273.75	274.98
Cu	ppm	226.07	241.11	274.52
Na	ppm	260.60	93.52	281.12
Ni	ppm	187.32	170.58	275.42
Rb	ppm	85.75	100.21	107.00
S	ppm	36.66	58.92	16.00
Zn	ppm	530.71	546.47	474.19
Zr	ppm	nd	17.71	nd

Notes:
nd - not detected

Table 13.11 Inductively coupled plasma-optical emission spectroscopy head assays on LOM, MSH, and FSH composite samples

Elements	Units	LOM	MSH	FSH
Mn	%	26.6	26.4	23.9
MnO	%	34.3	34.1	30.8
Al	%	4.75	4.94	5.46
Ca	%	0.10	0.14	0.12
Co	%	<0.05	<0.05	<0.05
Cr	%	<0.05	<0.05	<0.05
Cu	%	<0.05	<0.05	<0.05
Fe	%	13.70	15.5	13.5
Mg	%	0.16	0.17	0.25
Ni	%	<0.05	<0.05	<0.05
Pb	%	0.80	1.43	0.87
Si	%	9.83	7.68	10.6
Ti	%	0.18	0.21	0.19
V	%	<0.05	<0.05	<0.05
Zn	%	0.057	0.066	0.061

The assays showed high manganese grades in all the composite samples, ranging between 23.9% and 26.6%, based on ICP-OES. There was a notable difference between the ICP-OES and XRF analyses in relation to accessory elements; this is because the XRF method has a lower accuracy compared to the ICP-OES. However, the manganese assays from both techniques are in good agreement.

The major impurity elements were iron (13.5% to 15.5%), silica (7.68% to 10.60%), and aluminium (4.75% to 5.46%), based on ICP-OES. The XRF scan also detected a notable presence of lead (1.26% to 2.15%), potassium (1.13% to 1.65%), and barium (1.22% to 1.32%). The rest of the impurity elements were less than 1% or measured very low.

Specific gravity

Before 1st July 2020, a mean in-situ material density of 2.7 t/m³ was used for resource and process estimates. Subsequently a detailed sampling programme was implemented to confirm the in-situ material density. Following receipt and review of the comprehensive K.Hill pycnometric density data from multiple core samples at various collar locations around the mine site, the mean pycnometric density increased from 2.7 to 3.48 t/m³.

Mineralogy

The analyses were conducted on the 1.7 mm crushed samples. The head samples were screened into discrete size fractions based on particle size distribution. Representative samples from each of the size fractions were mounted into polished sections for analysis. Sizing of the sample produced a better result from the AutoSEM (Quantitative Evaluation of Materials by Scanning Electron Microscopy [QEMSCAN]) system. After analysis, the results from the individual size fractions were then recombined into a single result for the total sample and weighted according to the mass distributions of the size fractions. The product samples were pulverised and micronised for x-ray diffraction (XRD).

The bulk modal mineralogy of head samples FSH, MSH, and LOM are presented in Table 13.12. The samples predominantly comprise manganese bearing phases (approximately 70% m/m); cryptomelane and bixbyite are the main manganese bearing phases, while hollandite and manganese mica are the next most significant. The manganese oxide group includes pyrolusite and hausmannite, while "Other" includes sulphides such as pyrite as well as pyroxenes and carbonates. The manganese bearing phases have been validated using XRD. The MSH and LOM samples had a slightly higher cryptomelane content, while FSH had a high manganese mica content. The gangue phases in the sample include quartz, hematite, and kaolinite.

Table 13.12 Bulk modal mineralogy for the three samples: FSH, MSH and LOM

Mineral	Ideal Formula	Mass %		
		FSH	MSH	LOM
Cryptomelane	$K(Mn^{4+}, Mn^{2+})_8O_{16}$	19.5	23.3	24.1
Bixbyite	$(Mn^{3+},Fe^{3+})_2O_3$	17.3	16.4	17.3
Hollandite	$Ba(Mn^{4+}{}_6Mn^{3+}{}_2)O_{16}$	8.8	11.0	8.1
Manganese oxides	-	7.2	5.9	9.6
Birnessite	$(Na,Ca,K)_x(Mn)_2O_4.1.5(H_2O)$	2.2	1.4	1.6
Alabandite	MnS	0.2	0.6	0.3
Manganese mica	$KAl_3Si_3O_{10}(OH)_{1.8}F_{0.2}(Mn)$	14.0	10.8	9.0
Manganese silicate	$Na(Fe^{2+},Mn^{2+})_{10}(Fe,Al)_2Si_{12}O_{31}(OH)_{13}$	0.9	0.9	0.8
Hematite	Fe_2O_3	7.7	13.4	9.2
Mica	$KAl_3Si_3O_{10}(OH)_{1.8}F_{0.2}$	3.8	1.4	1.5
Quartz	SiO_2	10.8	5.7	11.4
Kaolinite	$Al_2Si_2O_5(OH)_4$	4.3	6.5	4.6
Actinolite	$Ca_2(Mg,Fe)_5Si_8O_{22}(OH)_2$	1.0	1.8	1.0
Chlorite	$(Mg,Fe^{2+})Si_3Al_2O_{10}(OH)_8$	0.9	0.3	0.1
Feldspar	$KAlSi_3O_8$	1.2	0.2	0.5
Ilmenite	$Fe^{2+}TiO_3$	0.1	0.0	0.1
Other (1)	-	0.2	0.4	0.8

Notes:
(1) Other includes sulphides such as pyrite as well as pyroxenes and carbonates.

Assay reconciliations for magnetic separation products are presented in Table 13.13, which shows the chemical and QEMSCAN generated mineralogical assay results. The QEMSCAN generated results are calculated using the mass proportions of the mineral phases present in conjunction with theoretical chemical composition, or energy dispersive spectroscopy (EDS) data, where available. The chemical and QEMSCAN assay results compare well; thus, the QEMSCAN modal results were confirmed as accurate.

Table 13.13 Chemical and QEMSCAN assays reconciliation

Element	FSH		MSH		LOM	
	Chemical Assay	QEMSCAN Assay	Chemical Assay	QEMSCAN Assay	Chemical Assay	QEMSCAN Assay
Al	5.5	5.3	5.0	4.7	4.8	4.4
Ba	1.3	1.1	1.3	1.2	1.2	0.9
Ca	0.1	0.1	0.1	0.2	0.1	0.1
Fe	13.6	13.9	15.5	16.1	13.7	14.4
Mg	0.3	0.3	0.2	0.3	0.2	0.2
Mn	23.9	23.8	26.2	26.5	26.4	27.3
Pb	0.9	0.7	1.4	1.3	0.8	1.0
Si	10.5	11.0	7.7	8.3	10.0	9.7

Element deportment

An element deportment provides the relative contribution of various mineral phases to the total element content in a sample; this was determined using the bulk modal mineralogy of the sample as well as the theoretical or measured chemistry of individual mineral phases.

The manganese and iron deportment of the head samples are presented in Table 13.14 and Table 13.15, respectively. The relative manganese contributions from the samples was similar, with approximately 40% of the manganese hosted in cryptomelane, and 30% in the bixbyite; 10% to 15% of the manganese is hosted in hollandite and manganese oxides. The iron is hosted in hematite in addition to the manganese bearing phases.

Table 13.14 Manganese deportment

Mineral	% Contribution		
	FSH	MSH	LOM
Cryptomelane	36.7	41.2	39.0
Bixbyite	30.4	27.0	27.8
Hollandite	13.1	15.1	10.6
Manganese oxide	14.3	10.7	17.8
Birnessite	3.7	2.1	2.4
Alabandite	0.5	1.2	0.5
Manganese mica	0.3	0.2	0.1
Manganese silicate	0.3	0.2	0.2
Hematite	0.8	2.3	1.5
Other	0.0	0.0	0.0

Table 13.15 Iron-bearing minerals

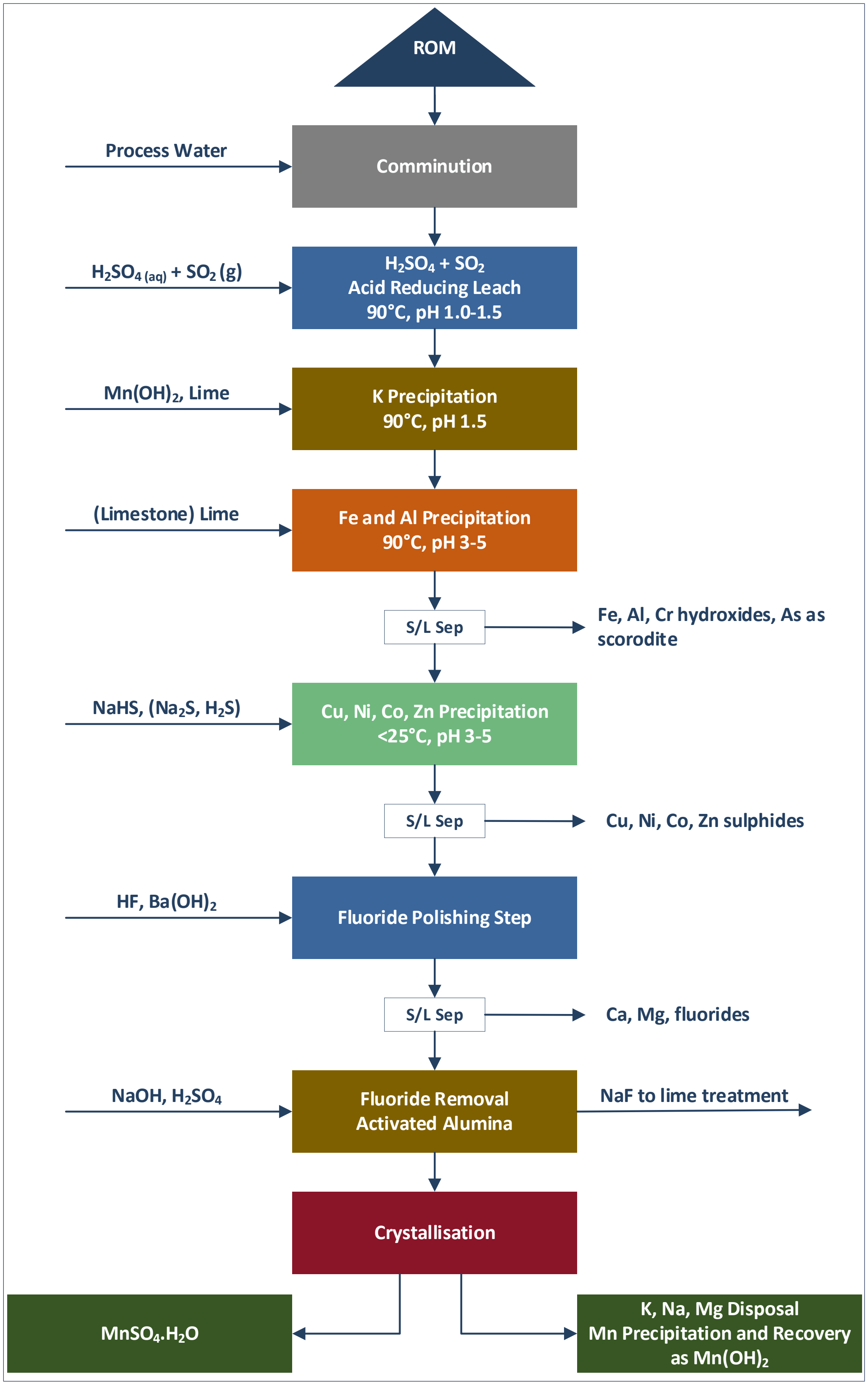
Mineral	% Contribution		
	FSH	MSH	LOM
Cryptomelane	24.6	22.6	26.1
Bixbyite	27.6	22.5	26.6
Hollandite	7.9	5.5	4.5
Manganese oxide	8.3	5.5	8.8
Alabandite	0.1	0.4	0.2
Manganese silicate	1.1	1.0	1.0
Hematite	28.1	40.6	31.1
Mica	0.1	0.0	0.0
Kaolinite	0.0	0.0	0.0
Actinolite	0.9	1.4	0.7
Chlorite	0.8	0.2	0.1
Ilmenite	0.3	0.1	0.2
Other	0.1	0.2	0.6

13.4.2 Metallurgical test work

Flow sheet development

The mineralised material comprises manganese and iron shales of variable hardness from soft to hard, that are amenable to reductive acid leaching in sulphate media using sulphur dioxide as a reductant. The process comprises crushing and grinding to a characteristic grind (P_{80}) of 150 µm, an acid reductive leach in sulphate media at elevated temperature using sulphur dioxide as a reductant, and a sequential purification process for the removal of base metal impurities.

Figure 13.2 Schematic of the metallurgical flowsheet



Source: Tetra Tech

Comminution test work

As part of Giyani's earlier studies for manganese recovery from the K.Hill deposit, comminution test work was conducted on various samples to obtain information on the ore deposit. Eleven variability samples and three composite samples were used in the investigation (Table 13.16). The following comminution tests were conducted:

- Bond crushability work index test (CWi)
- Bond abrasion index test (Ai)
- Bond ball work index test (BWi)

Table 13.16 Comminution test work results classification

CWi classification		Ai classification		BWi classification	
CWi (kWh/t)	Classification	Ai (g)	Classification	BWi (kWh/t)	Classification
<10	Very Soft	<0.2	Low	7-9	Soft
10-14	Soft	0.2-0.5	Medium	10-14	Medium
14-18	Medium	0.5-0.75	Abrasive	15-20	Hard
18-22	Hard	0.75-1	Very abrasive	>20	Very Hard
>22	Very Hard	>1	Extremely	-	-

The summary of the comminution results is presented in Table 13.17.

Table 13.17 Comminution results showing the 75th percentile

Sample	Sample ID	Sample type	SG (t/m3)	CWi (1) (kWh/t)	Ai (g)	BWi (kWh/t)
1	COM -VAR-FSH	DDKH21-0024	2.52	6.30	0.01	3.10
2	COM -VAR-FSH	DDKH21-0024	2.64	10.30	0.09	7.70
3	COM -VAR-MSH	DDKH21-0025	3.07	4.30	0.03	8.70
4	COM -VAR-UNCLASSIFIED	DDKH21-0024	2.46	10.40	0.29	10.90
5	COM -VAR-MSH	DDKH21-0023	2.66	8.60	0.03	8.30
6	COM -VAR-MSH	DDKH21-0022	2.66	7.10	0.06	15.90
7	COM -VAR-FSH	DDKH21-0020	2.55	5.10	0.01	2.60
8	COM -VAR-MSH	DDKH21-0021	2.76	1.40	0.02	5.20
9	COM -VAR-FSH	DDKH21-0021	Not Competent		0.26	10.00
10	COM -VAR-UNCLASSIFIED	DDKH21-0024	2.58	8.40	0.02	11.20
11	COM -VAR-FSH	DDKH21-0022	2.47	7.20	0.03	3.80
12	COM-COMP -MSH	-	2.48	6.80	0.01	8.20
13	COM-COMP -FSH	-	2.53	4.30	0.02	5.60
14	COM-COMP -LOM	-	2.62	6.50	0.09	8.00

Notes:
(1) 75th percentile results are reported.

In summary, the results represented in Table 13.17 show:

- The CWi test results classified the samples as very soft to soft.
- The Ai test results indicate that the samples have low to medium abrasiveness.
- The BWi test results indicate a very soft to hard deposit under ball milling.

Solid/liquid separation test work

Solid/liquid separation test work was carried out to generate process design data through test work for several solid / liquid separation steps in the hydrometallurgical process design. The test work was carried out by Vietti Slurrytec in South Africa.

Cyclone overflow (pre-leach)

The cyclone overflow material exhibited a fine particle size distribution of 90% passing 150 µm and 72% passing 22 µm. Suspension of the dry milled solids in tap water (owing to site raw water being unavailable) generated a naturally coagulated (settling) slurry, which is receptive to flocculation without any requirement for further slurry conditioning prior to flocculation.

Magnafloc 336 (M366) supplied by BASF was selected as the optimum flocculant type in terms of overall settling rate and supernatant clarity. M336 is a medium anionic charge, high molecular weight flocculant. The results are presented in Table 13.18.

Table 13.18 Cyclone overflow thickening conditions and results

Thickening Conditions	Units	Test 1	Test 2	Test 3
Flocculant dosing concentration	%w/w	0.025	0.025	0.025
Feed-well slurry solids concentration	%w/w	5	7.5	10
Flocculant dose rate	g/t	30	40	40
Static settling rate	m/h	59	26	19
Supernatant clarity (out of 50)	wedge no.	48	47	44

Combined leach residue and iron/aluminium precipitate

Bench top high-density thickening tests yielded a maximum underflow solids concentration of 25% w/w after a 5-hour residence time. The conditions and results of the pressure filtration are displayed in Table 13.19.

Plate and frame pressure filtration tests produced a competent hard cake, even though the moisture content was in the region of 36% m/m to 38% m/m, with a form time of 600 seconds at 10 bar pressure yielding a cake thickness of 22 mm to 24 mm.

Table 13.19 Combined leach residue and iron/aluminium precipitate test results

Parameter	Units	Value
Feed solids concentrate	% w/w	25
Filter	type	plate and frame pressure filter
Plate size	m × m	2 m × 2 m
Active plate filter area	m × m	1.8 m × 1.8 m
Chamber area	m^2	6.5
Number of chambers per unit	-	100

Parameter	Units	Value
Filter area per cycle	m^2	650
Number of cycles per hour	-	3
Pressure	bar	10
Cake thickness	mm	25
Cake moisture content	% w/w	36
Filter duty per unit	t/m^2.h	0.014

Base metal sulphide precipitate

Coagulation did not make a significant difference to solution clarity, and its impact could not be verified on settling rate due to the low solids content. Coagulation was therefore excluded for the rest of the tests. A preliminary flocculant dose rate of 0.3 ppm (mg/L) was determined during the static settling tests and yielded a supernatant clarity of 70 Nephelometric Turbidity Units (NTU). Dynamic thickening (clarification) tests showed that the flocculant dose rate had to be increased to 0.4 ppm to 0.45 ppm under dynamic conditions to maintain optimum flocculation. A hydraulic rise rate of 1 m/hour produced the best overflow clarity at 65 NTU during the dynamic tests. Only a very thin slice of mud bed formed because of the low solids content and lack of available sample volume. Therefore, no underflow density, mud bed consolidation behaviour, or rheological characteristics could be determined.

As a result of the very low solids content in the feed and the size of the bench top filter equipment, only a very thin film of "filter cake" developed, which could not be successfully isolated for moisture content determination. The design estimate values used were estimated from similar material results. Solid liquid separation test work will be carried out in the process demonstration plant. This is part of the planned reagent optimisation programme.

Extraction

Leach optimisation tests were carried out to establish suitable conditions for maximum extraction of manganese. The optimisation tests were carried out on the OLOM material, which was a reasonable indication of future plant feed comprising the Mn- and Fe-shales.

The OLOM composite sample was used to determine the optimum leaching conditions. The feed solids were pulped in deionised water, targeting a 30% m/m pulp density. The leach tests were conducted for 6 hours, and kinetic samples were only taken during the residence time optimisation test (Test 7) at hourly intervals. The redox potential was controlled at approximately 600 mV (vs. silver/silver chloride) by the addition of sulphur dioxide; the pH was controlled by the addition of 8 Molar sulphuric acid. The operating conditions under which the leach optimisation tests were conducted are summarised in Table 13.20.

Table 13.20 Leach optimisation test matrix

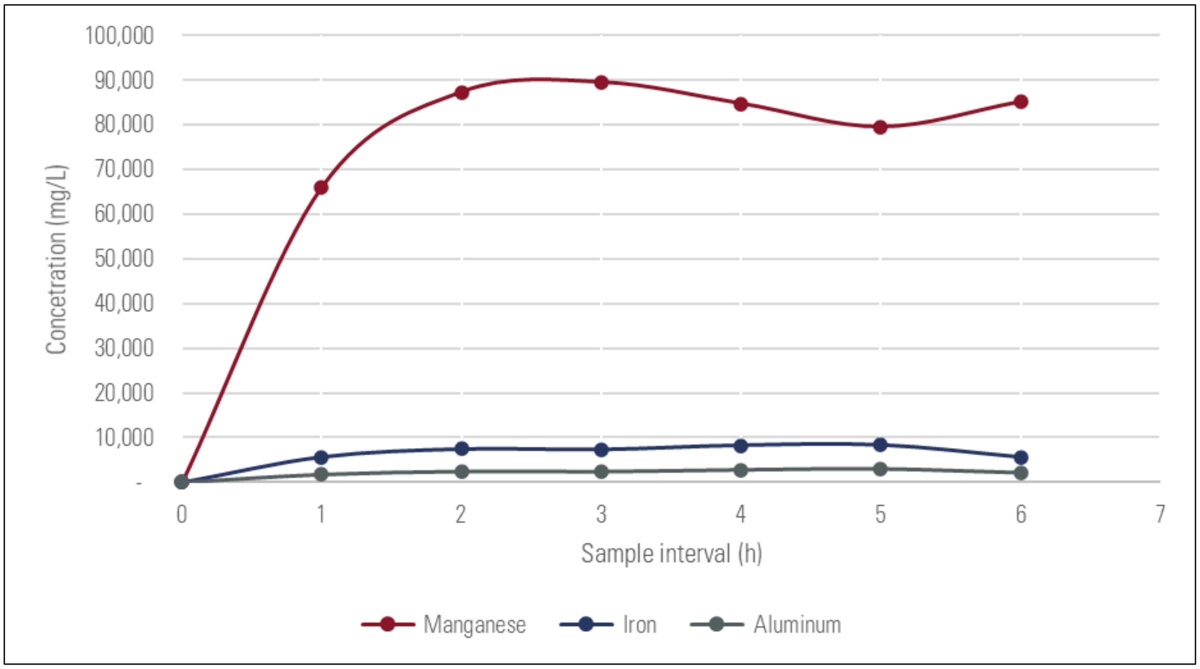
Parameter	Test 1	Test 2	Test 3	Test 4	Test 5	Test 6	Test 7
Grind size (P_{80})	-106 µm	-150 µm	-200 µm	-150 µm	-150 µm	-150 µm	-150 µm
Temperature (°C)	60	60	60	90	60	90	90
pH	1.5	1.5	1.5	1.5	1	1	1
Manganese extraction	81%	84%	75%	61%	80%	90%	94%

The results in Table 13.20 show that there was a small difference in manganese extractions between the three grind sizes and two pH conditions. Manganese leaching of the OLOM material ranged between 75% and 84%. The P_{80} 106 µm grind size was expected to yield the highest manganese leaching efficiencies owing to increased liberation; however, the P_{80} 150 µm performed better. The small difference in leaching efficiencies indicates that the majority of the manganese minerals were liberated in the P_{80} 212 µm grind size, and that not much improvement was made by milling to P_{80} 106 µm.

Extraction improved when the operating temperature was increased to 90°C and the pH set to a value of 1. Under these conditions, manganese leaching efficiency of the OLOM material increased to 90%.

Figure 13.3 shows that manganese leaching was relatively fast, reaching the maximum level within two hours. Extending the leach time beyond two hours did not yield any further improvement in the leaching efficiency.

Figure 13.3 Leach extraction profiles for manganese, iron, and aluminium at optimum conditions



Source: Tetra Tech

Variability tests

The parameters established in the optimisation test work were used for the variability tests, these were:

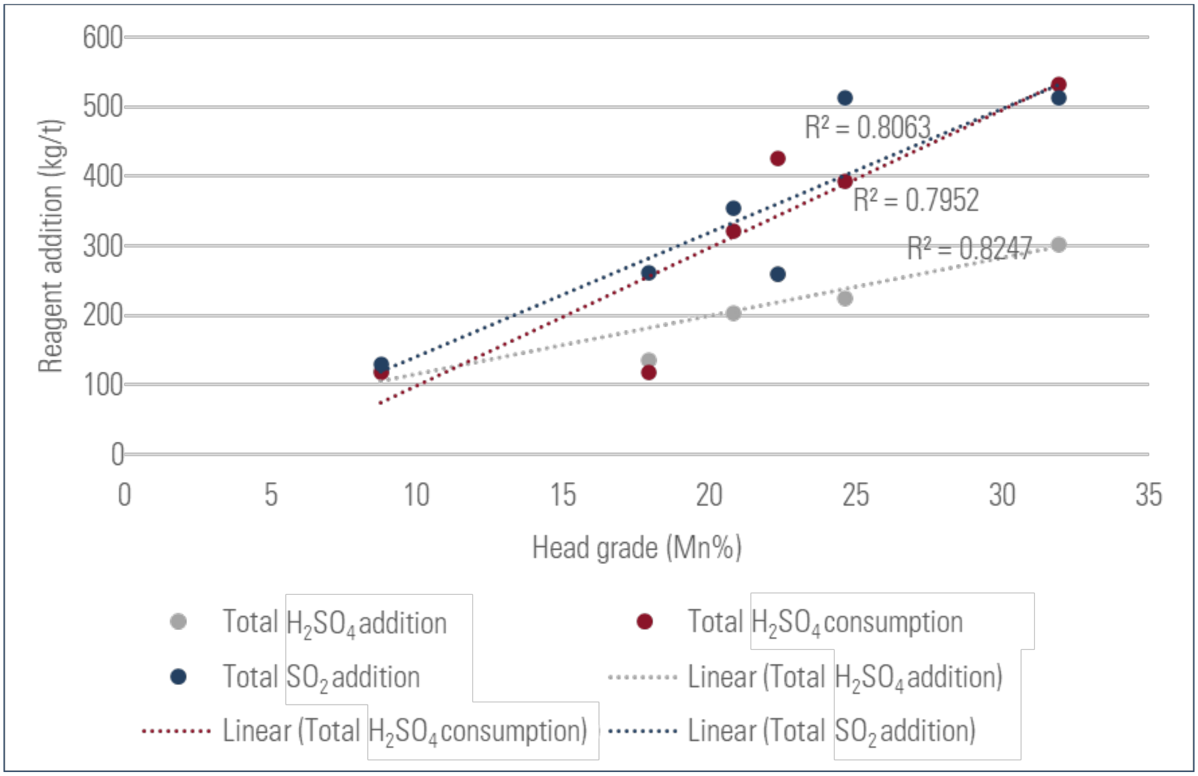
- grind size: P_{80} 150 µm
- slurry density: 30% (m/m)
- operating temperature: 90°C
- pH: 1.0
- oxidation reduction potential: 600 mV (vs. silver/silver chloride)

The redox potential was controlled at approximately 600 mV (vs. silver/silver chloride) by the addition of sulphur dioxide gas; the pH was controlled by the addition of 8 Molar sulphuric acid. The results of the six variability tests are shown in Table 13.21 and Figure 13.4.

Table 13.21 Leach variability test results

Parameter	Test V1	Test V2	Test V3	Test V4	Test V5	Test V6
Mn Head (%)	20.8	24.6	8.76	17.9	31.9	22.3
Mn Extraction (%)	97	97	96	95	99	97
Reagent Consumption						
Total H_2SO_4 addition (kg/t)	204	224	121	135	303	260
Total H_2SO_4 consumption (kg/t)	322	393	118	119	532	427
Total SO_2 addition (kg/t)	355	513	130	261	513	260

Figure 13.4 Variability test reagent addition



Source: Tetra Tech

The mean manganese extraction over the six tests was 96.7%, which was used for the process estimates. The reagent addition showed a strong correlation with the manganese head grade between the tests. The sulphur dioxide addition was significantly higher than anticipated and was attributed to varying mineralogy and test work equipment. However, mineralogy data showed the mineralogy was consistent; thus, the higher than anticipated sulphur dioxide addition was attributed to equipment and associated measurement error. To define the design sulphur dioxide addition, the mineralogy was used to estimate the Mn^{2+} to Mn^{3+} molar ratio. With the molar ratio of 1.38, the sulphur dioxide addition rate was set at 120% of the stochiometric molar ratio.

With the increased MRE and a decrease in the head grade, a linear regression model was used to predict the sulphuric acid addition.

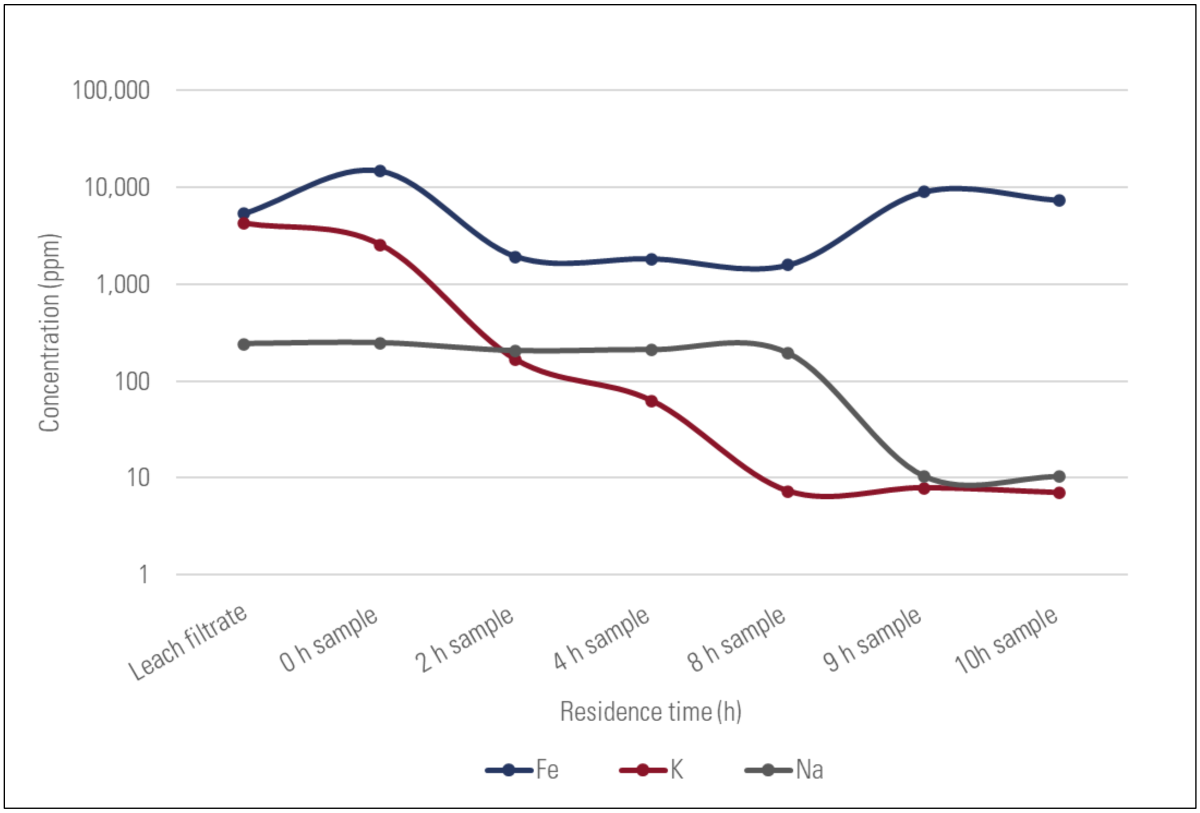
Purification

Jarosite precipitation

The objective of the jarosite precipitation was to remove the potassium and sodium impurities as a precipitate. Ferric sulphate ($Fe_2(SO_4)_3$) solution was used as a source of iron for the precipitation reactions.

Figure 13.5 shows that the potassium and sodium concentration could be removed to within the required concentrations with the addition of ferric sulphate.

Figure 13.5 Potassium and sodium removal by jarosite precipitation



Source: Tetra Tech

Iron/aluminium precipitation

Scoping iron/aluminium precipitation test work was conducted to evaluate the efficacy of selective pH-based impurity rejection as hydroxides to produce a relatively pure manganese sulphate solution. The test work comprised an initial neutralisation plotting the precipitation extent of impurities and manganese vs. pH to identify the optimum pH value at which most impurities are precipitated whilst retaining maximum manganese in solution. Thereafter, a bulk test was conducted using the optimum pH

Base metal precipitation

This purification step was undertaken to remove copper, cobalt, nickel, and zinc to trace levels. The initial tests were aimed at optimising pH, sodium hydrosulphide (NaHS) dosage, and residence time.

It is evident that metal sulphide precipitation is most efficient at a high pH (≥6.0); however, the optimum pH should be considered with other factors such as sodium hydrosulphide dosage and residence time. Too high a pH leads to the precipitation of metal hydroxides and the loss of manganese.

Fluoride polishing

Fluoride polishing describes the use of hydrofluoric acid (HF) to precipitate out the calcium and magnesium and the removal of fluoride by activated alumina. The results of this test work are described in the following subsections.

Calcium/magnesium precipitation

The solution was tested with three different reagents for pH adjustment to optimise the precipitation rate. The reagents were selected to limit addition of impurities to the solution. The reagents tests are listed in Table 13.22.

Table 13.22 Calcium and magnesium precipitation reagents tested

Scheme No	Reagents used
Scheme 1	HF and NH_3
Scheme 2	HF and NH_4OH
Scheme 3	HF and $Ba(OH)_2$

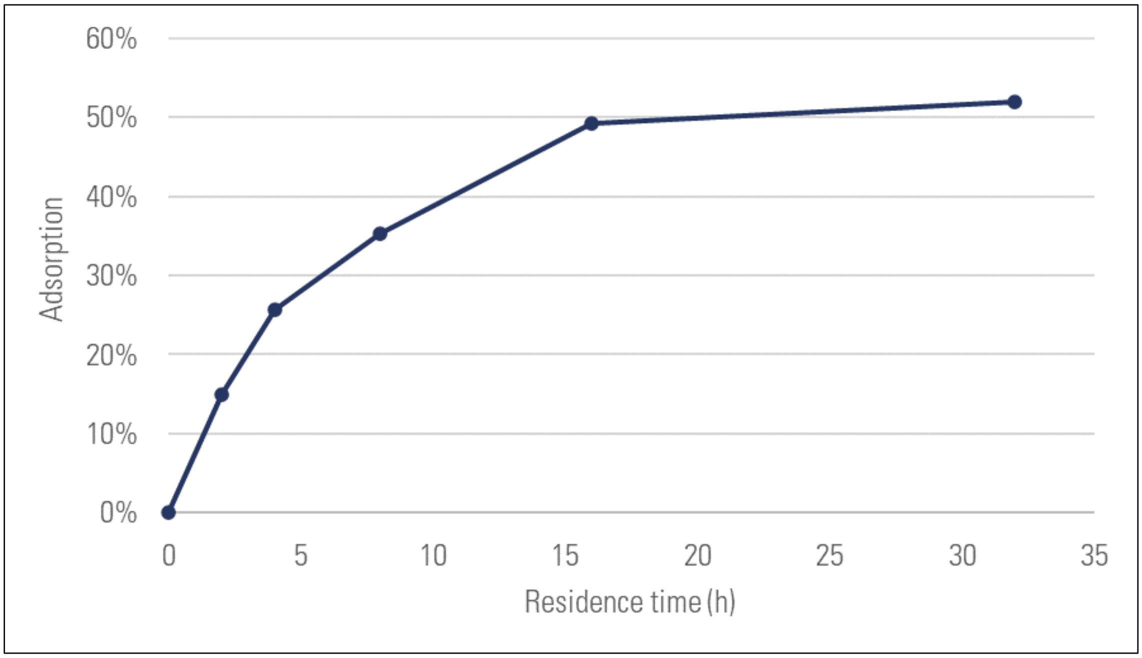
Notes:
NH_3 - ammonia
NH_4OH - ammonium hydroxide
$Ba(OH)_2$ - barium hydroxide

The reagents specified in Scheme 3 were found the be the most effective for the precipitation of calcium and magnesium.

Fluoride removal

The addition of hydrofluoric acid for the removal of calcium and magnesium introduces fluoride as an additional impurity, this was removed using activated alumina. The results from the equilibrium test showed that equilibrium was achieved after 32 hours. A loading curve was then generated (Figure 13.6).

Figure 13.6 Fluoride adsorption isotherm



Source: Tetra Tech

Crystallisation

Two routes were pursued for the basis of design development:

- use of synthetic solution produced from analytical reagent samples
- work at Mintek to produce a solution from representative ore that has been processed step by step through the intended recovery methods

The synthetic solution provided a means for rapid and inexpensive production of crystalliser feed, allowing the development of initial process feed purity requirements and expected operational criteria.

Synthetic solution

A synthetic solution was made up based on the analysis of the stock solution and the calculated values from the mass balance. The synthetic solution was then used to test the proposed crystallisation circuit.

Stock solution

A stock solution was made using the extraction and purification steps (prior to having jarosite precipitation purification step). The analysis of the stock solution can be found in Table 13.23.

Table 13.23 Crystalliser stock solution analysis without jarosite precipitation and alumina optimisation

Element	Units	Value
K	g/L	2.2
Al	ppm	23.3
Ca	ppm	33.5
Cd	ppm	<2.0
Co	ppm	<2.0
Cr	ppm	<2.0
Cu	ppm	<2.0
Fe	ppm	5.9
Li	ppm	<2.0
Mg	ppm	292.5
Mn	g/L	80.0
Mo	ppm	<2.0
Ni	ppm	7.4
Pb	ppm	9.6
S	g/L	53.3
Si	ppm	<2.0
Ti	ppm	<2.0
V	ppm	<2.0
Zn	ppm	<2.0
F	ppm	483.5

PPTech tested the synthetic solution with the composition stated in Table 13.23 and found that the product did not meet the proposed crystallisation method.

A new stock solution, using the OLOM sample, was made by carrying out all steps presented in the schematic (Figure 13.2). This work yielded a fully representative stock solution, which provided the opportunity to complete the design of the crystalliser and provide a significant amount of HPMSM for acceptance testing and trials by potential battery producers. The analysis of the stock solution is presented in Table 13.24.

Table 13.24 Final crystalliser stock solution analysis made from OLOM sample

Element	Units	Value
K	ppm	2.24
Na	g/L	1.33
Al	ppm	38.25
As	ppm	<2
Ca	ppm	19.05
Cd	ppm	<2
Co	ppm	<2
Cr	ppm	2.115
Cu	ppm	<2
Fe	ppm	6.44
Li	ppm	<2
Mg	ppm	277.5

Element	Units	Value
Mn	g/L	68.85
Mo	ppm	<2
Ni	ppm	<2
Pb	ppm	8.76
S	g/L	47.75
Si	ppm	10.85
Ti	ppm	<2
V	ppm	<2
Zn	ppm	<2
Se	ppm	<2
B	ppm	0.0562
Se	ppm	<0.01
F	ppm	37.5

With the inclusion of the jarosite precipitation, there was a thousand-fold reduction in the potassium concentration in the stock solution. With the alumina optimisation, there was a 20-fold improvement in the fluoride concentration below the required 50 ppm.

Crystal product

The stock solution produced was then processed through two crystallisation steps. The first, a crude crystallisation, with the crystals produced then being redissolved and going into the second crystallisation step to produce a pure product. The final step was filtering and washing the crystal product. The analysis of the final product is shown in Table 13.25.

Table 13.25 Washed and unwashed elemental composition of the pure MSM produced by crystallisation

Component	Unit (1)	Pure MSM		Specification
		Unwashed	Washed	
$MnSO_4.H_2O$	%	99.94	99.97	>99.93
Ca	ppm	44	26	<50
Mg	ppm	132	40	<100
Na	ppm	58	36	<300
F	ppm	33	25	<30
K	ppm	253	81	<200
Si	ppm	121	100	<200
Al	ppm	5.8	1.3	<10
Pb	ppm	-	1.1	<10
Cu	ppm	-	<0.1	<50
Ni	ppm	-	<1	<250
Fe	ppm	-	3.1	<10
Cr	ppm	-	1.1	<10
Co	ppm	-	1.1	<200
Zn	ppm	-	0.7	<50
Cd	ppm	-	<0.2	<10

Notes:
(1) dry basis

The product crystals exhibited a d50 crystal size of 220 µm. The results shown in Table 13.25 indicate that the washed pure MSM meets the desired product specification with the results confirming the validity of the crystallisation flowsheet.

Manganese hydroxide precipitation

With the size of the bleed stream, and the impact on the overall recovery of manganese, the bleed stream will be treated to recover manganese in another form and bring it back into the process. It was for this reason that hydroxide precipitation was tested for the recovery of manganese hydroxide. The tests were carried out using synthetic solution with two different bases (sodium hydroxide and lime [calcium hydroxide]) at different pHs.

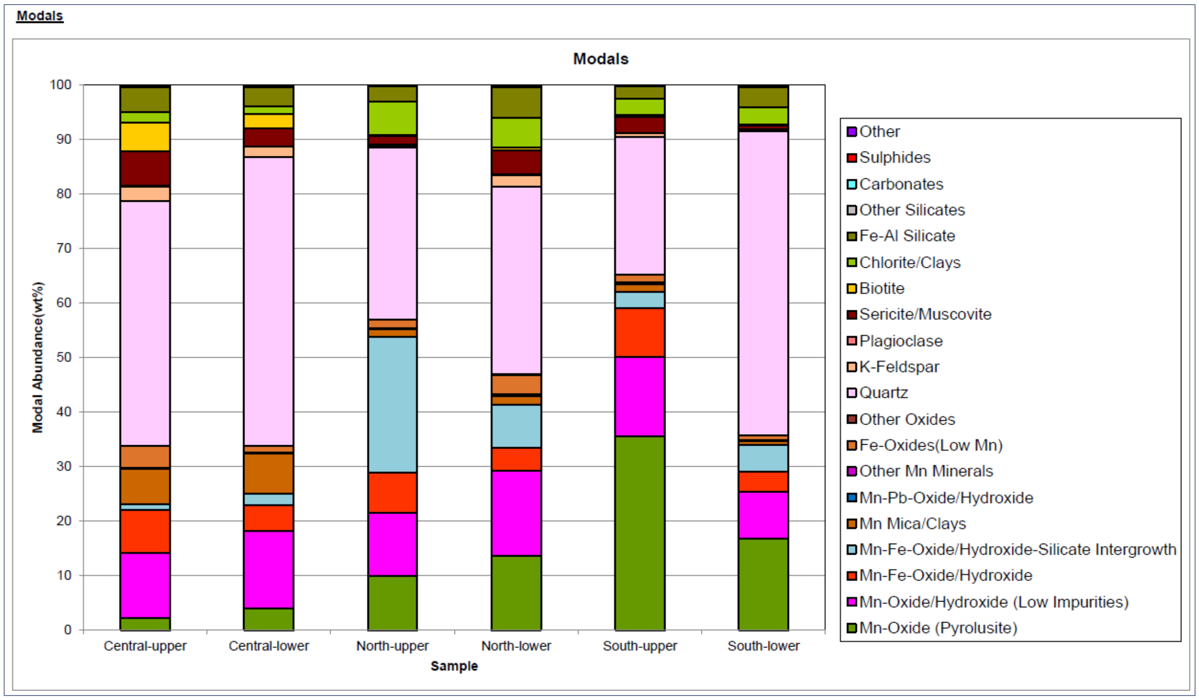
The results of the manganese precipitation using sodium hydroxide and lime were shown to be effective as a precipitating agent at a pH of 8.0 and above.

13.4.3 K.Hill B Horizon orebody

Mineralogy

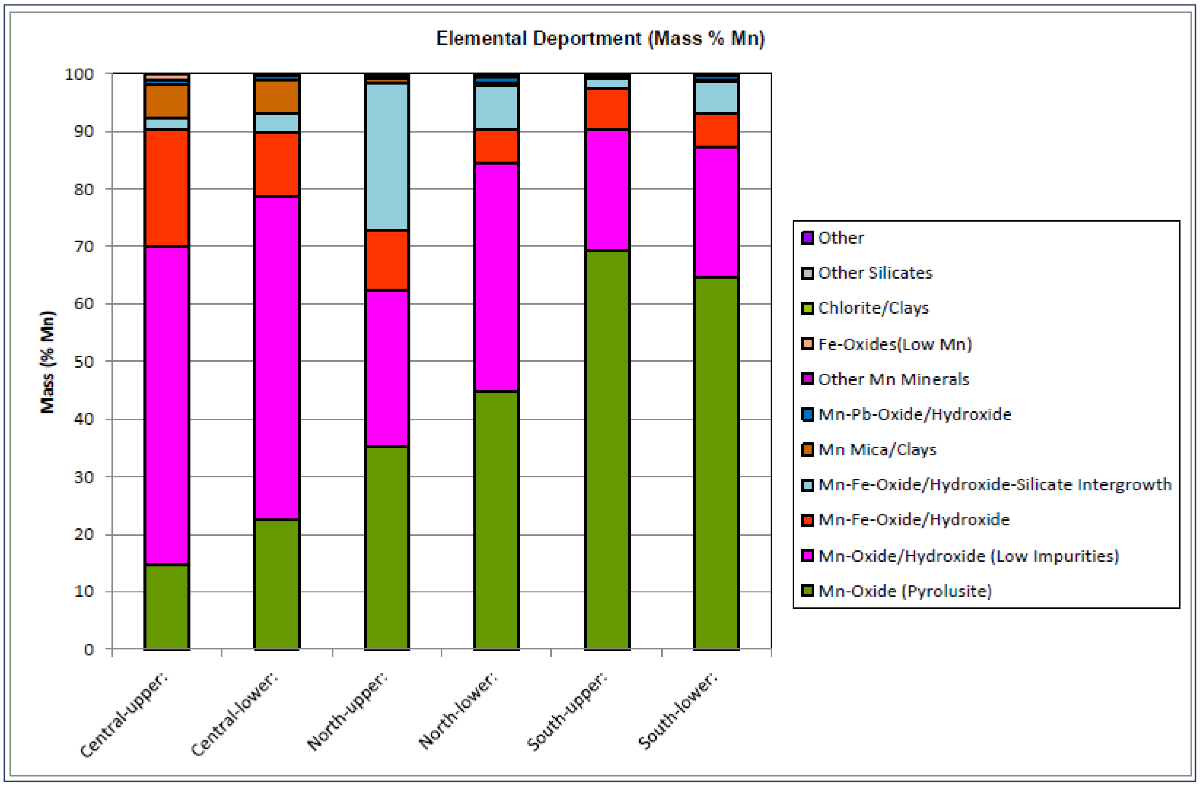
Figure 13.7 and Figure 13.8 show results from quantitative mineralogical work (QEMSCAN) undertaken by SGS Canada Inc (SGS Lakefield) in Lakefield, Ontario. Figure 13.7 shows the sample's mineralogical distribution and Figure 13.8 the manganese deportment. The mineralogy indicates that the K.Hill B Horizon material is not characteristically different to that of the OLOM composite sample, and the leach results are consistent with both the mineralogy and earlier leach tests.

Figure 13.7 B Horizon Samples Modal Mineral Distribution



Source: SGS Lakefield

Figure 13.8 B Horizon Manganese Distribution by Mineralogical Classification



Source: SGS Lakefield

Leach variability

The K.Hill B Horizon leach variability test work was carried out under the following conditions:

- grind P_{80} of 150 µm
- temperature of 90°C
- pH 1.0
- residence time of 6 hours

Summary leach extraction data from the Mintek K.Hill B Horizon test work programme is shown in Table 13.26.

Table 13.26 K.Hill B Horizon extraction results

Sample description	South upper	South lower	North upper	North lower	Central upper	Central lower
Head grade (%)						
Mn	27.80	14.00	13.10	17.10	10.10	9.33
Fe	6.25	4.59	9.01	8.55	9.21	6.19
Si	17.50	30.30	24.60	24.10	28.10	30.8
Al	2.38	1.45	3.34	3.10	2.97	2.02
Pb	0.15	0.09	0.08	0.08	0.08	0.13
Mg	0.29	0.22	0.31	0.28	0.42	0.26

Sample description	South upper	South lower	North upper	North lower	Central upper	Central lower
Extraction (%)						
Mn	98	99	99	99	93	87
Fe	29	29	3	15	5	7
Si	0	0	0	1	1	0
Al	12	11	8	8	6	6
Pb	11	14	17	-7	25	11
Mg	54	60	59	42	12	7
Reagent addition (kg/t)						
Total acid addition	159.9	139.6	159.9	127.0	92.5	78.4
Total SO_2 addition	272.3	145.6	200.4	180.9	122.6	91.3

The extractions were carried out on various samples selected from the ore body, as identified in the Table 13.26. The samples were subjected to a reducing sulphate leach under optimised leach conditions developed earlier for the OLOM composite sample. In general, manganese leach extractions were excellent, ranging between 87% and 99%. These compare well with the results from an earlier leach variability programme in which the average manganese leach extraction was 97%. The lower grade material approaching the manganese cut off does not leach quite as well, probably owing to the proportion of siliceous material (Figure 13.7). Nonetheless, extractions are acceptable.

Based on the foregoing, the metallurgical extraction characteristics of the B Horizon are not expected to differ significantly from those of the K.Hill North orebody. Cost may vary somewhat depending on levels of impurities, but the main difference in the material is in the proportion of siliceous material, which is inert and reports to tails.

It can be concluded that feeding B Horizon material to the processing plant is unlikely to have an appreciable impact on metallurgical extraction or plant operations in general.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

CSA Global completed an MRE for Giyani's K.Hill Project deposit. Ms. Susan Oswald (Senior Resource Consultant) conducted the modelling, estimation, and Mineral Resource classification. Mr. Anton Geldenhuys (Principal Resource Consultant) peer reviewed the MRE and is the QP.

The current MRE has an effective date of 6th June 2023, was prepared in accordance with *CIM Definitions and Standards on Mineral Resources & Mineral Reserves* (CIM 2014), and is reported in accordance with NI 43-101 *Standards of Disclosure for Mineral Projects* (CSA 2016a), Companion Policy 43-101CP (CSA 2016b), and Form 43-101F1 (CSA 2011).

This MRE is based on interpretations from assays and geological logging. Apart from the initial sample data preparation and intermediate spreadsheet processing, all interpretations, modelling, estimation, and model validation was conducted using Leapfrog™ and Datamine StudioRM™ software. Snowden Supervisor™ software was used to conduct statistical analysis.

The MRE workflow can broadly be summarised as follows:

- data validation and preparation
- interpretation of the geology and mineralisation domains
- coding, compositing, and capping of composites
- exploratory data analysis and statistical analysis
- variogram analysis
- block model construction
- grade interpolation
- block model validation
- density assignment
- reasonable prospects for eventual economic extraction
- Mineral Resource classification and Mineral Resource reporting

Reported Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no guarantee that all or any part of a Mineral Resource will be converted into a Mineral Reserve.

14.2 Database cut-off

The database is currently housed in Microsoft® Excel spreadsheets and were compiled by Giyani. Historical data were compiled in a different Microsoft® Excel spreadsheet and were combined with the 2022 drilling data. The data were exported in comma separated values (.csv) format and imported into Leapfrog™. The data used in the MRE were compiled from the Microsoft® Excel spreadsheets. The K.Hill Project MRE was prepared using data available up to 15th November 2022.

The following data were available:

- collar
- survey
- assay
- lithology
- density
- core recovery
- QA/QC
- topography

A total of 234 holes were imported. A high-level database validation was completed and it was found that the database was in good condition. Key fields within the drillhole data files were validated for potential numeric and alphanumeric errors. Data validation cross referencing collar, survey, assay, and geology files was performed to confirm drillhole depths, inconsistent or missing sample/logging intervals, and survey data. The data were validated and checked for logical or transcription errors, such as overlapping intervals. A few minor errors were encountered and corrected. Collar elevations were compared with the supplied topography and projected onto the topography surface where necessary. Sample distributions were reviewed relative to the expected nature of the mineralisation.

All drilling data is supported by a QA/QC program. The current MRE is supported by an additional 82 holes drilled since the previous MRE (Figure 14.1), which includes 75 RC holes and 7 DD holes (including redrills and twin holes), as discussed in Section 10.

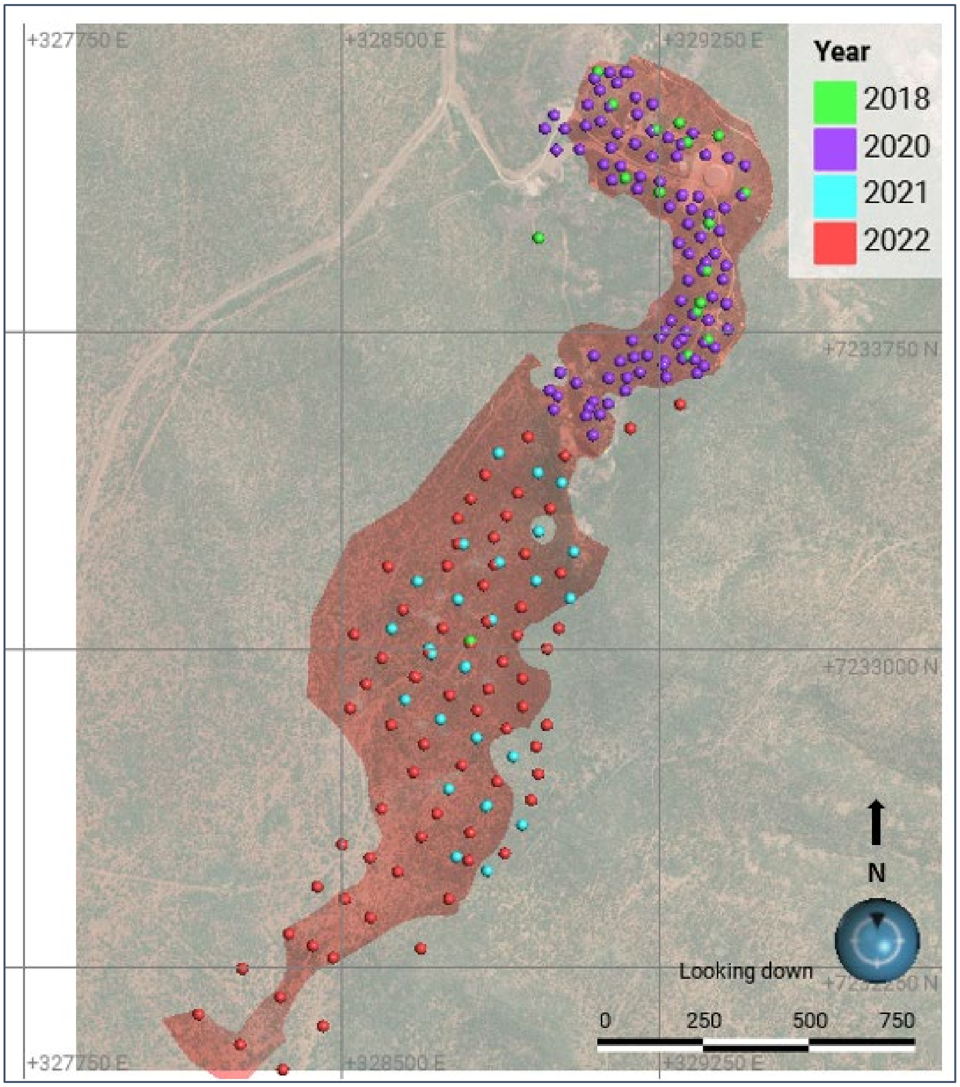
14.2.1 Core recovery

The Mn-shale is soft and weathered, within which harder layers of manganese oxide mineralisation occur at centimetre scale. Recovery was calculated by dividing the drilled length by the recovered length. In the first phase of drilling (2018), poor core recoveries were observed in most of the intersections of the Mn-shale, averaging approximately 50% in the high-grade mineralisation. Improved recoveries were achieved during the second phase of DD (2020-2022), with recoveries averaging 91% in the Mn-shale horizons. For the third and fourth phases, recoveries were similar averaging 90%.

14.2.2 Data excluded

Giyani drilled twin holes to test the variance between the DD and RC drilling. For modelling and estimation, the RC twin holes were excluded and DD holes included. A twin analysis was completed and is discussed in Section 10.

Figure 14.1 Drill collars relative to the modelled mineralisation



Source: CSA Global

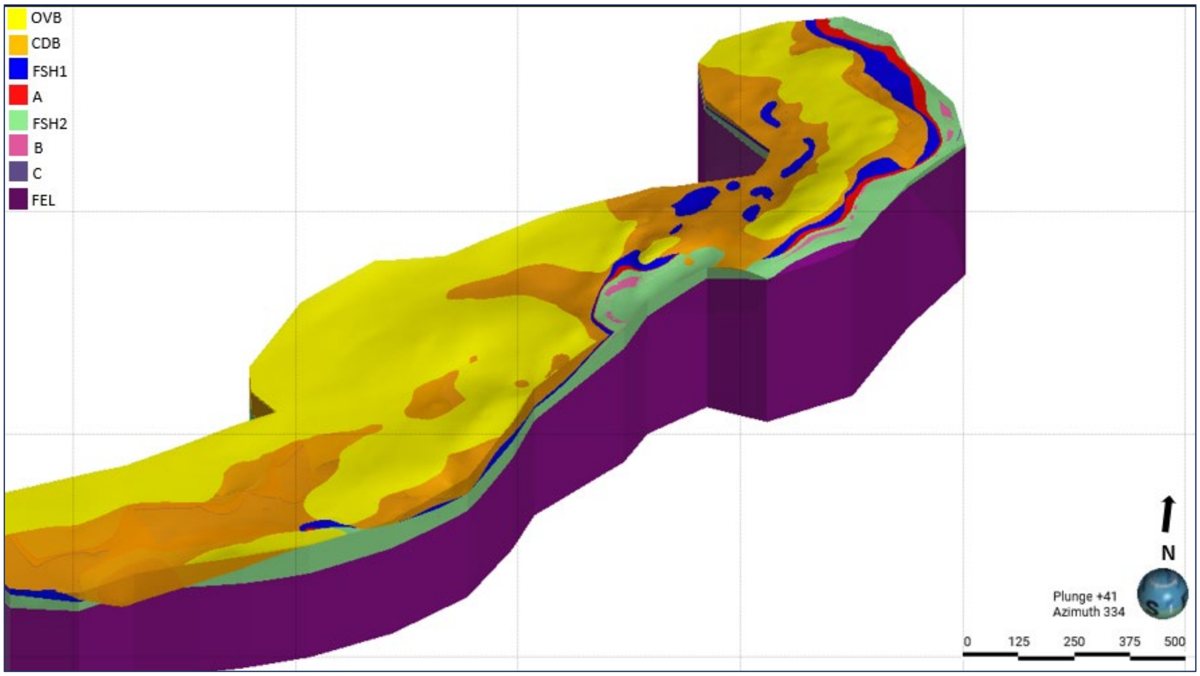
14.3 Preparation of wireframes

14.3.1 Lithological model

The lithology was modelled in Leapfrog™ and was based on the logged lithological units. The lithological units were simplified and modelled across the entire K.Hill Project area (Figure 14.2). CSA Global applied an adjustment to the contact location of the Mn-shale horizons to snap the contacts of the wireframes to the assays for manganese oxide, to adhere to a modelling threshold of approximately 7% manganese oxide for the mineralised Mn-shale horizons. This modelling threshold was chosen, as it appears to be a natural threshold between the non-mineralised sediments and the low-grade Mn-shale margins. The following units (from hanging wall to footwall) were modelled:

- overburden (OVB)
- chert dolomite breccia (CDB)
- Fe-shale (FSH 1)
- upper Mn-shale (A)
- lower Mn-shale (B)
- lower discontinuous Mn-shale (C)
- lower iron shale (FSH 2)
- felsite (FEL)

Figure 14.2 Lithological model



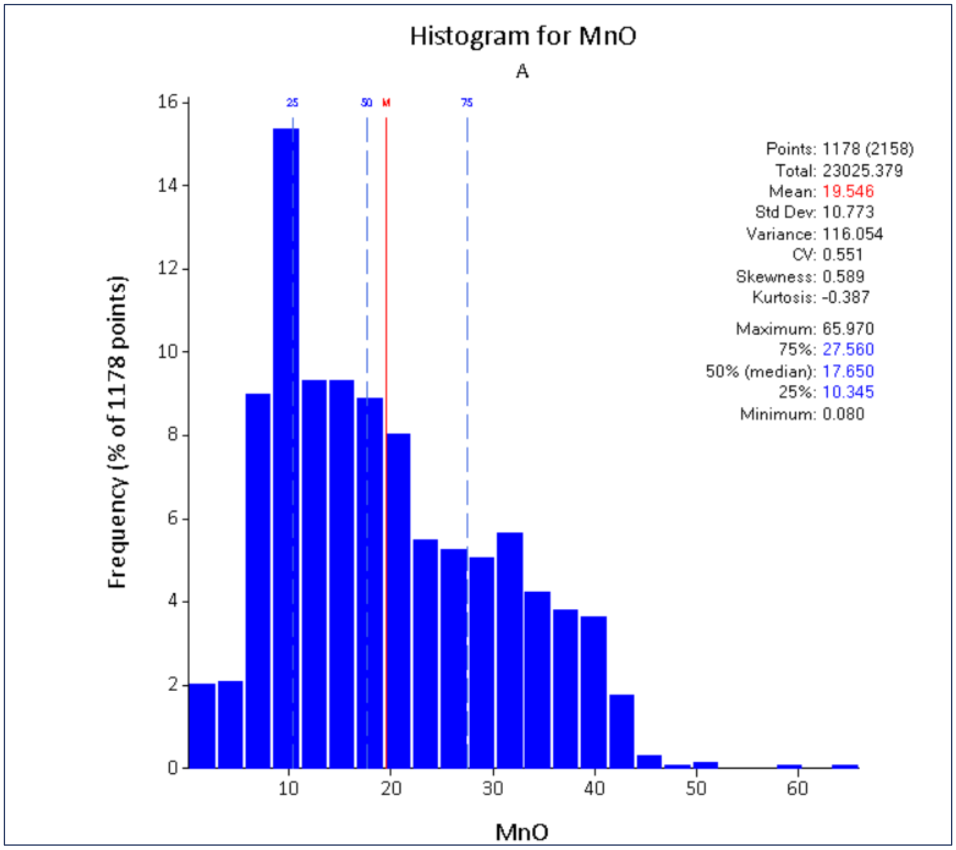
Source: CSA Global

14.3.2 Mineralisation domains

The mineralisation domains were interpreted from the lithological model. The upper Mn-shale consists of one continuous domain A and a smaller geometrically higher domain AA in the southern area. The lower Mn-shale consists of domain B. Domain B is fairly continuous throughout, with areas of discontinuity. A lower discontinuous Mn-shale or domain C exists below domain B. The mineralised domains are split with unmineralized (below <7% manganese oxide) zones included in the iron shales (Fe-shales).

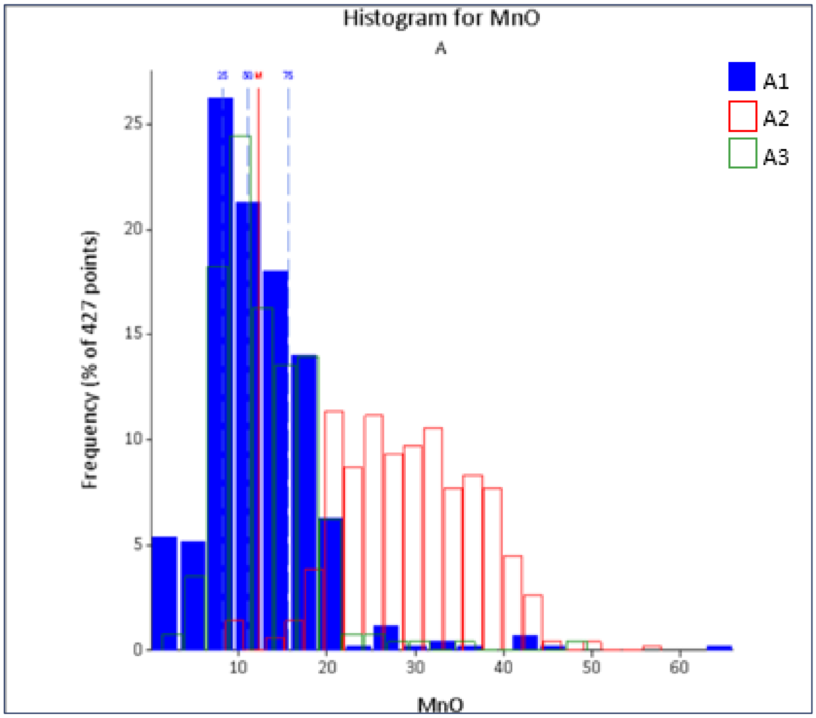
CSA Global conducted a statistical analysis of the manganese oxide assays within each of the lithological domains. This highlighted a bimodal manganese population within the upper Mn-shale domain A. After a review of the two populations in 3D and in sections, CSA Global decided to model low-grade horizons (A1 and A3) above and below a high-grade domain core (A2) within domain A, based on a 20% manganese oxide threshold. The high-grade core of the upper Mn-shale is generally continuous across the deposit area, except for areas where it has eroded. The lower-grade domains are largely continuous but are absent in some areas. Figure 14.3 shows a histogram of the sample manganese oxide grades for the entire A domain, showing a clear bimodal distribution. Figure 14.4 shows the resulting histograms once the three domains have been separated.

Figure 14.3 Histogram of the manganese oxide population for domain A, showing the bimodal distribution



Source: CSA Global

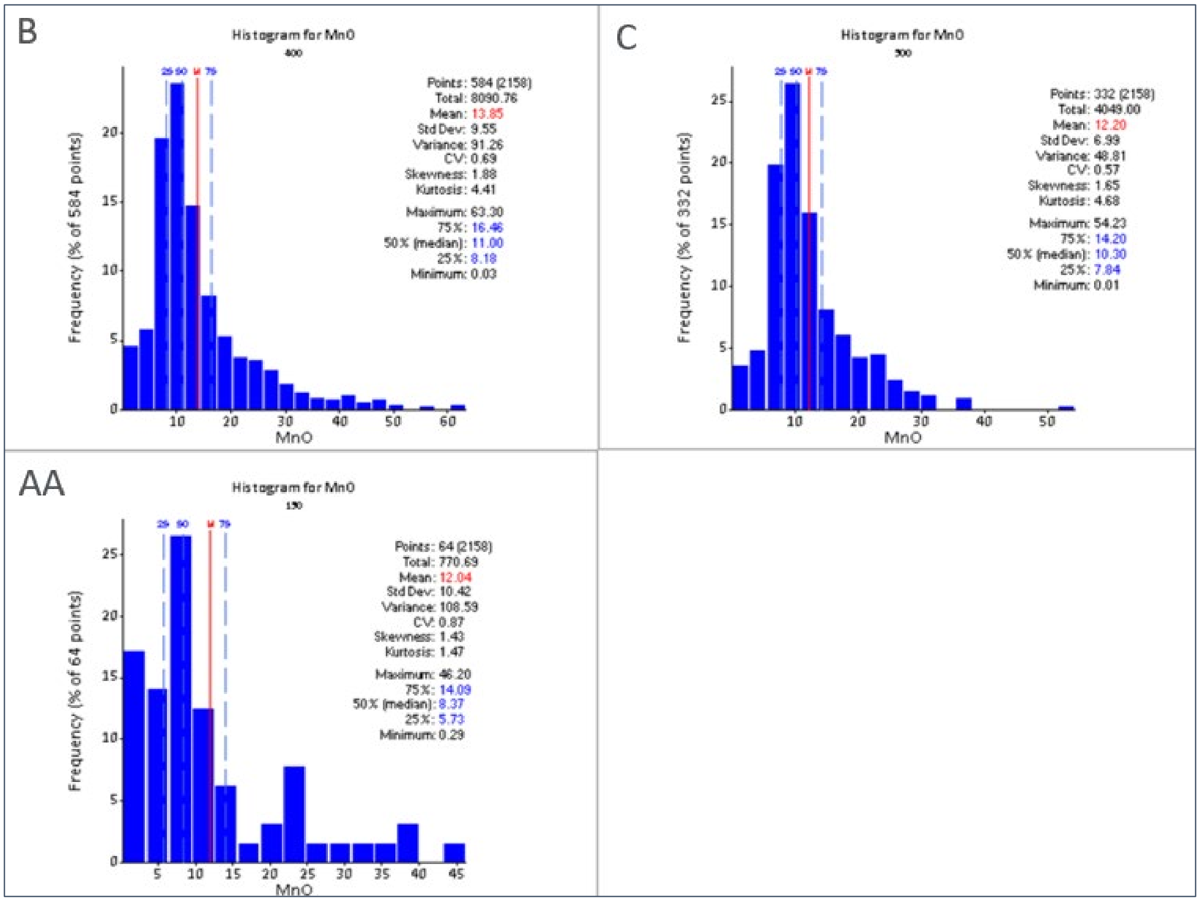
Figure 14.4 Histogram showing the A1, A2, A3 domains after splitting domain A



Source: CSA Global

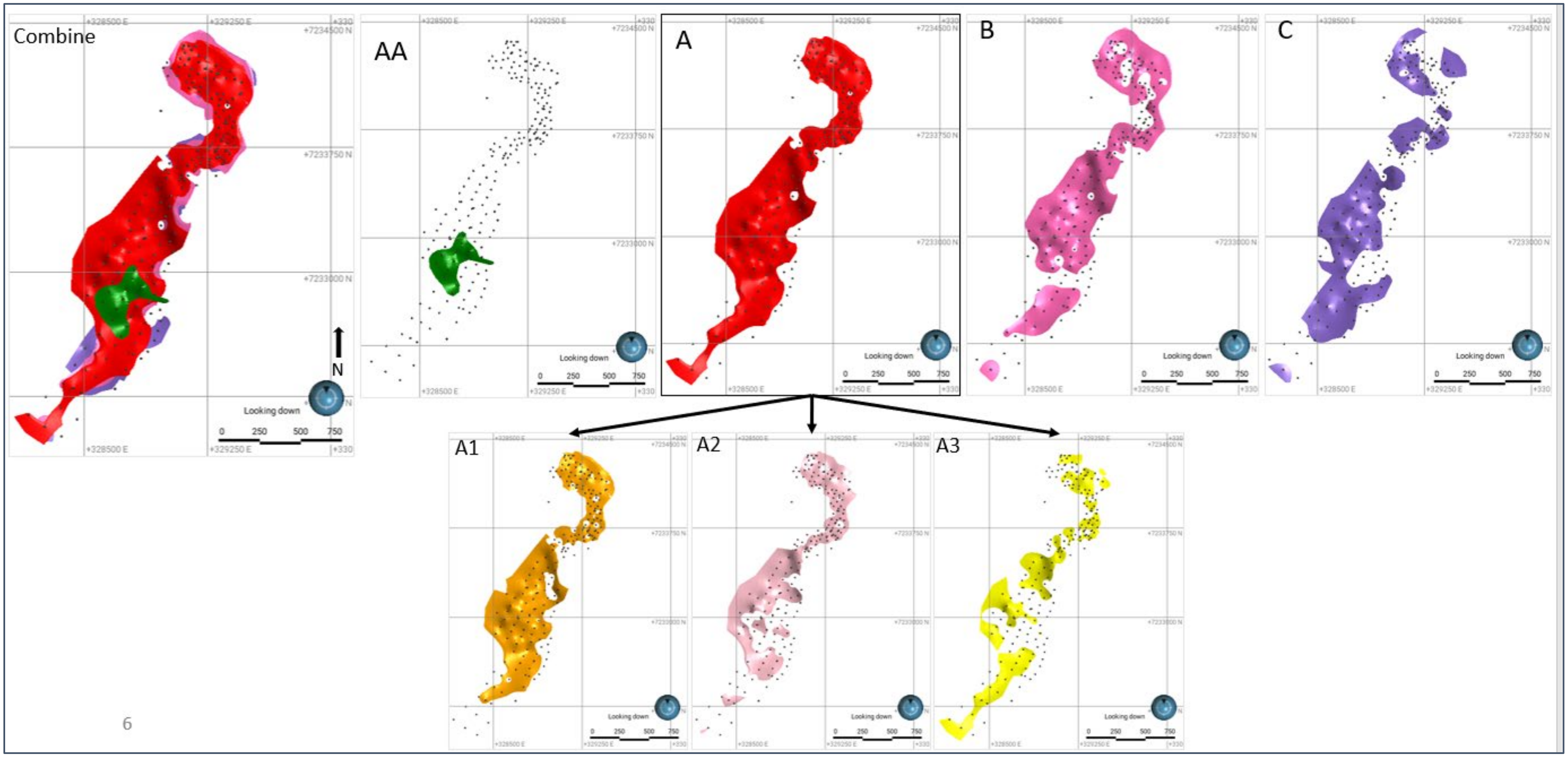
A similar approach was used for domains B and C, but they do not comprise multiple manganese oxide populations; therefore they did not require further sub-domaining. Domain AA did not comprise enough sample points and was modelled as a single domain. Figure 14.5 shows the histograms of manganese oxide values within the domains B, C, and AA. Figure 14.6 shows the final estimation domains.

Figure 14.5 Histograms for domains B, C, and AA



Source: CSA Global

Figure 14.6 Mineralisation domains showing the split of domain A into A1, A2, and A3



Source: CSA Global

14.3.3 Structure

Faults were previously interpreted in the Project area from mapping and geophysical data. No detailed information about the faults, nor the impact on mineralisation, was available. Therefore, the faults could not be verified by the QP and were excluded from the final model.

An alternative interpretation, including the faults, was modelled. The mineralised volumes of the faulted and unfaulted model were comparable, and a decision was made to exclude the faults from the final model. It is advisable to collect more information about the faults for possible inclusion in future model updates.

In constructing the geological model for the K.Hill Project, an area in the north of the deposit was identified that showed a significant thickening of the CDB unit. It appears that the thickening of the CDB unit replaced a section of the stratigraphy of the deposit where a large section of the central stratigraphy, including the upper mineralised horizon, has been replaced. Further drilling should allow the extents and margins of this feature to be more clearly defined. It should be considered that the nature of this contact warrants further investigation as part of a more in-depth structural interpretation or mapping exercise. The CDB unit was modelled using the available lithology data.

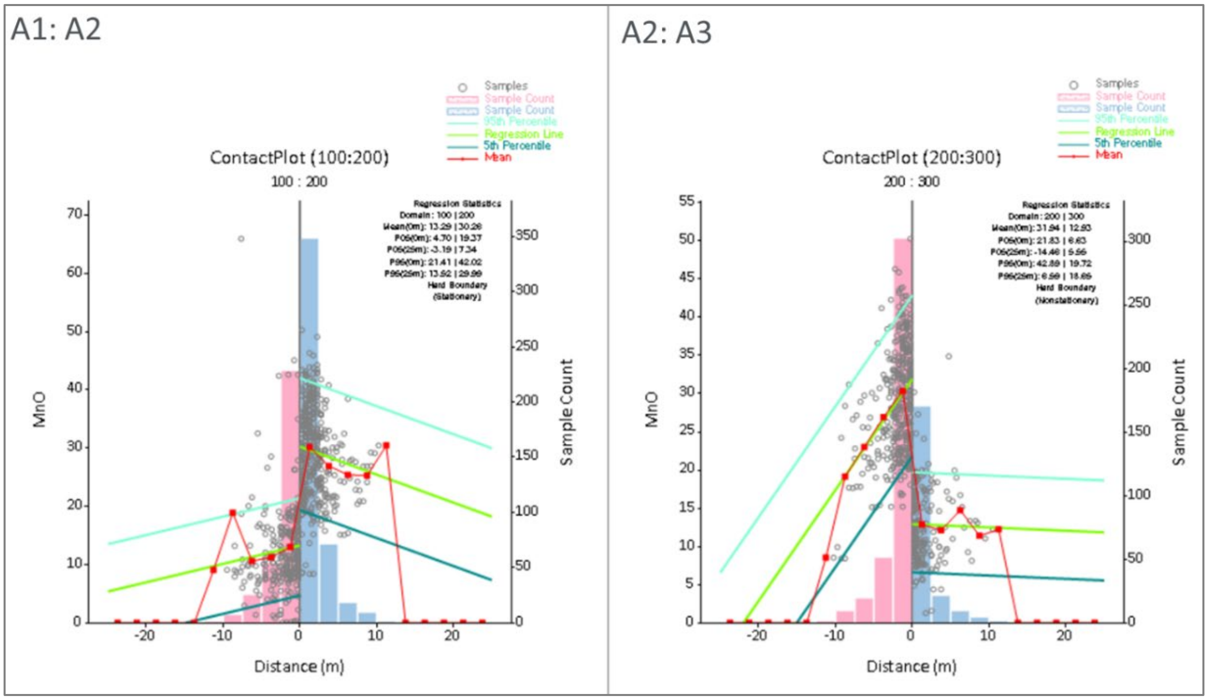
14.3.4 Topography

DGPS collar surveys and a high-resolution satellite survey surface provided topographic constraints to the model.

14.3.5 Boundary analyses

Statistical analysis showed that there are differences in grade distribution between the four estimation domains. Figure 14.4 and Figure 14.5 show that the histograms have normal distributions. The A domain has higher grades, with the A2 showing the highest mean grade. A boundary analysis investigation between A1 and A2 and A2 and A3 show the presence of hard boundaries (Figure 14.7).

Figure 14.7 Boundary analyses for the A sub-domains



Source: CSA Global

14.3.6 Coding and compositing

All samples within the estimation domains were flagged with estimation domain code (MINZON) as below:

- 100: A1
- 150: AA
- 200: A2
- 300: A3
- 400: B
- 500: C

Only samples within mineralised domains were used for Mineral Resource estimation. Drillhole samples were composited to 0.5 m intervals. During the compositing process in Datamine software, the MODE parameter was set to 1. This allowed the process to force all samples to be included in one of the composites by adjusting the composite length, while keeping it as close as possible to the interval (0.5 m). The maximum possible composite length is 1.5× the selected interval length, i.e., 0.75 m. The MODE parameter reduces the proportion of residual samples that would have been excluded from the estimate if forced to a single continuous interval composite length.

14.4 Global and domain statistics

Raw assay samples and 0.5 m composites were selected within each of the mineralised domains and statistics were calculated. Table 14.1 and Table 14.2 show the descriptive statistics per estimation domain for manganese oxide (%), aluminium dioxide (%), silicon dioxide (%), iron oxide (%), and LOI (%) for raw samples and composites, respectively.

Table 14.1 Summary statistics of raw samples

Domain	MINZON	Variable	Number of samples	Minimum	Mean	Maximum	Coefficient of variation
A1	100	MnO (%)	421	0.08	12.38	65.97	0.57
AA	150		64	0.29	12.04	46.20	0.87
A2	200		493	8.23	29.26	57.90	0.26
A3	300		258	1.27	12.87	49.60	0.41
B	400		584	0.03	13.82	63.30	0.69
C	500		333	0.01	12.24	54.23	0.57
A1	100	Al_2O_3 (%)	421	0.64	9.01	22.60	0.45
AA	150		64	2.06	6.34	13.95	0.50
A2	200		493	1.65	8.81	18.30	0.32
A3	300		258	1.44	9.55	21.72	0.46
B	400		584	0.87	9.39	22.30	0.53
C	500		333	0.77	9.28	19.84	0.43
A1	100	SiO_2 (%)	421	13.73	57.43	90.07	0.24
AA	150		64	23.56	66.21	91.13	0.27
A2	200		493	5.75	27.32	72.54	0.51
A3	300		258	14.50	52.86	82.09	0.24
B	400		584	5.64	53.30	93.25	0.28
C	500		333	13.32	51.05	83.99	0.26
A1	100	Fe_2O_3 (%)	421	0.01	9.84	44.58	2.64
AA	150		64	0.01	6.72	28.13	2.23
A2	200		493	0.01	17.52	31.30	1.54
A3	300		258	0.01	12.24	29.90	2.12
B	400		584	0.01	10.78	48.40	2.41
C	500		333	0.01	13.54	57.10	2.58
A1	100	LOI (%)	421	0.97	5.89	12.74	2.55
AA	150		64	1.24	4.96	11.66	2.22
A2	200		493	3.91	9.73	17.30	1.54
A3	300		258	2.46	6.45	32.29	2.63
B	400		584	0.87	6.49	27.28	2.31
C	500		333	2.14	7.25	33.40	1.79

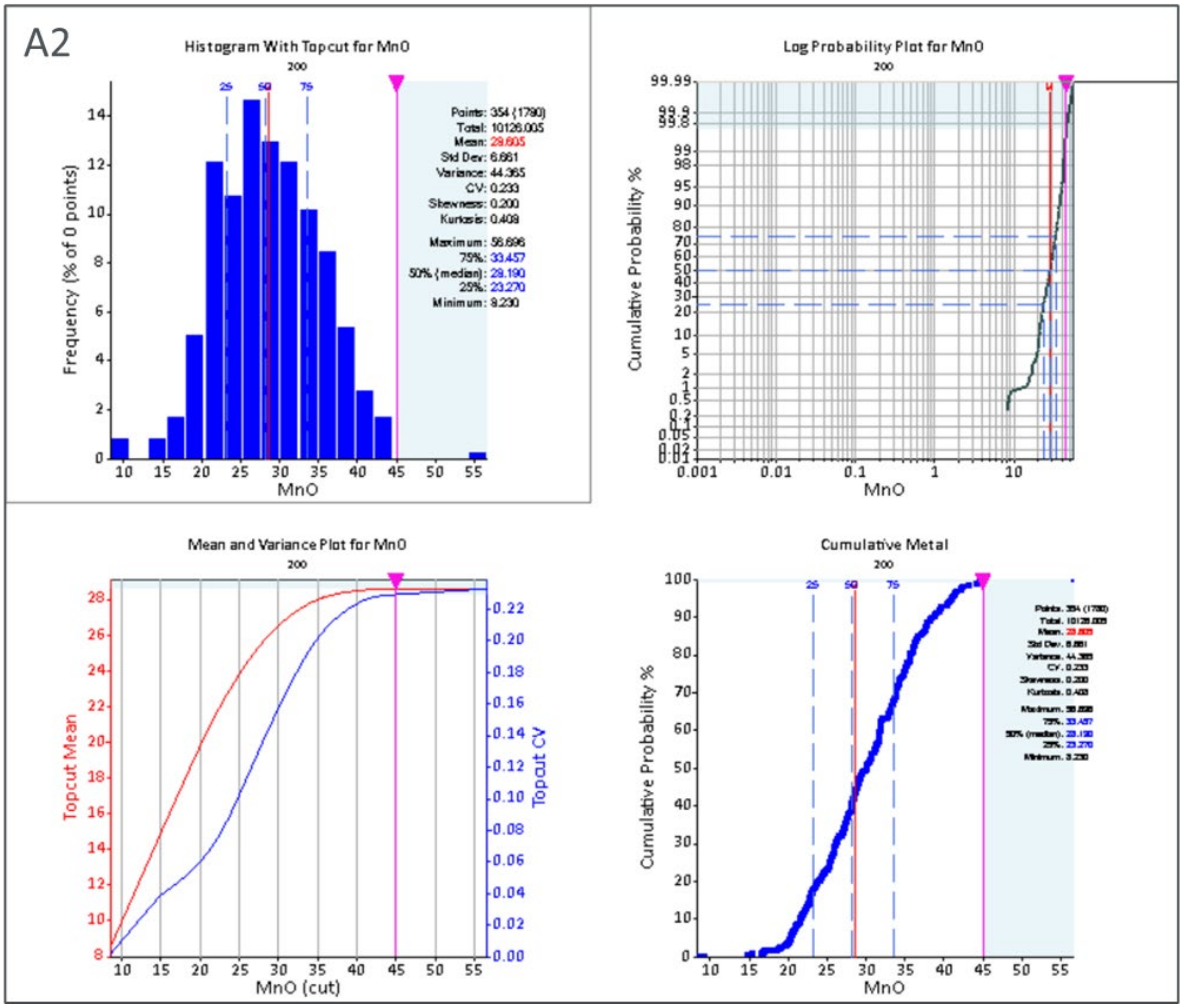
Table 14.2 Summary statistics of 0.5 m composites

Domain	MINZON	Variable	Number of samples	Minimum	Mean	Maximum	Coefficient of variation
A1	100	MnO (%)	616	0.08	12.15	50.30	0.52
AA	150		116	0.29	12.31	46.20	0.85
A2	200		597	8.23	28.82	57.90	0.26
A3	300		353	1.27	12.86	49.60	0.39
B	400		820	0.03	13.01	63.30	0.70
C	500		514	0.01	12.05	54.23	0.57
A1	100	Al_2O_3 (%)	616	0.64	8.54	22.60	0.47
AA	150		116	2.06	6.54	13.95	0.49
A2	200		597	1.65	8.40	18.30	0.34
A3	300		353	1.44	9.39	21.72	0.48
B	400		820	0.87	9.28	21.49	0.54
C	500		514	0.77	9.34	19.84	0.44
A1	100	SiO_2 (%)	616	14.20	58.50	90.07	0.23
AA	150		116	23.56	65.48	91.13	0.27
A2	200		597	5.75	29.09	72.54	0.49
A3	300		353	14.50	53.29	82.09	0.25
B	400		820	5.64	54.74	93.25	0.28
C	500		514	13.32	51.20	83.99	0.26
A1	100	Fe_2O_3 (%)	616	0.01	9.64	41.34	0.60
AA	150		116	0.01	6.78	28.13	0.90
A2	200		597	0.01	16.76	31.30	0.36
A3	300		353	0.01	11.77	29.90	0.47
B	400		820	0.01	10.26	48.40	0.61
C	500		514	0.01	13.22	57.10	0.57
A1	100	LOI (%)	616	1.41	5.78	12.74	0.34
AA	150		116	1.24	5.05	11.66	0.52
A2	200		597	3.91	9.58	17.30	0.21
A3	300		353	2.46	6.62	32.29	0.54
B	400		820	0.87	6.30	27.28	0.43
C	500		514	2.14	7.17	33.40	0.46

14.5 Treatment of outliers (top cuts)

Top cuts were applied after compositing. These top cuts were quantified according to the statistical distribution of the sample population. Histograms and log normal cumulative probability plots for each of the domains were reviewed to identify inflection points at the upper end of the distribution and derive a capping value. Capping was used only where an abrupt increment in grade was identified. Figure 14.8 shows the graphs used for establishing the top cut values for the A2 domain for manganese oxide. Summary composite statistics by domain and the impact of top cuts are shown in Table 14.3.

Figure 14.8 Top capping analyses for the A2 domain for manganese oxide (%)



Source: CSA Global

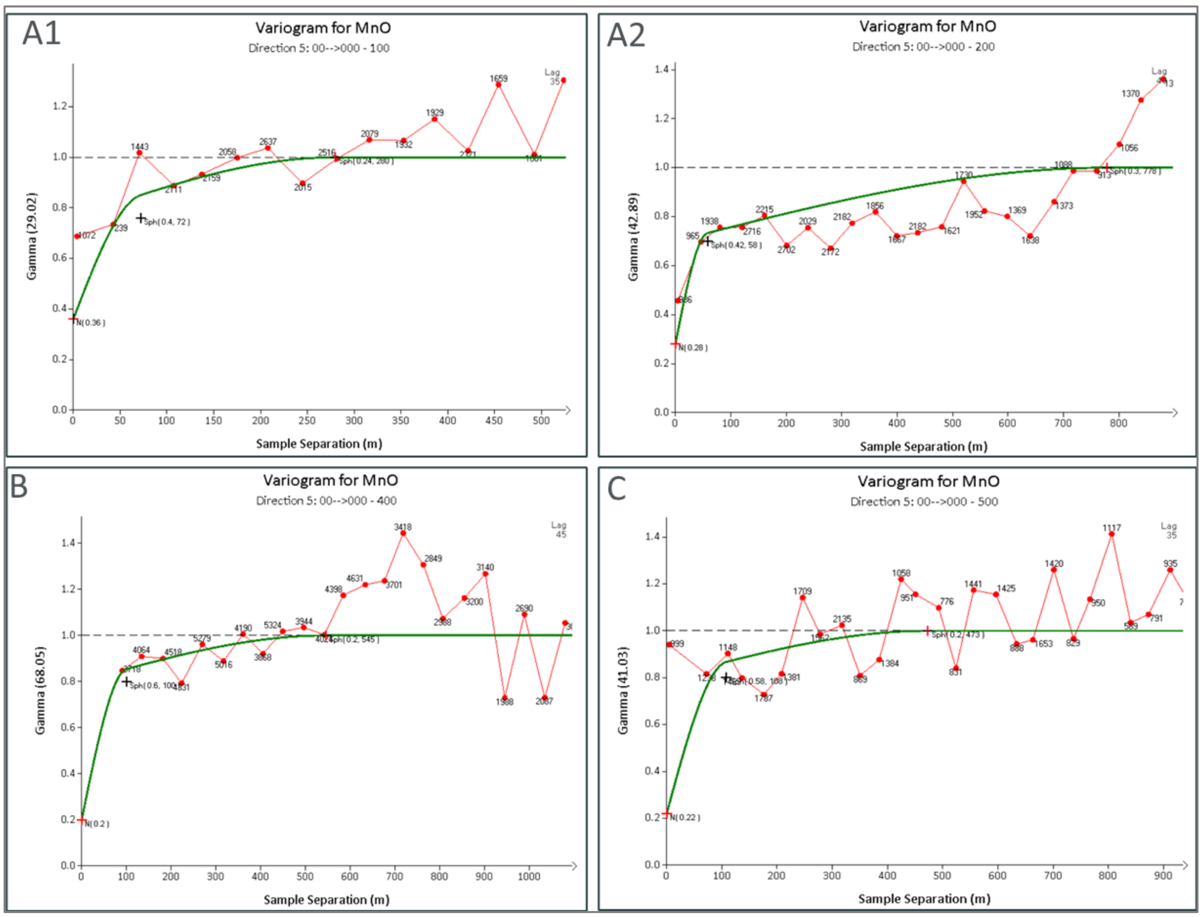
Table 14.3 Top cuts per domain and variable

Domain	MINZON	Variable	Top cut	Mean before	Mean after capping	Coefficient of variation before capping	Coefficient of variation after capping
A1	100	MnO (%)	30	12.15	11.96	0.52	0.46
AA	150		25	12.31	11.11	0.85	0.71
A2	200		45	28.82	28.77	0.26	0.25
A3	300		31	12.86	12.80	0.39	0.37
B	400		50	13.01	12.97	0.70	0.68
C	500		36	12.05	12.01	0.57	0.56
A1	100	Al_2O_3 (%)	-	8.54	8.54	0.47	0.47
AA	150		-	6.54	6.54	0.49	0.49
A2	200		-	8.40	8.40	0.34	0.34
A3	300		-	9.39	9.39	0.48	0.48
B	400		-	9.28	9.28	0.54	0.54
C	500		-	9.34	9.34	0.44	0.44
A1	100	SiO_2 (%)	-	58.50	58.50	0.23	0.23
AA	150		-	65.48	65.48	0.27	0.27
A2	200		-	29.09	29.09	0.49	0.49
A3	300		-	53.29	53.29	0.25	0.25
B	400		-	54.74	54.74	0.28	0.28
C	500		-	51.20	51.20	0.26	0.26
A1	100	Fe_2O_3 (%)	26	9.64	9.55	0.60	0.57
AA	150		15	6.78	6.20	0.90	0.76
A2	200		27	16.76	16.72	0.36	0.36
A3	300		26	11.77	11.75	0.47	0.46
B	400		26	10.26	10.20	0.61	0.60
C	500		35	13.22	13.15	0.57	0.55
A1	100	LOI (%)	15	5.78	5.78	0.34	0.34
AA	150		11	5.05	5.03	0.52	0.51
A2	200		15	9.58	9.57	0.21	0.20
A3	300		17	6.62	6.46	0.54	0.42
B	400		15	6.30	6.27	0.43	0.41
C	500		13	7.17	7.00	0.46	0.34

14.6 Variography

Experimental (directional) variograms were calculated in various orientations along strike, down dip, and perpendicular to the dip direction. The directional variograms did not show any preferential orientations and were impacted by the data configuration. Omni-directional variograms were investigated. The omni-directional variograms displayed better stability; therefore, it was decided to model omni-directional variograms. Nugget values were modelled from downhole variograms and reviewed for reasonableness. Variogram analyses were completed for each variable in each domain. Figure 14.9 shows examples of variograms modelled for manganese oxide. Table 14.4 summarises the variogram parameters. Due to the lack of data in domain AA, domain A1 variogram parameters were used for the estimation of domain AA.

Figure 14.9 Examples of the modelled omni-directional variograms



Source: CSA Global

Table 14.4 Omni-directional variogram parameters

Domain	Variable	Nugget	Range 1	Structure 1 Sill	Range 2	Structure 2 Sill
A1	MnO (%)	0.36	72	0.40	280	0.24
	Al_2O_3 (%)	0.22	33	0.36	848	0.42
	SiO_2 (%)	0.30	62	0.39	765	0.31
	Fe_2O_3 (%)	0.23	57	0.55	860	0.22
	LOI (%)	0.28	230	0.72	280	0.24
A2	MnO (%)	0.28	58	0.42	778	0.30
	$Al2O_3$ (%)	0.28	835	0.72	-	-
	SiO_2 (%)	0.22	145	0.22	655	0.56
	Fe_2O_3 (%)	0.26	85	0.16	900	0.58
	LOI (%)	0.23	165	0.49	930	0.28
	MnO (%)	0.40	80	0.35	250	0.25
A3	Al_2O_3 (%)	0.20	107	0.50	382	0.30
	SiO_2 (%)	0.20	60	0.66	465	0.14
	Fe_2O_3 (%)	0.30	113	0.27	375	0.43
	LOI (%)	0.23	135	0.41	370	0.36
	MnO (%)	0.20	100	0.60	545	0.20
	Al_2O_3 (%)	0.20	52	0.47	203	0.33
B	SiO_2 (%)	0.25	78	0.68	524	0.07
	Fe_2O_3 (%)	0.22	74	0.58	700	0.20
	LOI (%)	0.22	100	0.42	485	0.36
	MnO (%)	0.22	108	0.58	473	0.20
	Al_2O_3 (%)	0.20	100	0.69	390	0.11
	SiO_2 (%)	0.20	108	0.67	415	0.13
C	Fe_2O_3 (%)	0.20	111	0.71	536	0.09
	LOI (%)	0.20	77	0.53	232	0.27
	MnO (%)	0.36	72	0.40	280	0.24
	Al_2O_3 (%)	0.22	33	0.36	848	0.42
	SiO_2 (%)	0.30	62	0.39	765	0.31
	Fe_2O_3 (%)	0.23	57	0.55	860	0.22

14.7 Block model

A block model was constructed with cell dimensions of 20 m × 20 m × 1 m (XYZ) within an area that covers the Project area (Table 14.5). The block sizes were selected based on half the average drillhole spacing. The wireframes representing the mineralisation boundaries were filled with cells to a minimum sub-cell size of 5 m × 5 m × 0.5 m to fill the volumes with blocks. The blocks were coded according to the appropriate domain codes.

Table 14.5 Block model definition

Model definition parameter	Value
Parent block X	20
Parent block Y	20
Parent block Z	1
Lower left corner, east coordinate	327,920
Lower left corner, north coordinate	7,231,600
Lower left corner, elevation coordinate	1,000
Number of panels along east direction	100
Number of panels along north direction	164
Number of panels along RL direction	600

14.8 Grade interpolation

Grade variables (manganese oxide [%], aluminium oxide [%], silicon dioxide [%], iron oxide [%] and LOI [%]) were interpolated within each domain using ordinary kriging (OK). Search distance and maximum and minimum number of samples were determined using the variogram ranges and drillhole spacing, respectively. The search ellipse was aligned with the dip and plunge to support domain layering. Estimation of the grade variables was carried out into parent cell panels. Hard boundaries between mineralisation domains were used during grade estimation. The estimation was performed using a 3 × 3 × 3 discretization. A minimum of 6 and a maximum of 12 composites were used to derive an estimate. Only four samples per drillhole could be used in the estimation. Composite selection was controlled by the maximum number of samples rather than the search ranges (which would have included more distant composites had the maximum number of allowable composites been increased).

A three-phased search pass was applied. This process involved the estimation being performed three times, in which two expansion factors were used. During each individual estimation run, this factor increased the size of the search ellipse used to select composites. The method ensured that blocks that were not estimated and populated with a grade value in the first run were populated during one of the subsequent runs. Search parameters are summarised in Table 14.6 and first pass search distances are summarised in Table 14.7. Almost all blocks were estimated using the first and second pass.

Table 14.6 Search parameters for estimation

Parameter	Value
Minimum number of composites	6
Maximum number of composites	12
Maximum number of composites per drillhole	4
Size factor for second pass	2
Size factor for third pass	3

Table 14.7 First pass search distances per domain and per variable

Domain	Variable	Search distance X	Search distance Y	Search distance Z
A1	MnO (%)	280	280	30
	Al_2O_3 (%)	625	625	30
	SiO_2 (%)	620	620	30
	Fe_2O_3 (%)	650	650	30
	LOI (%)	677	677	30
A2	MnO (%)	760	760	30
	Al_2O_3 (%)	835	835	30
	SiO_2 (%)	655	655	30
	Fe_2O_3 (%)	900	900	30
	LOI (%)	930	930	30
A3	MnO (%)	250	250	30
	Al_2O_3 (%)	382	382	30
	SiO_2 (%)	465	465	30
	Fe_2O_3 (%)	375	375	30
	LOI (%)	370	370	30
B	MnO (%)	545	545	30
	Al_2O_3 (%)	235	235	30
	SiO_2 (%)	524	524	30
	Fe_2O_3 (%)	600	600	30
	LOI (%)	485	485	30
C	MnO (%)	235	235	30
	Al_2O_3 (%)	540	540	30
	SiO_2 (%)	415	415	30
	Fe_2O_3 (%)	536	536	30
	LOI (%)	415	415	30

14.9 Estimation validation

Estimated grades were validated per domain and were validated using:

- global statistics
- alternative estimate: inverse weighted distance to the power of two (IDW^2)
- swath analyses, to identify local over and under estimation and smoothing
- localised visual validation on sections

14.9.1 Global statistics and IDW^2

Global mean values were calculated for the input composites and output estimates (Table 14.8). Following Mineral Resource classification (see Section 14.11) the comparison was also completed for Indicated (Table 14.9) and Inferred areas (Table 14.10). The mean composite grades were compared to the block grades estimated using OK per estimation domain. Comparison of the mean grades showed that some of the grade variable estimates varied from the input data but are in the expected range. Due to the spacing of the drillhole data (more drilling in the northern area) the mean block grades were compared to a weighted input grade. The Indicated areas show better correlation for most variables than the Inferred areas, as would be expected. Domain C shows the highest variability and might be due to the domain being more variable.

Table 14.8 Weighted mean composite grades vs. the block model grades and IDW^2

Domain	Variable	Composite mean grade	Block mean grade (OK)	Relative difference (%)	IDW^2
AA	MnO (%)	10.83	10.32	-5.0	10.30
A1		12.40	12.36	-0.3	12.29
A2		28.28	26.53	-6.6	26.89
A3		12.28	12.23	-0.4	12.26
B		12.30	11.76	-4.6	11.94
C		11.80	13.07	9.7	13.13
AA	Al_2O_3 (%)	6.32	6.46	2.2	6.36
A1		8.66	8.40	-3.1	8.20
A2		8.12	7.66	-6.1	7.68
A3		9.09	8.18	-11.1	8.24
B		9.53	9.36	-1.8	9.13
C		9.46	8.95	-5.6	8.99
AA	SiO_2 (%)	66.89	66.25	-1.0	67.58
A1		57.64	58.17	0.9	58.69
A2		30.72	34.01	9.7	33.79
A3		55.36	56.36	1.8	56.67
B		54.69	56.56	3.3	56.97
C		51.16	48.77	-4.9	48.77
AA	Fe_2O_3 (%)	5.76	5.75	-0.1	5.55
A1		9.49	9.12	-4.1	9.25
A2		16.18	15.29	-5.9	15.18
A3		11.04	10.98	-0.5	10.63
B		10.53	10.29	-2.4	10.02
C		13.66	14.24	4.1	14.02
AA	LOI (%)	4.93	5.00	1.6	4.87
A1		5.92	5.91	-0.1	5.89
A2		9.36	9.28	-0.9	9.35
A3		6.08	6.37	4.5	6.32
B		6.32	5.98	-5.6	5.96
C		6.97	7.43	6.2	7.43

Table 14.9 Weighted mean composite grades vs. the block model grades for Indicated areas

Domain	Variable	Composite mean grade	Block mean grade (OK)	Relative difference (%)
AA	MnO (%)	-	-	-
A1		12.45	11.95	-4
A2		28.73	27.87	-3
A3		12.66	12.68	0
B		12.80	12.25	-5
C		10.25	10.75	5
AA	Al_2O_3 (%)	-	-	-
A1		8.85	8.69	-2
A2		8.32	8.09	-3
A3		9.59	9.03	-6
B		10.28	10.52	2
C		9.67	9.81	1
AA	SiO_2 (%)	-	-	-
A1		57.12	58.16	2
A2		29.26	30.96	5
A3		52.90	52.45	-1
B		52.15	52.27	0.2
C		52.38	50.05	-5
AA	Fe_2O_3 (%)	-	-	-
A1		9.64	9.14	-5
A2		16.69	16.21	-3
A3		11.68	12.31	5
B		11.50	11.64	1
C		13.07	13.83	6
AA	LOI (%)	-	-	-
A1		5.97	5.91	-1
A2		9.48	9.35	-1
A3		6.33	6.61	4
B		6.69	6.71	0.2
C		6.69	6.71	0.2

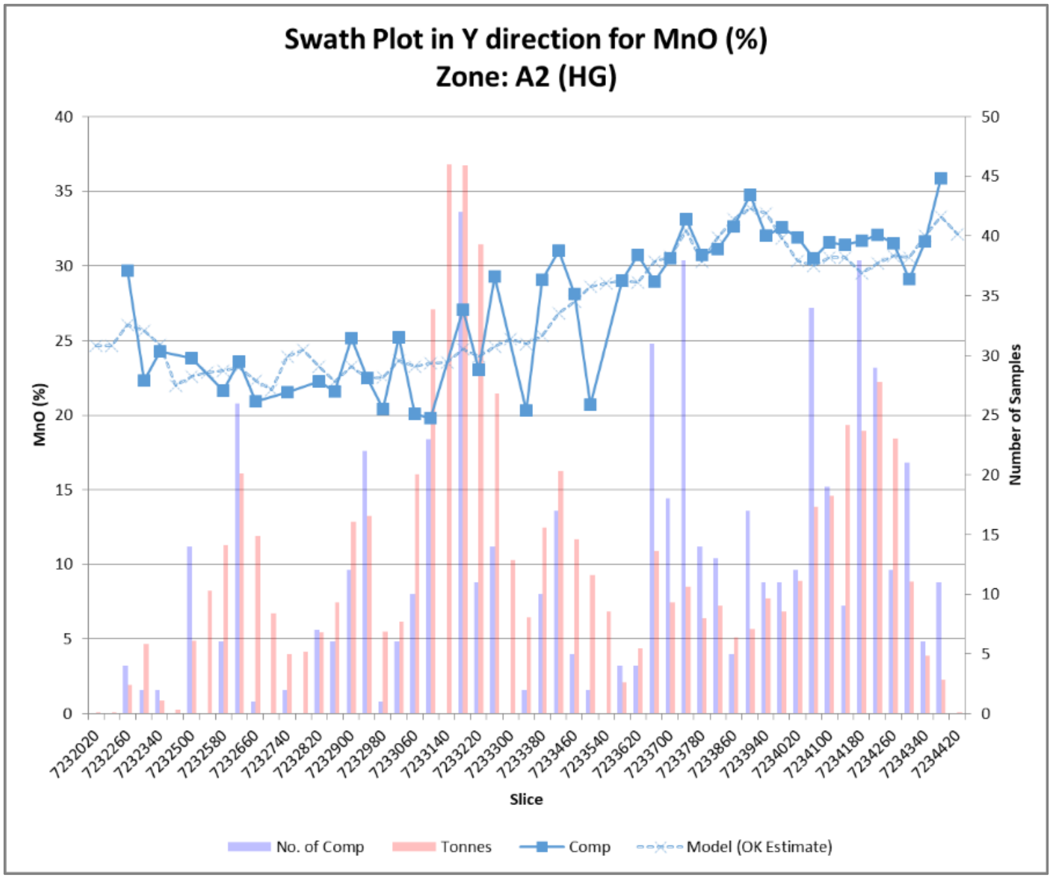
Table 14.10 Weighted mean composite grades vs. the block model grades for Inferred areas

Domain	Variable	Composite mean grade	Block mean grade	Relative difference (%)
AA	MnO (%)	10.83	10.32	-5
A1		11.01	13.25	17
A2		25.32	25.26	0
A3		10.26	11.60	12
B		11.33	11.13	-2
C		12.67	12.14	-4
AA	Al_2O_3 (%)	6.32	6.46	2
A1		8.85	8.69	-2
A2		8.32	8.09	-3
A3		9.59	9.03	-6
B		10.28	10.52	2
C		9.31	11.08	16
AA	SiO_2 (%)	66.89	66.25	-1
A1		63.62	56.43	-13
A2		43.98	36.09	-22
A3		68.30	61.18	-12
B		60.81	59.96	-1
C		50.49	48.13	-5
AA	Fe_2O_3 (%)	5.76	5.75	-0.1
A1		8.65	10.02	14
A2		10.97	14.83	26
A3		7.45	9.24	19
B		8.00	9.33	14
C		14.12	13.61	-4
AA	LOI (%)	4.93	5.00	1.6
A1		5.28	6.15	14
A2		8.62	9.55	10
A3		4.73	5.76	18
B		5.41	5.26	-2
C		6.93	7.31	5

14.9.2 Swath plots

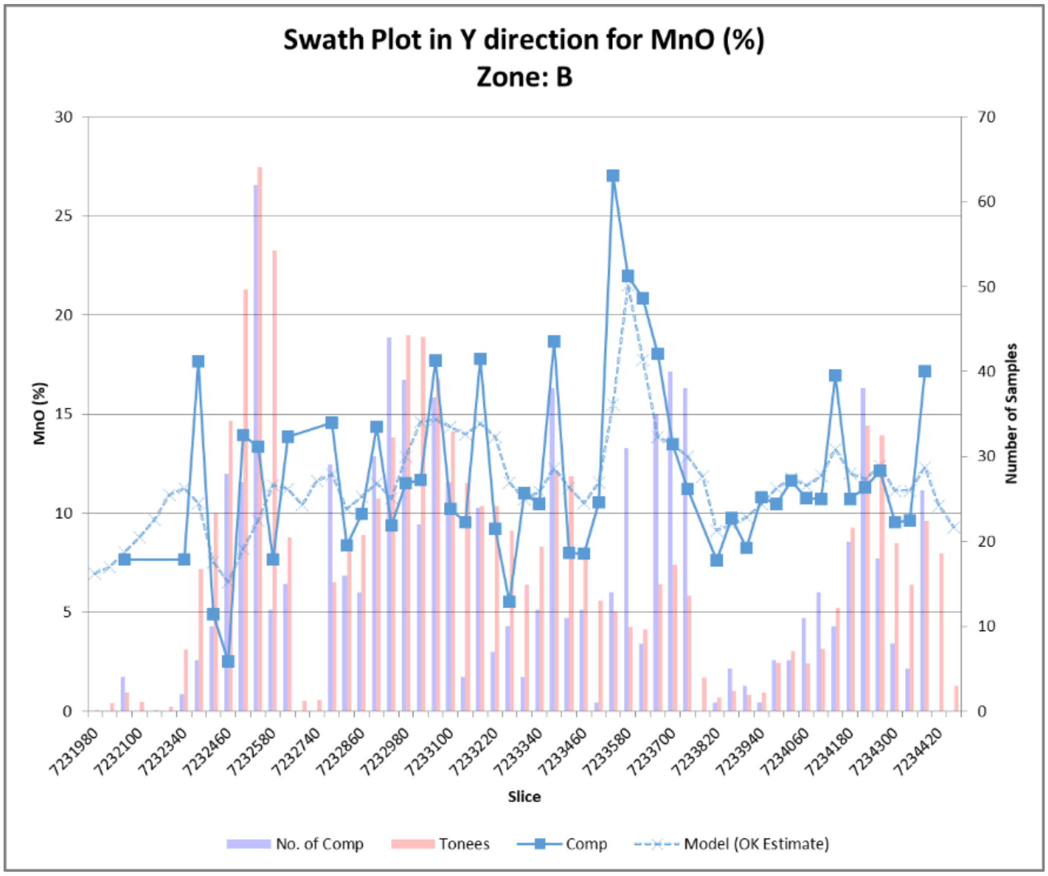
Swath plots were compiled to validate the estimates on a semi-local scale. This entailed comparing the mean of the input composites to the mean of the output OK estimates (model) in 40 m wide east-west corridors. This was completed separately for each estimation domain (Figure 14.10) and (Figure 14.11).

Figure 14.10 Swath plot for manganese oxide (%) for domain A2 in Y direction



Source: CSA Global

Figure 14.11 Swath plot for manganese oxide (%) for domain B in Y direction



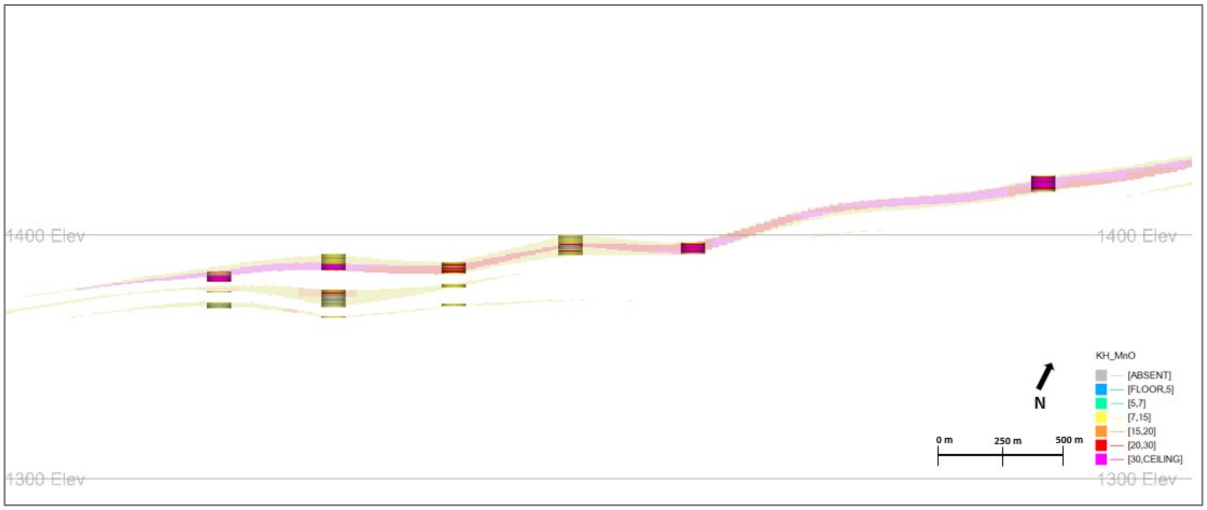
Source: CSA Global

The swath analysis shows that the semi-local estimation is acceptable. A degree of smoothing is evident, especially in areas where fewer data are available. The general trend of the composite grades is reflected in the block model.

14.9.3 Localised visual validation

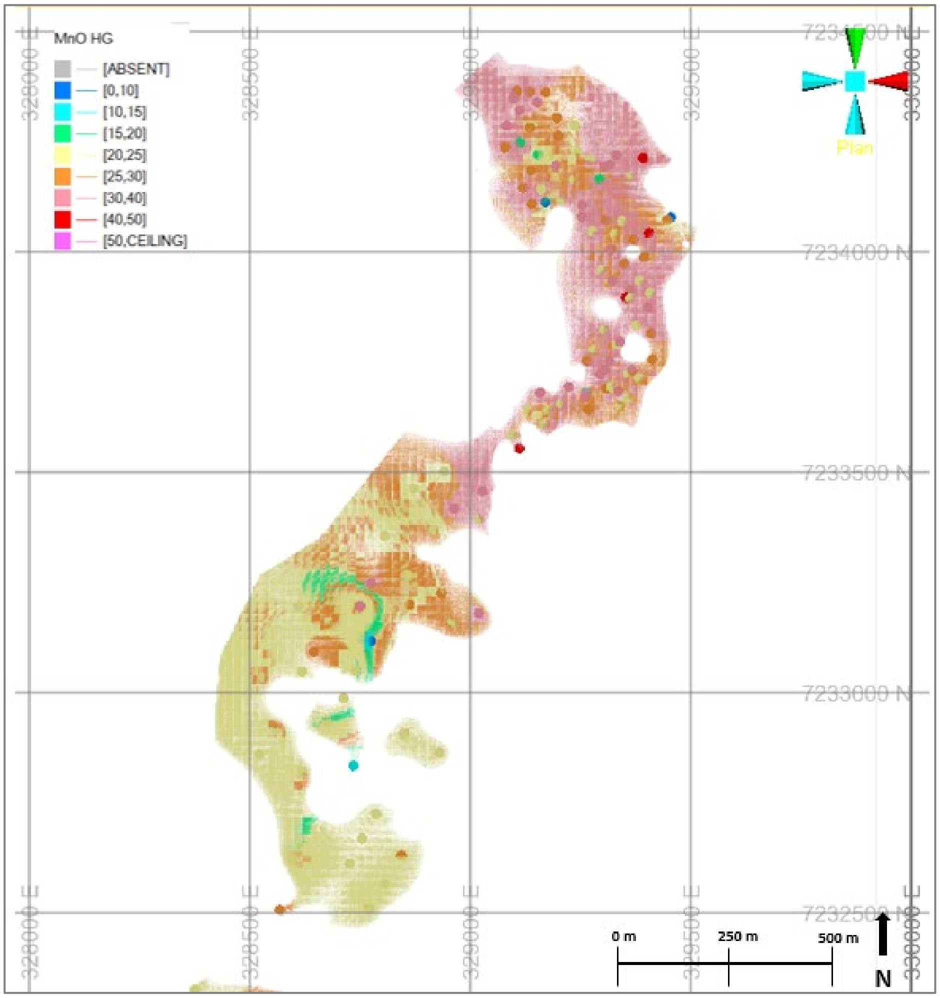
Cross sections were visually reviewed section by section and in 3D to compare the assay data against the estimated block model. This process validated the model on a local scale when comparing the estimated blocks in the vicinity of the input composites. The process showed a reasonable correlation between composites and OK estimates (Figure 14.12 and Figure 14.13).

Figure 14.12 Oblique section for visual validation, manganese oxide (%)



Source: CSA Global

Figure 14.13 Plan view of domain A2 for visual validation



Source: CSA Global

14.9.4 Bulk density assignment

In 2022, SRK reviewed the available density determinations (SRK 2022). The following sets of data were available:

- SET 1: a dataset of 732 density determinations taken from the Phase 1 DD programme and determined using the Archimedes method, of which 207 were taken within mineralisation. Core was sampled wet and not dried prior to measurement.
- SET 2: a dataset of 49 density determinations taken from the Phase 2 DD programme and determined using the Archimedes method, of which 25 were taken within mineralisation. Samples were dried and wax coated prior to measurement.
- SET 3: a dataset of 30 core samples taken from outcrop using a handheld coring machine. Sample densities were determined using the volume of the core barrel and the weight of the sample after drying rather than by the Archimedes method. All samples were taken in mineralisation.
- SET 4: a dataset of three trial pits in mineralisation where the pits were filled with water to measure volume and the excavated material was dried and weighed to determine density. All samples were taken in mineralisation.
- SET 5: a data set of 84 samples taken from diamond core from the K.Hill Project southern extension area drilling.

In deciding on an approach to assign density to individual blocks and domains, SRK investigated the correlation between manganese oxide grade and density with a regression. The study of the relationship between manganese oxide grade and density yielded poor results, with no clear relationship between manganese oxide grade and density identified.

SRK considered that there were insufficient samples within the SET 2 dataset to justify direct estimation of the data into the block model for the northern part of the deposit. Therefore, it was decided to apply average density values to the block model on a domain-by-domain basis. No samples were available for the thin overburden cover (OVB). As such, a standard density for soil and loose rock of 1.5 g/cm^3 was applied to this domain. Similarly, no density samples were taken within the minor Horizon C. SRK assigned an average density value from Horizon B to this domain, as both domains have a broadly similar grade profile. There were also no samples available in the SET 2 database for the FEL waste rock domain. Here, the SET 1 data were used, with the application of a 10% decrease in density to account for moisture content.

CSA Global reviewed the SRK density study and found that it is still applicable for the northern area in the block model. In addition, the 2022 drilling campaign included more Archimedes method (similar to dataset 2) density determinations for the southern area. Seventy-six density determinations were available in the southern area. A mean density was determined per domain for the southern area of the block model. The areas do not have a sufficient number of density determinations to allow for the spatial estimation of density.

No bulk densities were determined. Bulk sampling of the deposit is required to determine how the deposit will behave at a mining scale. CSA Global recommends continued density determinations of DD core. The effect of densities on contained metal is significant. The extent of the issue and the ability to report representative density values requires resolution prior to considering upgrading the confidence classification as part of future MRE updates. Table 14.11 and Table 14.12 show the densities used in the block model per domain.

The AA domain does not extend to the northern area, and in the southern area, as no density determinations were conducted, the adjacent domain values were used. This is deemed appropriate as the AA domain is relatively small and does not contribute significantly to the Mineral Resources. Similarly, no density determinations were available for the C domain in the northern area and the adjacent B domain value was used.

Table 14.11 Mean density values assigned by domain

Domain	Mean density northern area (t/m^3)	Number of density determinations	Mean density southern area (t/m^3)	Number of density determinations
AA	N/A	N/A	1.69	N/A
A1	2.18	2	1.69	16
A2	2.14	14	1.72	21
A3	2.28	5	1.93	10
B	2.32	0	1.83	16
C	2.32	0	2.15	13

Table 14.12 Mean density of waste rock

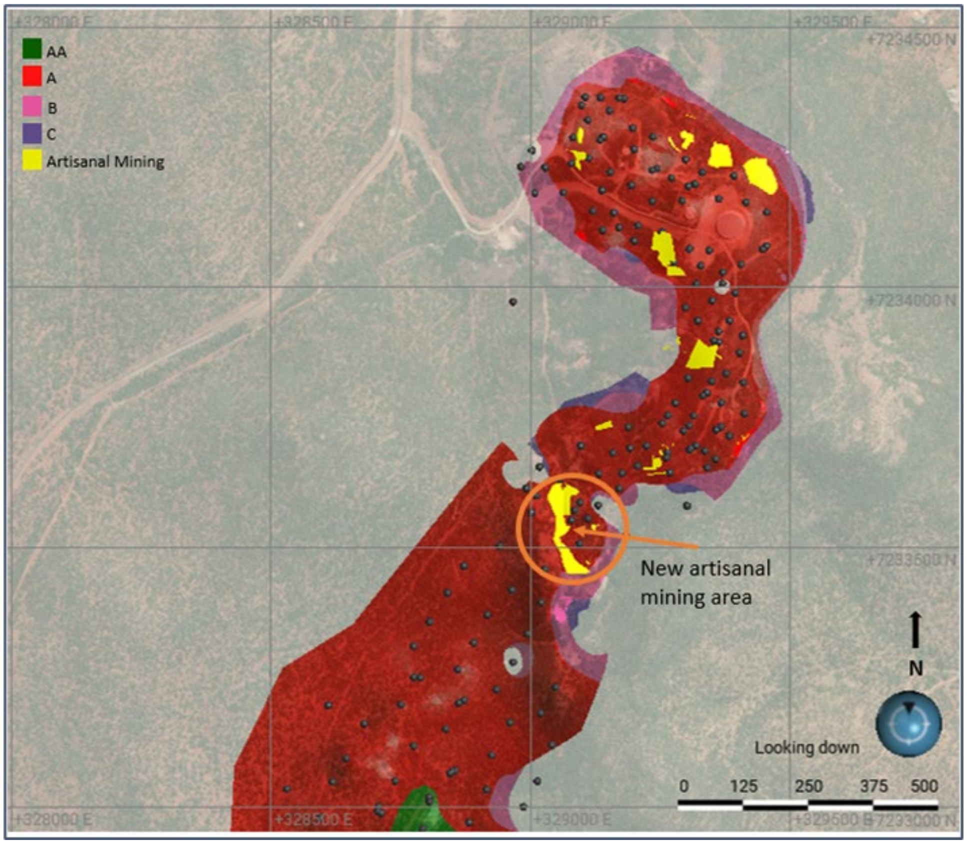
Waste rock	Mean density (t/m^3)	Number of density determinations
OVB	1.50	N/A
CDB	2.48	7
FSH1	2.01	17
FSH2	2.01	17
FEL	2.38	22

14.10 Artisanal Mining

Artisanal mining activities have been documented at the K.Hill Project for a significant period of time. Giyani engaged Terravision Exploration (Terravision) in December 2020 to undertake a survey of the workings using ground penetrating radar and laser rangefinders. A total of 46 survey lines were made, covering a total of 5 km. Although these surveys were not exhaustive as they were limited to known areas of artisanal workings, the identified areas were surveyed systematically and it is considered likely that the majority of major workings will have been intercepted by these survey lines. Terravision interpreted the survey data and provided a 3D dataset outlining areas of artisanal workings, which has subsequently been applied as depletion prior to reporting the Mineral Resource. CSA Global reviewed this interpretation, and it appears to be reasonable. It is possible that unidentified workings with obscured accesses exist and have not yet been captured. The workings are understood to be largely within the highest-grade zones. No artisanal mining has been recorded in the southern extension area.

Since the last MRE update, Giyani identified an additional area of mining. This area was mapped and included in the artisanal workings. It is likely that the artisanal mining did not include all domains in the K.Hill model, but due to uncertainty of the depth of these workings, all domains were depleted (Figure 14.14).

Figure 14.14 Artisanal mining areas



Source: CSA Global

14.11 Mineral Resource classification

14.11.1 Mineral Resource classification parameters

The K.Hill Project Mineral Resource has been classified as Indicated and Inferred Mineral Resources. The classification category is based on an assessment of geological understanding of the deposit, geological and mineralisation continuity, drillhole spacing, quality control results, and available density information.

There is confidence in the accuracy and precision of the assay data, which is considered appropriately robust for the estimation and reporting of a Mineral Resource. While there is grade variability in the manganese oxide, the spatial variability is well understood based on the omni-directional variograms. Geological confidence in the extent of the mineralisation domains is reasonable, but there is recognition that the local-scale variability in terms of thickness and grade requires further investigation by infill drilling. The drilling recoveries in the Phase 1 DD programme are poor due to the friable nature of the near surface material; however, significant improvements in drilling practices have been made of late. This later improvement in drilling technique resulted in density determinations by means of dry wax immersion. The low number of density determinations is a risk;

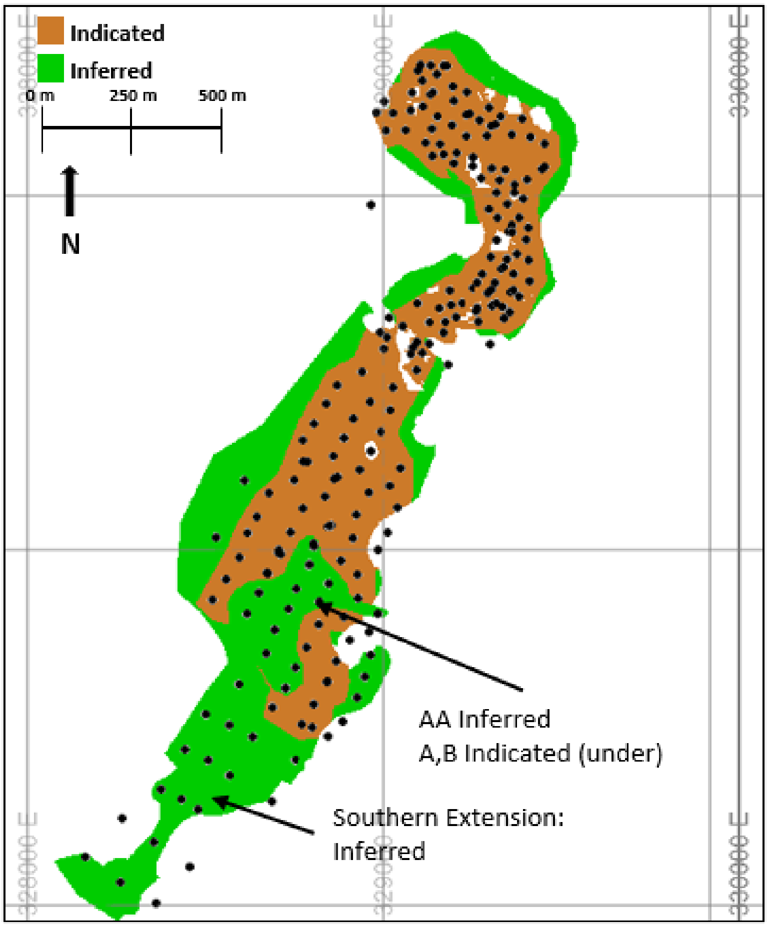
therefore the continuation quality drilling and density determinations is encouraged. The topographic survey used to limit the vertical extent of the geological interpretation and block model is considered accurate. The extent of artisanal underground workings is reasonably well understood, although a conservative approach was taken in terms of depths of these workings.

Infill drilling and step-out drilling with density determinations will be required to increase the Mineral Resource confidence. A more detailed fault investigation should also be conducted to understand the impact of faults on the mineralisation.

Inferred Mineral Resources were classified from drilling data spaced approximately 125 m apart and Indicated Mineral Resources were classified from drilling data spaced 100 m apart or extrapolated 50 m beyond the last available data.

Given the above considerations, CSA Global classified the A1, A2, A3 and B domains as Indicated in areas where the blocks are within the 50 m from available data. A smaller part of domain C has been classified as Indicated, only where the geology shows continuity. Domain AA was classified as Inferred. The southern extension area has been classified as Inferred (Figure 14.15).

Figure 14.15 Plan view of Mineral Resource classification



Source: CSA Global

14.11.2 Reasonable prospects for economic extraction

CIM *Definition Standards for Mineral Resources & Mineral Reserves* (CIM 2014) require that resources have RPEEE. This generally implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off grade, considering possible extraction scenarios and processing recoveries.

To satisfy the requirement of reasonable prospects for economic extraction by open pit mining, reporting pit shells were determined based on conceptual parameters and costs supplied by Giyani and reviewed for reasonableness by the QP (Table 14.13).

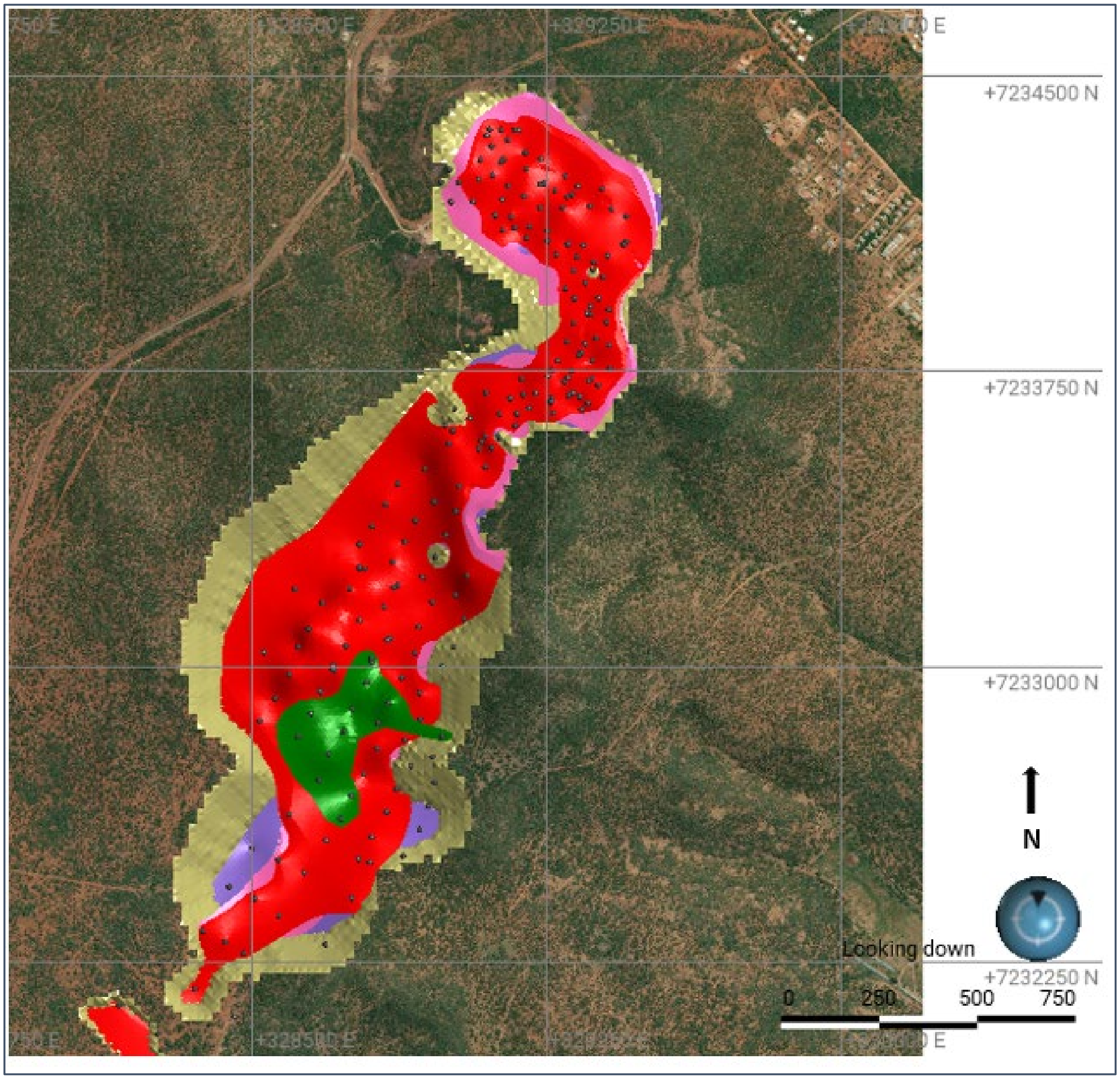
The RPEEE parameters are guided by the latest available data from the previously completed Technical Report (SRK and Tetra Tech 2022). Figure 14.16 shows the RPEEE conceptual pit. The pit final depth is approximately 240 m below surface.

Table 14.13 Conceptual mining and cost parameters used to determine the reasonable prospects for eventual economic extraction pit shell

Parameter	Units	Value
Production parameters		
Production Rate ROM	t/a	400,000
Slope Angles	degrees	41
Mining parameters		
Dilution	%	0
Recovery	%	100
Processing recovery		
Total process recovery MnO	%	88.5
Operating Cost		
Mining cost	US$/t rock	1.95
Processing	US$/t ROM	523
Royalty Mn	%	3
Selling cost*	US$/t MnO	0
G&A	US$/a	3,216,000
	US$/t ROM	8.04
Metal Price		
MnO	US$/t	9,054
HPMSM	US$/t	3,800

Note:
*Selling costs are included in the metal price

Figure 14.16 Plan view of reasonable prospects for eventual economic extraction pit shell with mineralised domains



Source: CSA Global

14.12 Mineral Resource statement

14.12.1 Results

The Mineral Resource is reported above a cut-off grade of 7.3% manganese oxide and comprises 8.6 Mt of Indicated material at a grade of 15.2% manganese oxide and 6.1 Mt of Inferred Material at a grade of 14.1% manganese oxide. Mineralisation below the reporting pit shell is not considered as Mineral Resource, as it does not have reasonable prospects for economic extraction at the time of reporting (Table 14.14).

Table 14.14 K.Hill Project Mineral Resource at a cut-off grade of 7.3% manganese oxide as of 6th June 2023

Mineral Resource Classification	Tonnage (Mt)	MnO (%)	Al_2O_3 (%)	SiO_2 (%)	Fe_2O_3 (%)	LOI (%)
Indicated	8.6	15.2	9.1	49.5	12.2	7.2
Inferred	6.1	14.1	8.0	53.5	11.4	6.7

Notes:
a) The Mineral Resource has been classified and reported under the guidelines defined by the Canadian Institute of Mining, Metallurgy and Petroleum in their document *CIM Definition Standards for Mineral Resources and Mineral Reserves* (CIM 2014).
b) Mineral Resources are not Mineral Reserves and have not demonstrated economic viability.
c) Mineral Resources are stated as in situ dry tonnes; figures are reported in metric tonnes.
d) Figures have been rounded to the appropriate level of precision for the reporting of Mineral Resources.
e) Estimation has been completed within six different mineralization domains.
f) Mineral Resources are reported assuming open pit mining methods.
g) The Mineral Resource is reported within a conceptual pit shell determined using a price of US$3,800/t HPMSM (equivalent to US$9,054/t manganese oxide), conceptual parameters, and costs to support assumptions relating to reasonable prospects for eventual economic extraction.
h) The Mineral Resource is reported at a cut-off grade of 7.3% manganese oxide.
i) The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. CSA Global is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other any other relevant factors affecting the MRE.
j) HPMSM price quoted is based on 2022 market data, which was available at the time of reporting the Mineral Resource. Additional pricing information will be available for input into subsequent technical studies, and this may impact on the Mineral Resource reported.

The estimated block model was tabulated at various cut-off grades (Table 14.15). This tabulation does not represent a Mineral Resource and only serves to illustrate the sensitivity to various cut-offs.

Table 14.15 Grade-tonnage scenarios at various cut-offs within the reasonable prospects for eventual economic extraction pit shell

Cut-off MnO (%)	Tonnage (Mt)	MnO (%)	Al_2O_3 (%)	SiO_2 (%)	Fe_2O_3 (%)	LOI (%)
7.3	14.7	14.7	8.7	51.1	11.9	7.0
8	14.2	15.0	8.6	50.7	12.0	7.0
9	13.1	15.6	8.5	49.9	12.3	7.1
10	11.6	16.3	8.4	49.0	12.5	7.3
11	10.1	17.2	8.2	47.9	12.7	7.5
12	8.7	18.1	8.1	46.7	12.9	7.7
13	7.3	19.1	8.0	45.3	13.2	7.9
14	6.1	20.3	7.9	43.7	13.5	8.1
15	5.0	21.5	7.8	41.9	13.9	8.4
16	4.1	22.9	7.7	39.9	14.2	8.6
17	3.5	23.9	7.7	38.5	14.3	8.8
18	3.1	24.8	7.6	37.2	14.5	9.0
19	2.8	25.4	7.6	36.2	14.7	9.1
20	2.6	26.0	7.6	35.3	14.8	9.2

14.12.2 Comparison with previous Mineral Resource estimates

SRK previously reported a Mineral Resource estimate for the K.Hill Project in February 2022. Table 14.16 shows a comparison of the February 2022 and current Mineral Resource.

Table 14.16 Comparison of the SRK February 2022 Mineral Resource and the current July 2023 Mineral Resource at a 7.3% manganese oxide cut-off

	February 2022		July 2023	
	Tonnage (Mt)	MnO (%)	Tonnage (Mt)	MnO (%)
Indicated	2.1	19.3	8.6	15.2
Inferred	3.1	16.9	6.1	14.1

The current Mineral Resource represents an updated interpretation and estimate. The updated model includes additional RC and DD data drilled in the southern area, as well as updated density determinations. The following was investigated to show the changes:

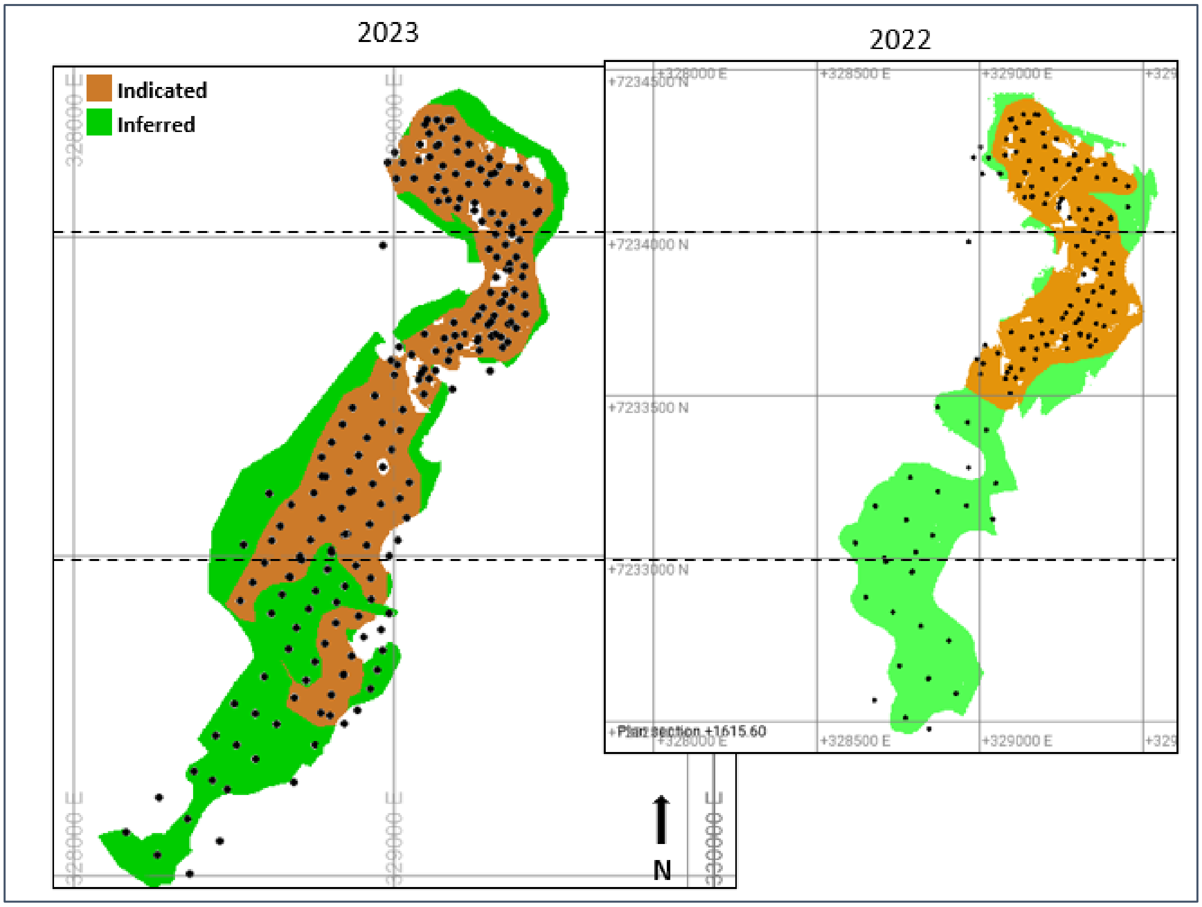
- Infill and step-out drilling has resulted in remodelling of the mineralisation wireframes.
- An updated block model and set of estimation parameters were compiled based on new drillhole data. The new data were mainly from the southern area and include infill and step-out drilling.
- The addition of drilling data has resulted in more robust omni-directional variograms for all domains, instead of using directional variograms or inverse distance methods.
- The northern area was re-classified, with only minor changes and the southern area was upgraded from Inferred to Indicated due to additional drilling in the area.
- Figure 14.17 shows the classification comparison between 2022 and 2023.
- Updated density data for the southern area resulted in higher tonnages in the area.
- RPEEE conceptual pit parameters were updated. The input parameters are compared in Table 14.17

Table 14.17 Comparison between 2022 and 2023 input parameters for the reasonable prospects for eventual economic extraction conceptual pit

Parameter	Units	Value 2022	Value 2023
Production parameters			
Production Rate ROM	t/a	200,000	400,000
Slope angles	degrees	41	41
Mining parameters			
Dilution	%	3	0
Recovery	%	98	100
Processing recovery			
Total process recovery MnO	%	90.7	88.5
Operating Cost			
Mining cost	US$/t rock	3.46	1.95
Processing	US$/t ROM	213	523
Royalty Mn	%	3	3
Selling cost	US$/t MnO	114	0

Parameter	Units	Value 2022	Value 2023
G&A	US$/a	3,500,000	3,216,000
	US$/t ROM	20.0	8.04
Metal Price			
MnO	US$/t	9,054	9,054
HPMSM	US$/t	3,800	3,800
Other			
Discount rate	%	10	0
Marginal Cut-off	% MnO	7.3	7.3

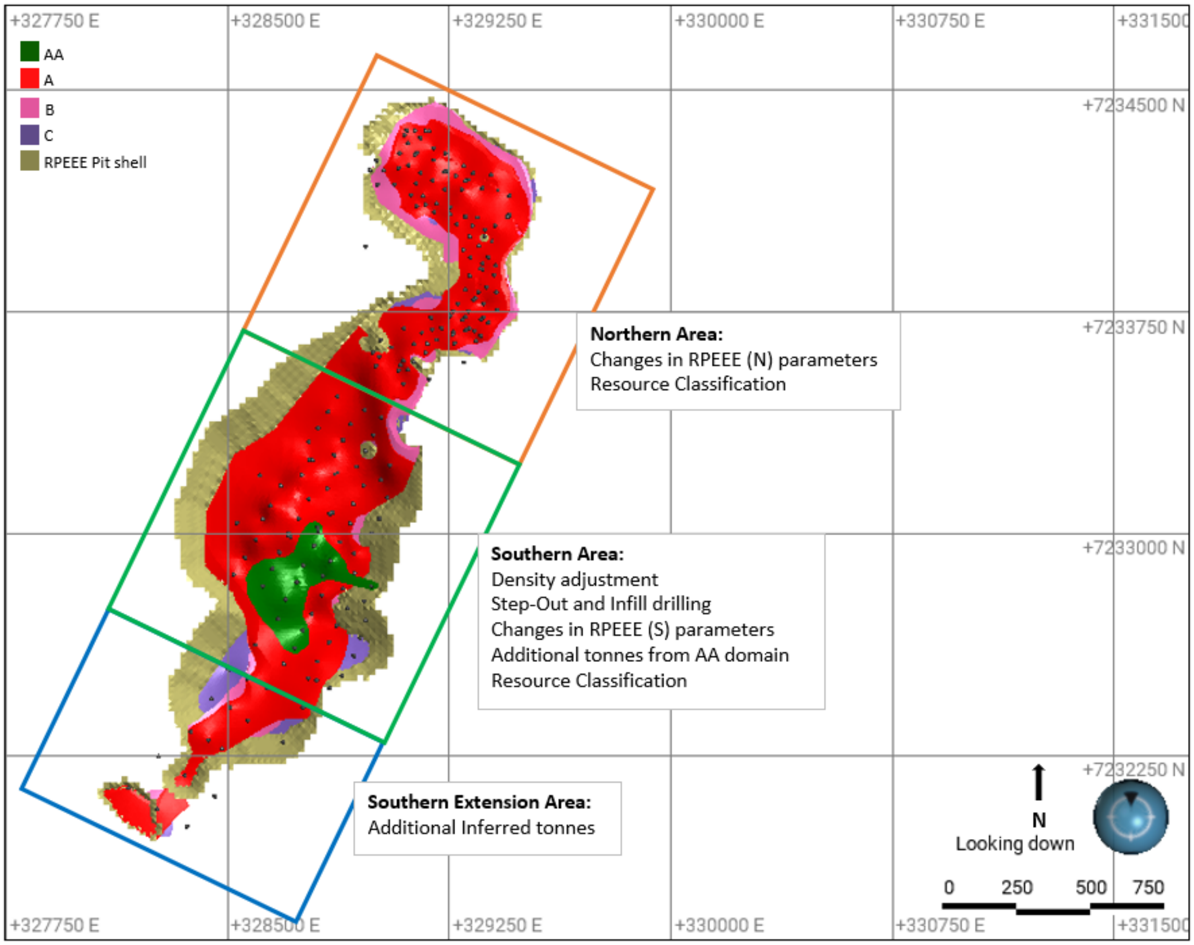
Figure 14.17 Classification comparison with drill collars, 2022 and 2023



Source: CSA Global

The model was divided into three areas for comparison of the previous and current MRE (Figure 14.18). The changes are categorised as discussed above, namely step-out and infill drilling, updated density data, southern extension area, RPEEE (north and south) and the AA domain.

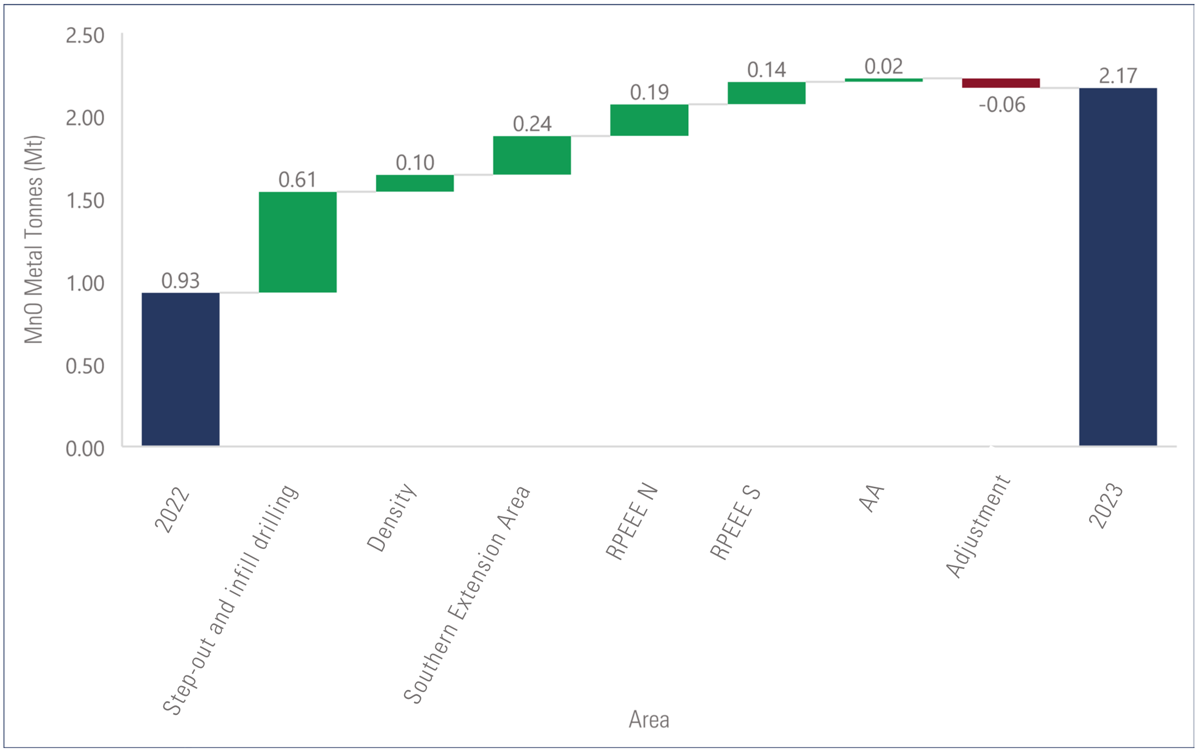
Figure 14.18 Areas investigated for changes from the 2022 to the current Mineral Resource



Source: CSA Global

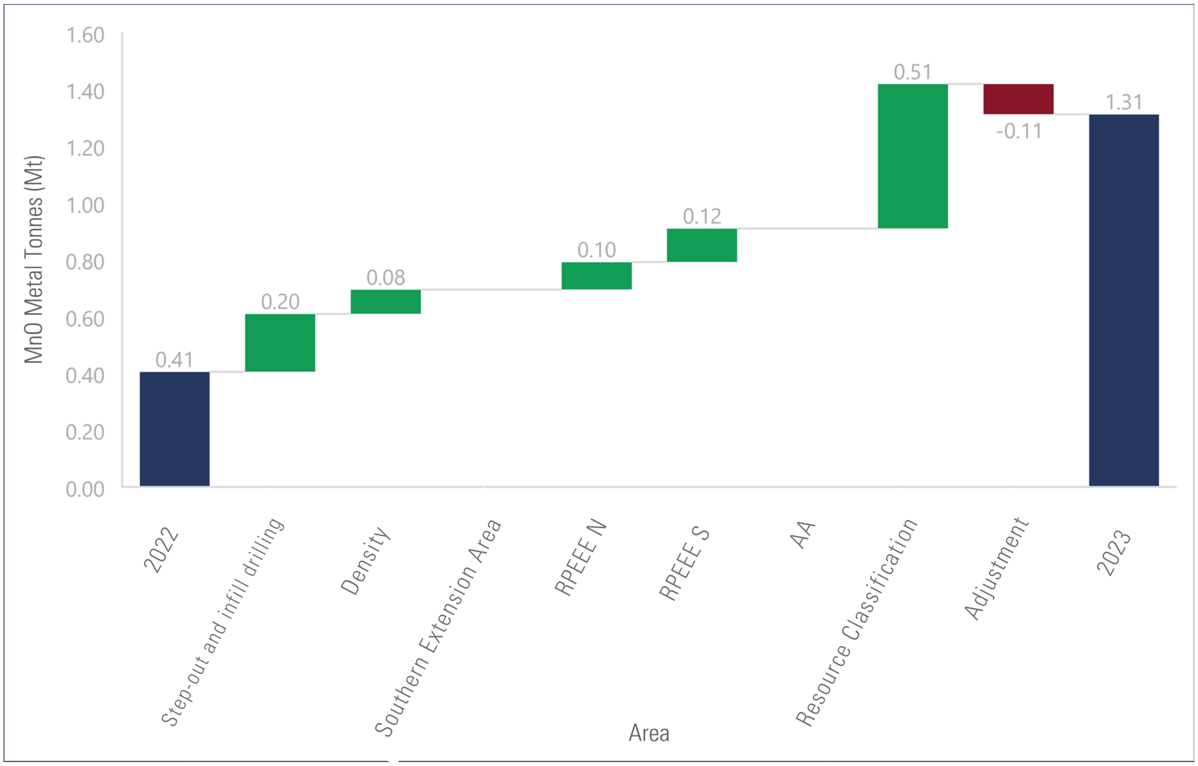
The changes are illustrated in a waterfall graph for total metal content (Figure 14.19). Similarly, Figure 14.20 and Figure 14.21 show the changes to Indicated and Inferred Mineral Resources using the same categories. The adjustment can also be ascribed to changes in the estimation parameters and volume changes due to interpretational changes.

Figure 14.19 Waterfall graph showing the comparison between 2022 MRE and 2023 MRE



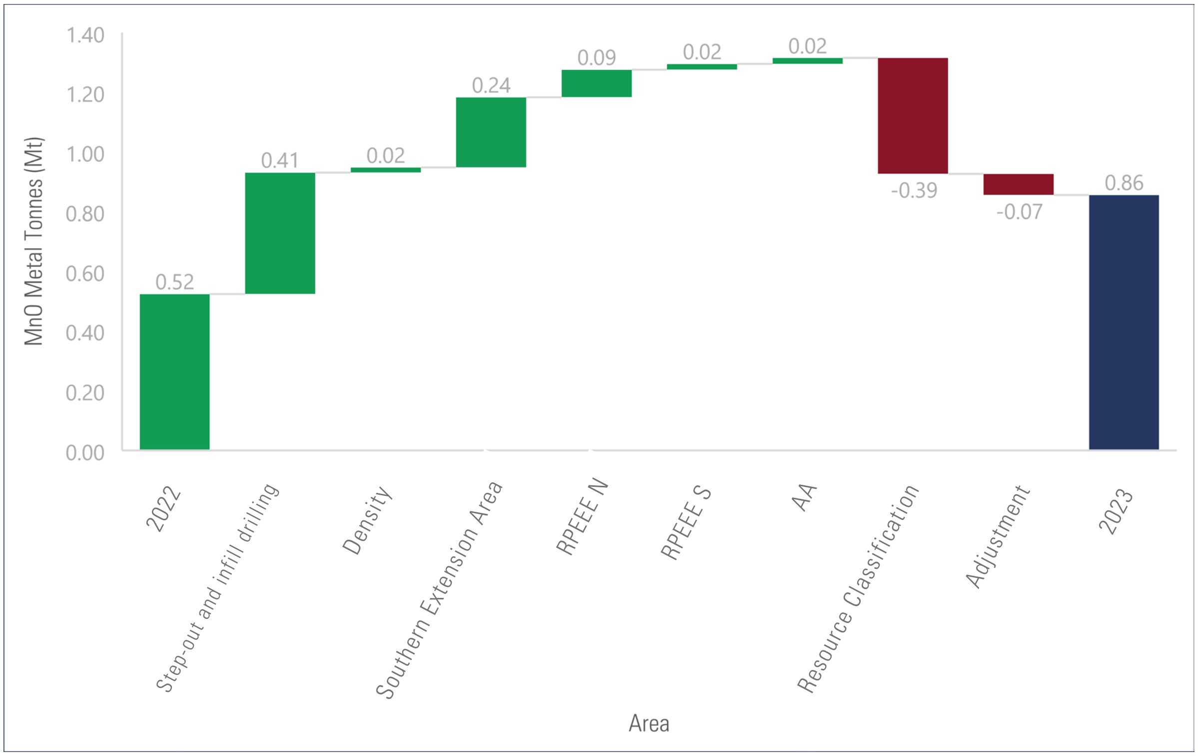
Source: CSA Global

Figure 14.20 Waterfall graph showing the comparison between 2022 MRE and 2023 MRE for Indicated Resources



Source: CSA Global

Figure 14.21 Waterfall graph showing the comparison between 2022 MRE and 2023 MRE for Inferred Resources



Source: CSA Global

The updated MRE represents a 65% increase in total reported tonnages at a lower grade compared to the 2022 MRE.

14.13 Risk

The following risks have been evaluated for the Mineral Resource:

- Average densities for each domain in the block model were taken from a small database of densities calculated from wax-coated core samples. Although of higher quality, the quantity of data present a risk to the K.Hill Project. The densities may not be entirely representative of the mineralised units. It is recommended that the wax-coated core densities continue during the next phase of exploration and are re-evaluated during the next MRE update.
- Channel structures interpreted in the base of the overlying CDB unit that erodes the upper Mn-shale horizon have been observed. These structures have been modelled where they are intersected, and any similar structures that may exist are expected to be smaller than the approximately 50 m drill spacing at K.Hill. These structures may influence the interpretation and estimation in this area.
- Inferred Mineral Resources account for approximately 40% of the tonnage above the reporting cut-off. Inferred Mineral Resources have a lower level of confidence than that applied to Indicated Mineral Resources and are based on limited information and sampling.

15 MINERAL RESERVE ESTIMATES

There are no Mineral Reserve estimates for this Technical Report.

16 MINING METHODS

The relatively low processing throughput of 200 kt/a of material, which typically requires a total volume of material moved to be <1.5 Mm^3/a, makes the K.Hill deposit suitable to use a traditional open pit truck-and-shovel mining method. These mining activities are easily manageable with a relatively small fleet of hydraulic excavators and articulated dump trucks.

The ROM material destined for processing will be hauled either to the ROM pad or to stockpiles. Material that is between the economic cut-off grade and the operational cut-off grade will be sent to the medium- and low-grade stockpiles in order to achieve a high grading policy in the early years of the Project. The ROM pad will consist of several "fingers" that will be used to separate material by grade and type. A front-end loader will selectively reclaim the material to achieve the required blend, which will then be fed to the primary crusher.

Material that is below the economic cut-off grade will be sent to the WRD located east of the open pit. The area immediately west of the open pit remains available for future step-out drilling, and the area further west is reserved for mine and processing plant infrastructure.

16.1 Mine design criteria

Datamine's NPV Scheduler software was used to run the open pit optimisation, which is based on the industry standard Lerch-Grossman algorithm, to determine the optimal pit limit, mining sequence, and optimised LOM schedule. The mine optimisation is based on the inputs shown in Table 16.1. Note that the processing cost for the K.Hill Project is very high compared to other costs such as mining.

Table 16.1 Optimisation input parameters

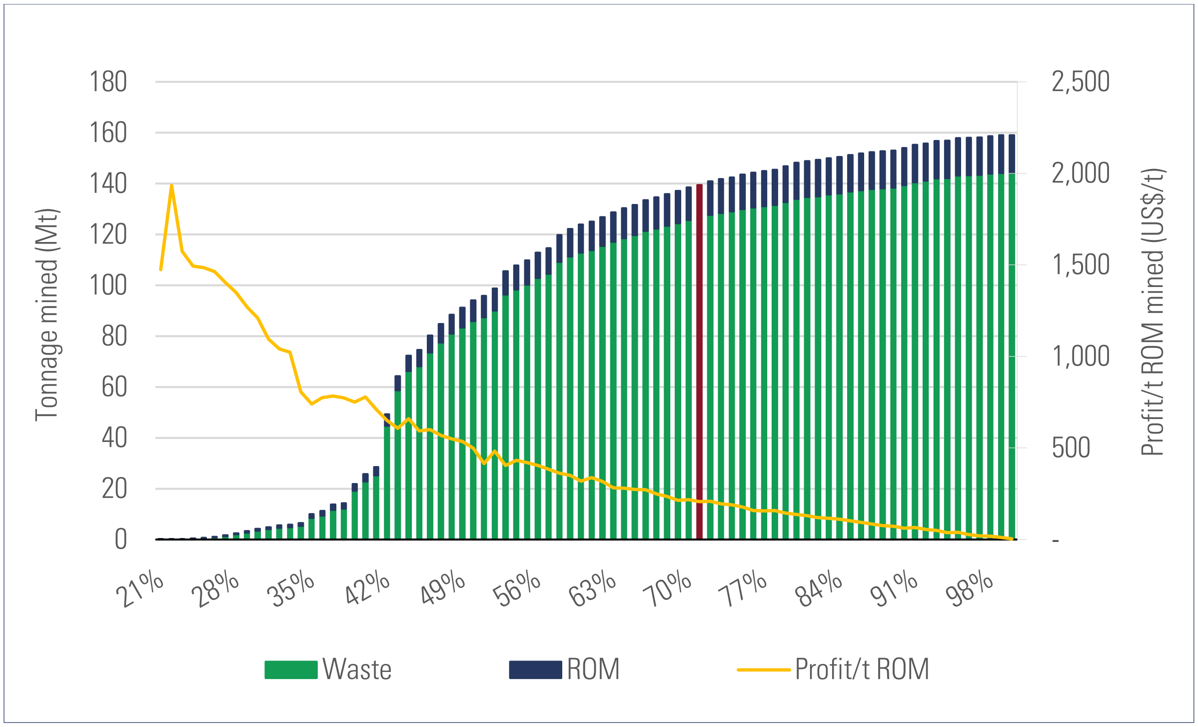
Parameter	Units	Value	Comments
Metal price	US$/t HPMSM	3,800	Based on conversion factor
Mining cost – waste	US$/t mined	2.50	Factored from previous studies
Mining cost adjustment factors	factor	1.2	Allow for additional mineralised material costs
Mining dilution	%	5.0	Factored from previous studies
Mining recovery	%	95.0	Factored from previous studies
Process recovery	%	88.5	See Section 17
Process cost	US$/t ROM	531	Includes G&A
Overall slope angle	degrees	41.0	See Section 16.4
Royalty	%	3.0	-
Discount rate	%	10.0	-
ROM production target	kt/a	200	-

16.2 Pit optimisation

Using the input parameters listed in Table 16.1, the pit optimisation was run over a range of RFs to generate a set of nested pit shells that were used to select the optimal pit limit and to guide the design of several internal pit expansion stages (pushbacks) that represent the mining sequence.

With the RF 1.0 pit corresponding to a metal price of US$3,800/t HPMSM and a cut-off of 7.2% manganese oxide, the cumulative waste and ROM tonnes mined were plotted against the RF to show the sensitivity to price (Figure 16.1). Additionally, the incremental profit per tonne (revenue – [mining cost + process cost]) is shown as a guide to the selection of the open pit limit for mine design.

Figure 16.1 Optimisation results



Source: Axe Valley Mining

At an RF of 70%, the incremental profit is reduced to approximately US$220/t of ROM material, the total inventory is approximately 12.9 Mt at 14.75% manganese oxide (diluted), and the strip ratio is 9.7:1. This was considered a reasonable point at which to limit the open pit; this corresponds to a metal price of US$2,660/t HPMSM.

The traditional method of open pit selection based on cumulative NPV was not used in this exercise due to the very low mining rate and long LOM. At a discount rate of 10%, there would be a long tail to the graph, and it would be difficult to make an informed decision based on discounted cashflow.

The approach adopted was to use the incremental profit per tonne of ROM material as the basis for open pit selection. By applying a minimum profit margin of US$200/t of ROM material, this equates to a cut-off grade of 10.1% manganese oxide at a price of US$3,800/t of HPMSM.

Increasing to RF1.0 would increase the inventory to 14.8 Mt at 14.0% manganese oxide (2,066 kt manganese oxide recovered) and increase the waste movement by more than 19 Mt (a 15% increase). The net increase in recovered metal would be 152 kt manganese oxide (a 9% increase).

16.3 Mine hydrology considerations

16.3.1 Hydrography

The K.Hill Project site is located at the top of the Kgwakgwe Hill, which acts as a localised hydrological catchment divide, with runoff draining in a both an eastern and western direction into local, non-perennial streams.

Runoff from the eastern side of the Kgwakgwe Hill, which also drains the southern slopes of Kanye, flows approximately 2 km into the Mmamokhasi Dam, which is situated to the southeast of the proposed mining area. The Mmamokhasi Dam has a capacity of approximately 250,000 m^3 and is classified as an agricultural dam. The Mmamokhasi Dam Group, comprising locally appointed representatives, manages the dam on behalf of the local community. The dam water is used for livestock and irrigation for small-scale farming.

Runoff from the western slopes of the Kgwakgwe Hill flows approximately 3 km south into the Nneneke River, which then converges with runoff from the eastern slopes (via the Mamakhosi Dam) into the Matlhapise River, which eventually flows into the Taung River. The Taung River drains in a northerly direction toward the Ngotwane River, which subsequently flows through and feeds the Gaborone reservoir, just south of the city of Gaborone. The Ngotwane River, which originates in South Africa, flows along the border between the two countries towards Zimbabwe where it then flows into the Limpopo River, eventually making its way through Mozambique and into the Indian Ocean.

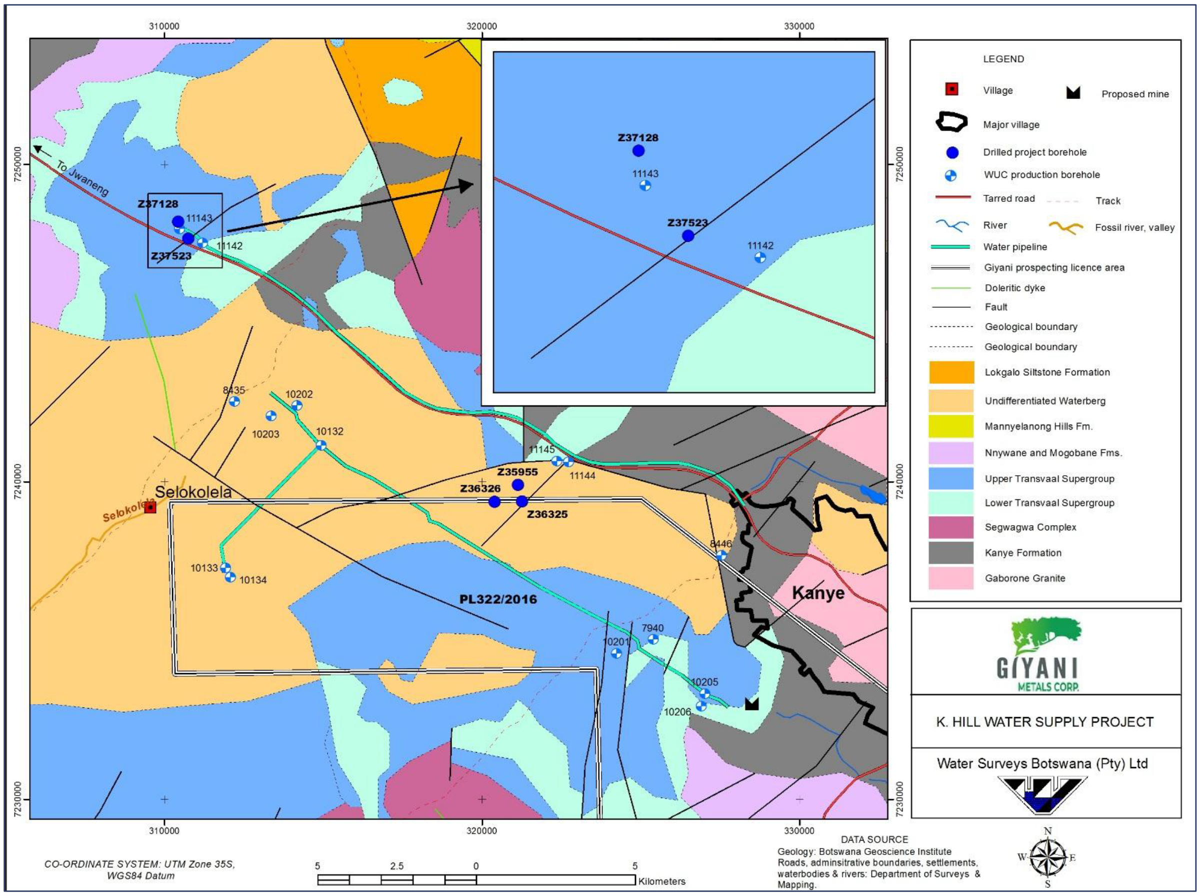
Portions of the open pit, together with the footprint of the WRDs and a significant portion of Kanye, are located in a separate subcatchment to that of the TMF and processing plant. However, any water drainage generated within these mine facilities will converge to the same watercourse approximately 4 km downstream of the site. The topographic elevation in the Project area ranges between 1,268 masl and 1,475 masl.

16.3.2 Hydrogeology

The K.Hill Project area is underlain by an aquifer system that is described as being a low permeability fractured rock system associated with shales. In general, this aquifer is a poor yielding aquifer (low yield data suggesting yields of approximately 0.5 m^3/hour); however, at greater depths where fractures are encountered, higher yields can be expected. Overall, the aquifer underlying the proposed mining area is not regarded as being suitable for groundwater supply purposes. Furthermore, it is understood that the Water Utilities Company (WUC) has informed Giyani that any existing boreholes situated on the K.Hill Project site cannot be used for Project water supply purposes, as they are resting and recovering. The WUC has adopted a strategy of having multiple water sources to reduce the risk of interruptions to water supply to communities. The sources of water available are discussed in detail in Section 18.9.1.

The area to the south and to the west of the proposed mining area comprises mainly dolomitic aquifers in which significantly higher yields and high flow characteristics can be expected. Historical pumping tests have indicated that the dolomitic aquifers to the south and to the west of the Project site are high yielding aquifers, with pump test results indicating sustainable yields of approximately 50 m^3/hour. The WUC currently has a production well, approximately 3 km southwest of the proposed mining area, drilled into the dolomitic aquifer, which supplies water to Kanye. The WUC also owns and operates a water well field in the dolomitic aquifer to the west-northwest area of the Project site, approximately 12 km from the proposed mining area. The regional geology and locations of known supply wells are indicated on Figure 16.2.

Figure 16.2 Regional geology and locations of existing groundwater supply wells



Source: SRK and Tetra Tech (2022)

Groundwater level measurements, collected as part of a number of historical studies, suggest that the depth to groundwater on a regional scale ranges between 30 mbgl to over 150 mbgl. The deeper groundwater levels, measured in the dolomitic aquifers around water supply wells operated by the WUC, are interpreted as being due to groundwater abstraction, which has caused depressions in the groundwater table, and do not necessarily represent steady-state levels. The majority of the available regional groundwater levels were measured in the 1980s and 1990s; therefore, it is recommended that groundwater levels be re-measured. Near the proposed open pit, the groundwater levels (measured in 2020) are approximately 45 mbgl.

The available groundwater level data indicates that the groundwater flow direction is generally in a south-southeast direction, in the vicinity of the proposed open pit. In general, a moderate correlation exists between groundwater level data and topography for shallow boreholes (an average of 54 mbgl), a good correlation exists for medium-depth boreholes (77 mbgl), and there is a poor correlation for deep boreholes (most likely due to groundwater abstraction targeting the aquifers at greater depths and resulting in deeper than expected groundwater levels). However, it should be noted that the groundwater levels used for the contouring are historical, with the majority of the groundwater levels dating back to the 1980s and 1990s. It is recommended that updated groundwater levels be recorded to confirm the current groundwater elevations, gradients, and flow directions.

The groundwater quality in the region of the K.Hill Project is generally regarded as being good, when compared to the BOS drinking water specification (BOS 32:2015). Only slightly elevated manganese and iron concentrations exceeding the drinking water standards were found within the groundwater supply wells that were sampled. All other ions, including trace elements and heavy metals analysed, fell within BOS 32:2015 standards. Therefore, groundwater should not have any impact on processing. Furthermore, total suspended solids (TSS), biochemical oxygen demand (BOD), and chemical oxygen demand (COD) levels were all within acceptable limits.

16.3.3 Mine inflows and dewatering

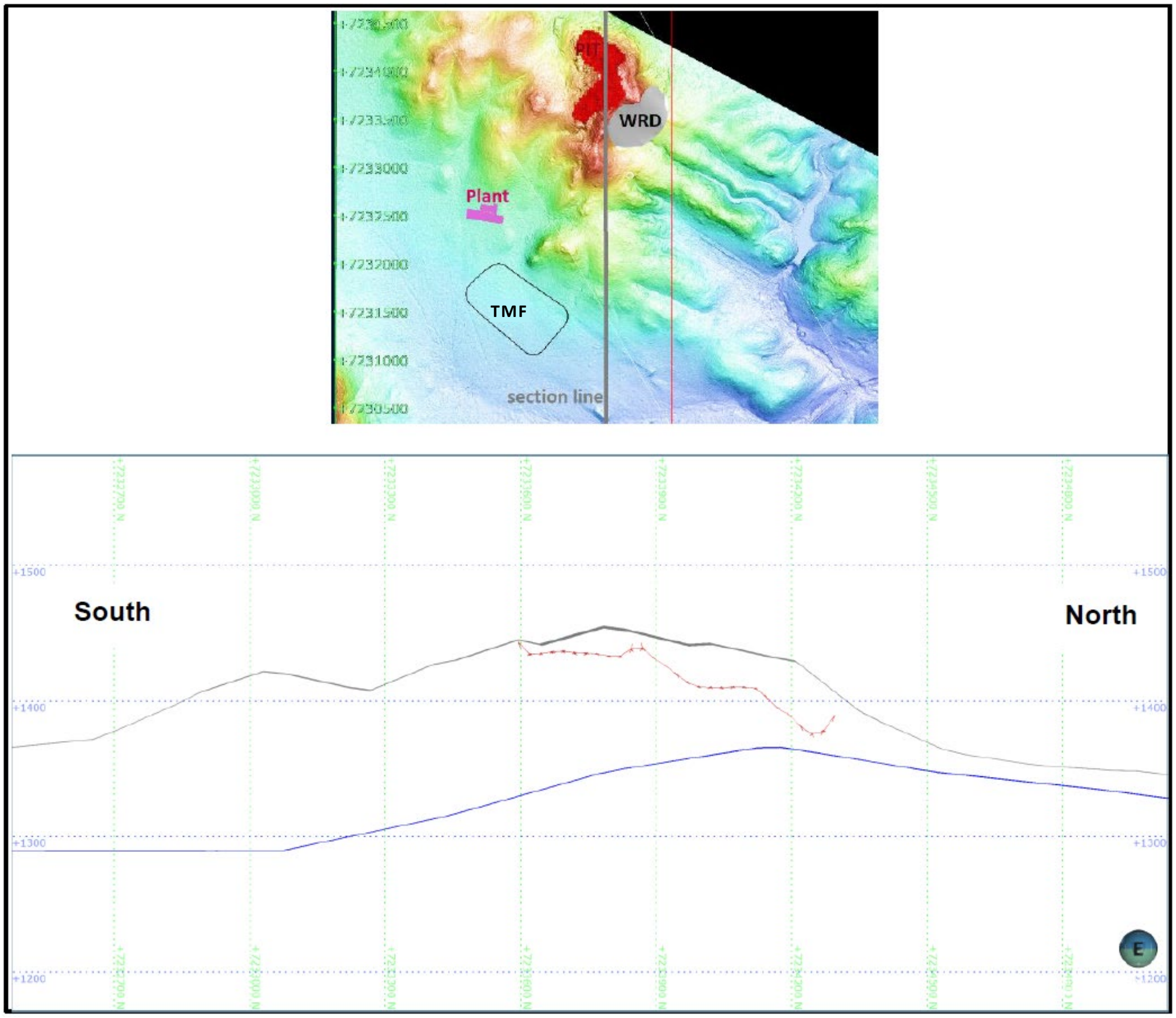
The extent of the proposed pit in the previously completed Technical Report (SRK and Tetra Tech 2022) is notably smaller than the proposed pit extent for the updated MRE (Section 14). The available mine layout plans show that the previous pit covered an area of approximately 35.23 ha, while the current pit covers an area of approximately 55.5 ha.

Based on a detailed study of the mining sequence in SRK and Tetra Tech (202), the final elevation of the pit floor would be above the water table, and therefore no groundwater inflows into the open pit (Figure 16.3). The main water inflows to be managed at the open pit would only comprise surface water runoff in the pit catchment.

The previously completed Technical Report (SRK and Tetra Tech 2020) calculated the anticipated volume of surface water runoff into the pit, and the results indicated that in the first year of mining, recurrent storms generating 4,000 m^3/day (roughly 170 m^3/hour) of water are expected to occur every 2 years. Additionally, in first year of mining, a total volume of approximately 16,000 m^3/day (roughly around 670 m^3/hour) of water should be expected to drain into the pit sump in 24 hours if a 1:100-year return period storm event occurs. Toward the end of mine life, when the pit extent is at its maximum, a 24-hour duration, 1:100-year return period storm event would generate as much as 54,500 m^3/day of water (roughly equivalent to 2,270 m^3/hour).

Surface water runoff into the pit has been recalculated based on the available updated open pit layout. The open pit extent for the first year of operation is not available; therefore, the surface water runoff into the open pit during the first year of operation cannot be updated for comparison. However, using the layout of the open pit at the end of the LOM (Year 50), it is calculated that surface runoff into the open pit area is expected to be approximately 85,850 m^3/day in the event of a 1:100-year return period storm event (equivalent to 3,577 m^3/hour). These calculations do not take into consideration possible backfilling of the open pit. It should be noted that this calculation has been completed to a PEA level of accuracy, and it is recommended that Giyani undertake a full surface runoff calculation as part of further studies.

Figure 16.3 North-south cross-section of the initial proposed pit and water table



Source: SRK and Tetra Tech (2022)

The mining sequence, based on the updated MRE in Section 14, will extend significantly deeper than the schedule proposed in the previous Technical Report, and it is more than likely that groundwater will be intersected during the LOM. The updated mining sequence contours relative to the interpolated groundwater levels are illustrated on Figure 16.4. It is evident that from approximately the central portion of the updated mining sequence (pit bottom at approximately 1,300 masl), all the way to the southern extension area, groundwater will be intersected based on the existing groundwater level information currently available.

Not only will there be a larger volume of surface water runoff flowing into the updated pit (due to the larger surface area of the updated MRE) as previously described, it is likely that there will also be a groundwater contribution flowing into the pit.

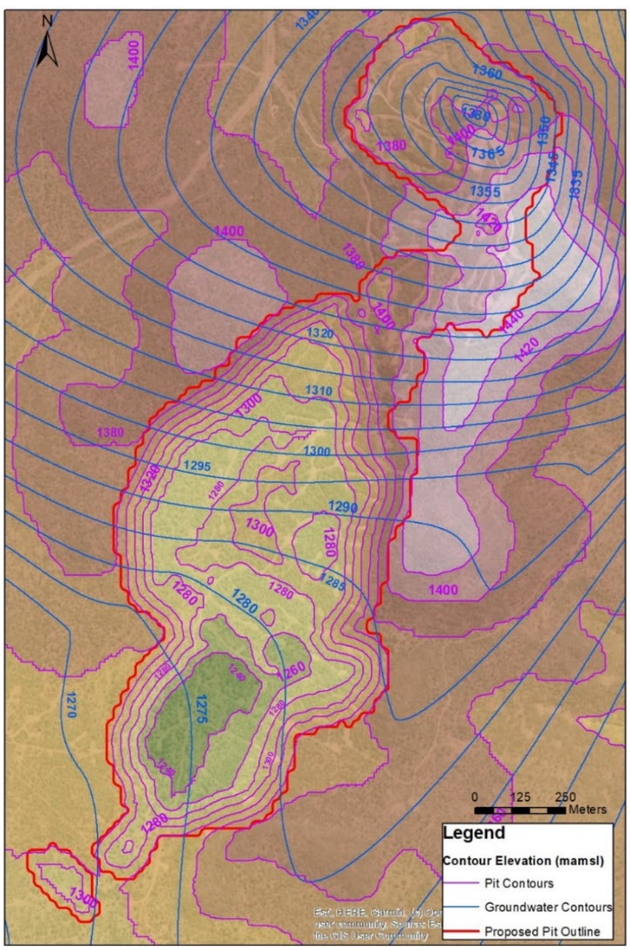
The expected groundwater inflows into the mined-out pit have been calculated using analytical methods. The analytical methods take into consideration aspects such as the aquifer transmissivity, depth to groundwater level, extent of the excavated pit, depth of the mine pit floor, recharge into the aquifers from rainfall, and aquifer storage. The available mine layout shows only the final pit extent and depth; therefore, only the expected inflows at the end of the LOM have been calculated. This inflow is expected to be representative of the upper range of inflow volumes, depending on actual aquifer recharge and transmissivities.

Groundwater inflow volumes into the excavated main open pit at the end of the LOM are expected to be in the range of 430 m^3/day to 690 m^3/day. Groundwater inflows into the southern satellite pit are expected to be in the order of 40 m^3/day to 50 m^3/day.

The following should be noted:

- There is no site-specific aquifer characterisation data available. The aquifer thicknesses, transmissivity, and storage have been obtained from literature.
- The calculations do not take into consideration potential backfilling of the open pit and the associated potential of water in storage in the backfilled material.
- Analytical methods are not ideally suited to calculate groundwater inflows in a dynamic scenario, such as an active mine. It is recommended that Giyani update inflow calculations using a 3D numerical groundwater flow model.

Figure 16.4 Proposed pit contours relative to the interpolated groundwater contours for the updated Mineral Resource estimate



Source: CSA Global

A numerical groundwater model was developed as part of the previous Technical Report (SRK and Tetra Tech 2022) of what is now the northern area of the deposit. Groundwater inflow into this smaller pit was not evaluated, as it did not intersect the water table. Given the updated MRE, the updated mining sequence, and the uncertainty associated with the analytically calculated pit inflow volumes previously described, it is recommended Giyani update the numerical groundwater model to include the proposed pit extension and to use the groundwater model to estimate the groundwater inflows into the pit. Using this information, the mine water balance should also be updated to ensure that enough water storage capacity is designed and installed at the processing plant site to effectively manage the additional volume of storm water due to the pit extension and to account for groundwater inflows.

In terms of pumping infrastructure, it is recommended to make provisions for a dewatering system capable of handling the 1:10-year return period storm runoff (as per the recommendation in SRK and Tetra Tech [2022] but taking into consideration the larger pit footprint area) as well as the groundwater inflow contributions.

Water that is pumped from the pit can be used for dust suppression and as preferential processing plant raw water makeup to reduce the water demand from other sources.

As the open pit is likely to require dewatering, the aquifer will be drained (to a certain extent) and locally there will be a drawdown in groundwater levels. The extent of the zone of influence of the drawdown cone has been calculated using analytical methods to range between 60 m and 150 m. The extent of the zone of influence is limited by the expected low permeability of the host geology. It is recommended that Giyani simulate the extent of the dewatering cone/drawdown to a higher level of certainty by undertaking a numerical groundwater model during later stages of the Project. While it is unlikely that the dewatering of the pit, which is situated within the low permeable aquifer associated with the shales, will have any significant impact on the adjacent high-yielding dolomitic aquifers (which are used for groundwater supply to Kanye and surrounding farms), this will need to be confirmed in a numerical groundwater model. It is also recommended to drill a network of groundwater monitoring wells around the proposed open pit footprint so that groundwater levels can be monitored throughout the LOM.

16.4 Mine geotechnical considerations

16.4.1 Limitations

A number of limitations with respect to this geotechnical assessment are noted as follows:

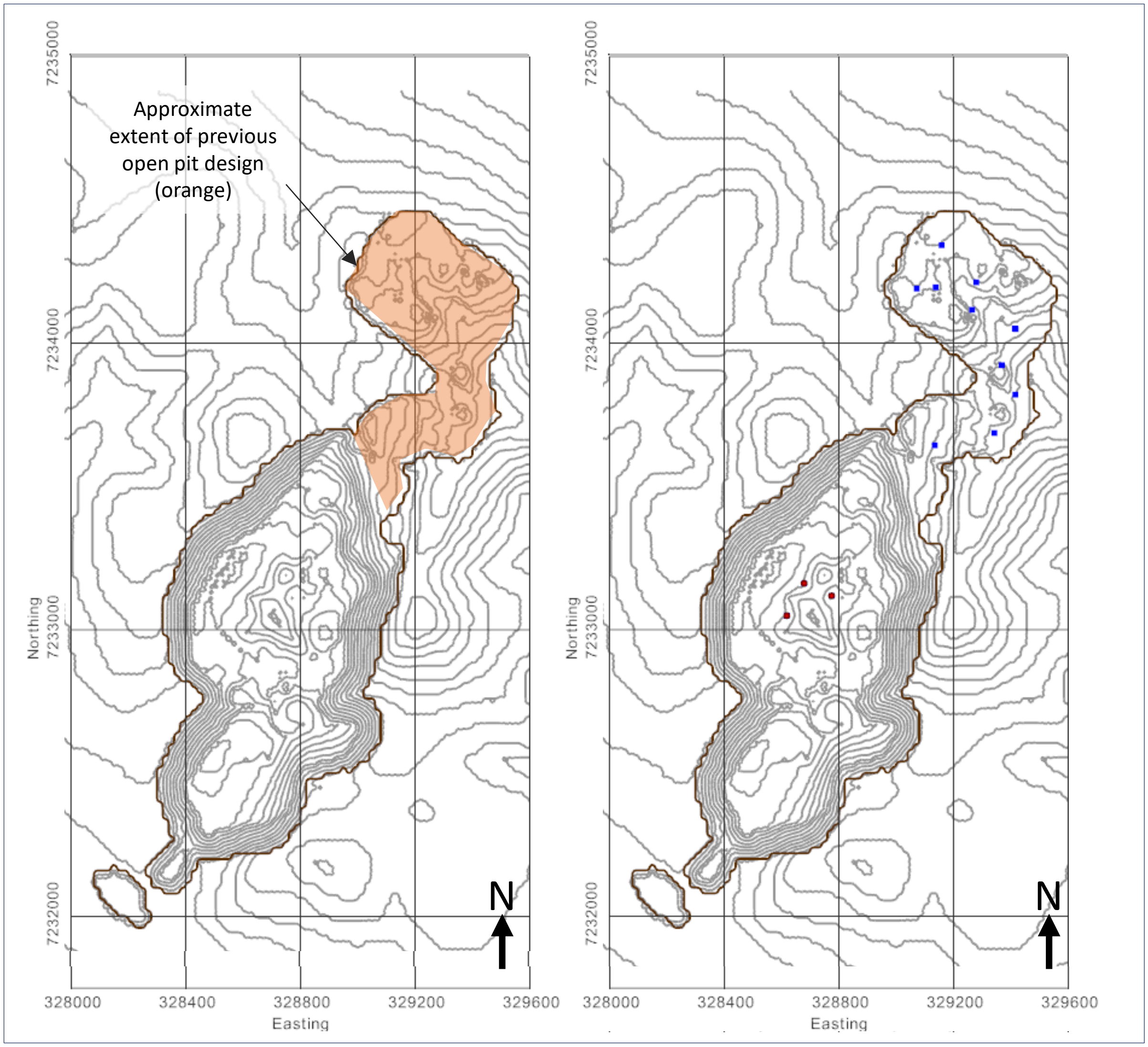
- The geological models do not extend to the vicinity of pit walls; geology has been extrapolated using judgement.
- Available geotechnical data predominantly comes from <30 m in depth in the northern open pit area completed for the previous Technical Report (SRK and Tetra Tech 2022) and may not be representative of the material that will host the pit walls, especially at depth and in the southern and central parts of the pit (where the pit walls are the highest).
- Hydrogeological data is limited, and it is assumed that the pit walls will be dewatered and depressurised during mining.

- Limited information on major and minor structures is available, and an assessment of structural instability mechanisms has not been undertaken.
- Rock mass shear strengths have been adopted from the previously completed Technical Report (SRK and Tetra Tech 2022), based on limited additional data.
- A single representative section has been analysed for each pit wall; other areas of the pit will have different factors of safety; further geotechnical analysis is considered likely to identify areas that will need to be flatter than those areas used in the optimisation and other areas which can be steeper.
- WRDs have not been considered as part of this geotechnical analysis.

16.4.2 Slope design

The extent of the selected open pit (contoured) is shown on Figure 16.5 as well as the approximate area of the previous pit design (shaded in orange; SRK and Tetra Tech 2022), which formed the basis of the historical geotechnical assessment. It is evident that the pit has been extended significantly to the south, with slopes of the proposed pit now exceeding 100 m in height. The highest pit slopes can be seen to be outside the shaded area, and distal to the area of geotechnical investigation undertaken during the previous Technical Report (SKR and Tetra Tech 2020; Figure 16.5, right). Geotechnical logging from three additional drillholes has been provided (collars shown in Figure 16.5, right).

Figure 16.5 Contours of pit optimisation



Notes:
Optimized Pits_2 Pit 76 100.dxf, pit limit in brown, and topography
Left: approximate extent of pit design from SRK and Tetra Tech 2022; right: collars from SRK and Tetra Tech 2022 (blue) and additional diamond drillholes with geotechnical logging (red)
Source: CSA Global

16.4.3 Geotechnical model

Geological and structural models

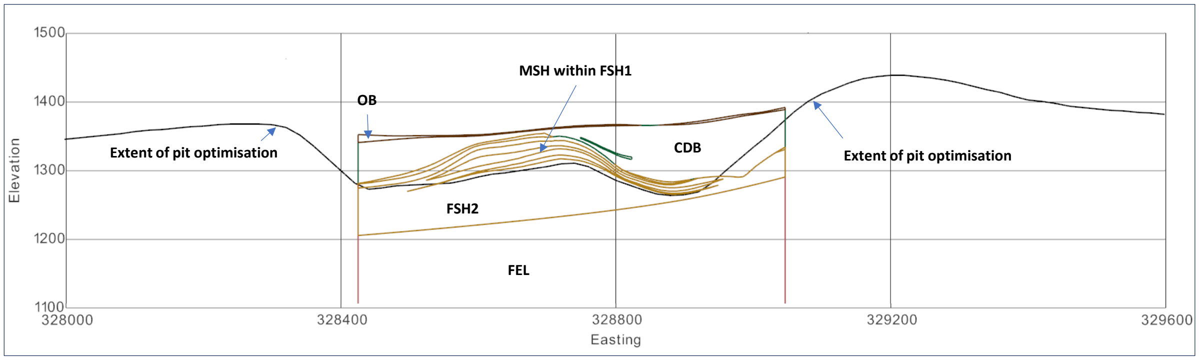
The geological model provided for the open pit area is similar to that used in the previous Technical Report, with the main units comprising:

- overburden (OVB)
- chert dolomite breccia (CDB)
- Fe-shale (FSH 1)

- Mn-shale (MSH – multiple wireframed solids)
- lower Fe-shale (FSH 2)
- felsite (FEL)

The cross section presented in Figure 16.6 shows the mineralised MSH units to be hosted within the upper and lower FSH units. The shales are overlain by CDB and minor overburden. The shales (FSH 2) uncomfortably overly the FEL unit. The geological model is constrained to the mineralised areas and does not extend behind the pit walls.

Figure 16.6 Cross section showing the relationship of the geological units and the limits of the geological model relative to the proposed pit optimisation



Note: Viewing north at 7,233,000 mN
Source: CSA Global

Structural models have not been used in this assessment; it is understood that vertical faulting has been interpreted within the area, which is unlikely to influence slope design at this level of assessment. It is understood from the previous Technical Report that the stratigraphy may have been duplicated by the presence of thrusting; such features could have a significant influence on slope stability and should be investigated for future phases of assessment.

Hydrogeology

As discussed in Section 16.3.3, the increased depth of the selected pit has resulted in the potential for the lower pit wall to extend below the inferred water table and that pit dewatering will be undertaken using sumps on the pit floor. For this assessment, it is assumed that limited vertical advance rates will allow dewatering of the pit walls in line with the advance of mining, and as such, pit walls are assumed to be dewatered and depressurised. There is currently a limited understanding of groundwater conditions and further work should investigate likely seasonal variations in groundwater levels and the potential of elevated groundwater to influence pit wall stability.

Rock mass conditions

Logs and core photographs of diamond drilling to the depth extent of the proposed pit in the southern area were reviewed (DDKH21_029, DDKH21_030, and DDKH21_031). The core generally appears to indicate similar conditions to those seen in the previous Technical Report (SRK and Tetra Tech 2022); however, high-level validation checks highlighted a number of apparent issues with the logs, including:

- possible over-estimation of rock quality designation (e.g., DDKH21_029, 36.7 m to 39.7 m) and commensurate under-estimation of degree of fracturing
- assigned strength values appear to be high in the shale units (FSH and MSH)
- assigned strength values appear to be low in the FEL unit
- limited joint data recorded, especially in the shale units (FSH and MSH)

Laboratory testing of samples from these drillholes has not been undertaken, meaning field strength estimates cannot be validated/checked against laboratory results.

As a result of these findings, rock mass conditions for this assessment have been adopted from the previous Technical Report in the northern area of the deposit (SRK and Tetra Tech 2022). An additional FEL unit has been added, with parameters estimated based on observations of drill core. Input parameters for rock mass shear strengths using the Generalised Hoek-Brown strength criterion are presented in Table 16.2.

Table 16.2 Adopted rock mass shear strength parameters

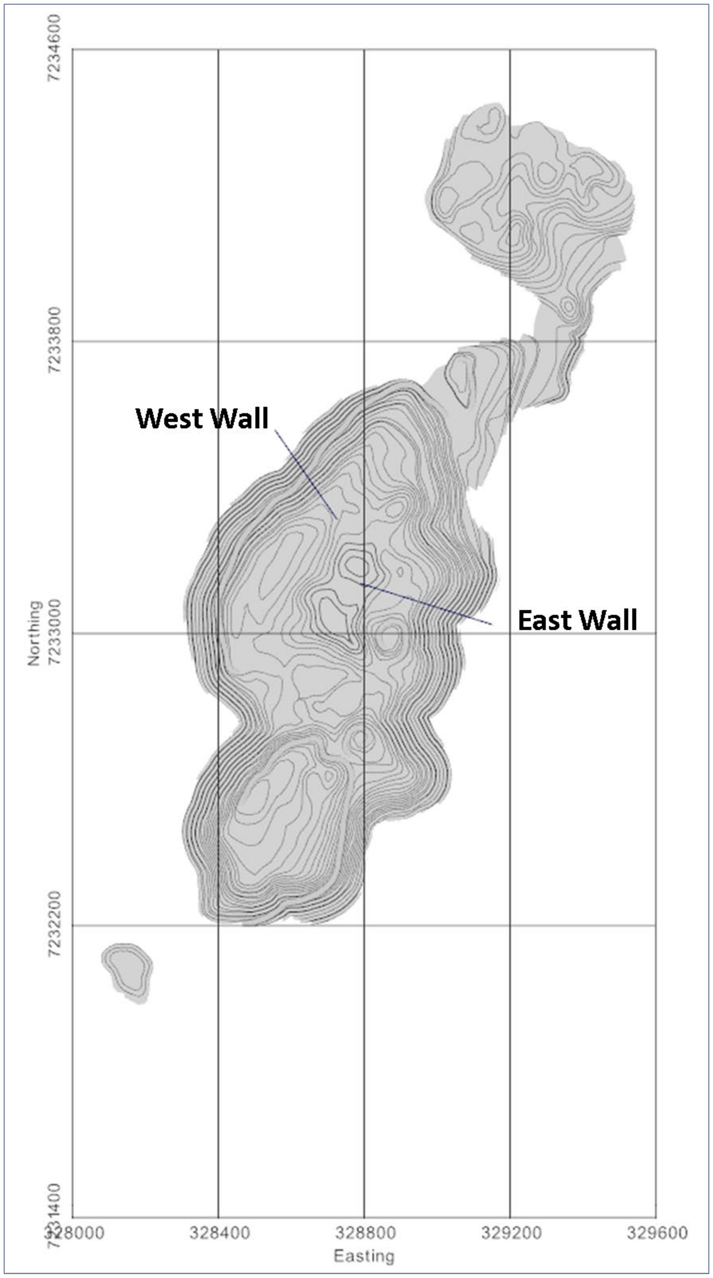
Unit	Colour (in Figure 16.8)	Unit Weight (kN/m^3)	Strength Type	UCS (MPa)	GSI	mi	D	Comment
CDB		25.2	Generalized Hoek-Brown	75	25	9	0	After SRK and Tetra Tech 2022 (Undisturbed)
FSH		22.8		5	32	6	0	
MSH		22.8		5	33	6	0	Estimated (Undisturbed)
FEL		25.2		37.5	50	25	0	

Notes:
UCS - unconfined compression strength
GSI - geological strength index
mi - Generalized Hoek-Brown constant
D - Generalized Hoek-Brown disturbance factor

16.4.4 Stability analysis

Given the available data and findings from the previously completed Technical Report, limit equilibrium analysis of maximum slope heights on the hanging wall (west wall) and footwall (east wall) has been undertaken. The pit design was provided for this analysis; the design and line of sections are shown in Figure 16.7. The sections selected are not critical (worst-case) sections, rather target representative sections of the proposed design. Given the extents of the geology model, the stratigraphic boundaries have been extended to allow lithological units to be assigned.

Figure 16.7 Representative sections selected for stability analysis



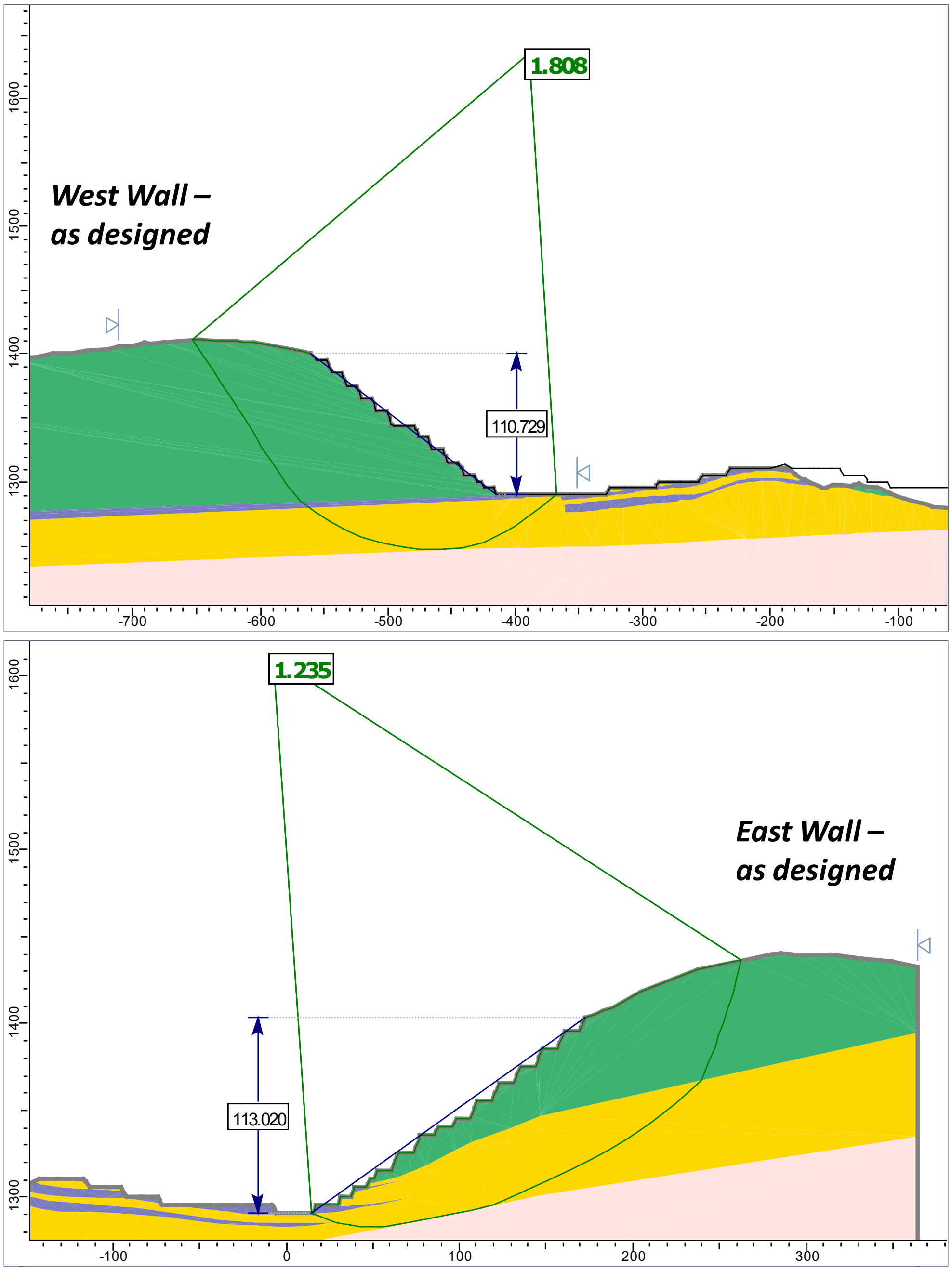
Source: CSA Global

Stability analysis was carried out using non-circular Cuckoo search algorithm, with the factor of safety from the GLE method reported. Results of the stability analysis are summarised in Table 16.3 and presented in Figure 16.8. Results of stability analysis require comparison to Design Acceptance Criteria (DAC); a factor of safety of 1.2 has been provided as the DAC. Analysis shows both sections exceed the DAC, and as such the design is considered appropriate for the PEA.

Table 16.3 Results of stability analysis

Pit wall	Approximate slope dimensions – critical section	Factor of safety
West wall	110 m high, 39°	1.8
East wall	113 m high, 38°	1.24

Figure 16.8 Results of slope stability analysis, west (top) and east (bottom) walls



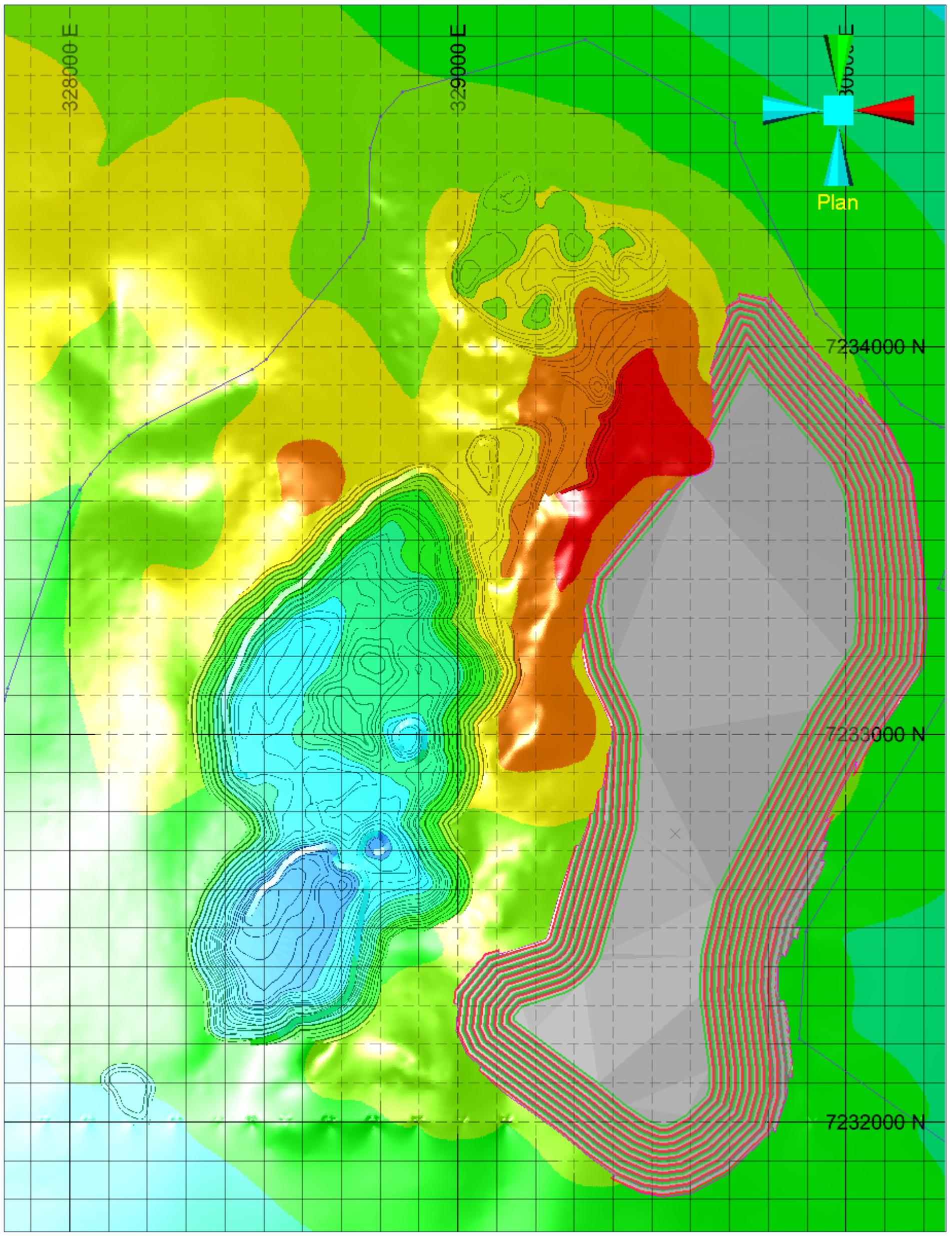
Source: CSA Global

Further geotechnical investigation and data collection will need to be undertaken for future studies. Improved confidence in geological, structural, and hydrogeological models in and behind pit walls will also be required. This will allow for the development of geotechnical domains and appropriate recommendations on pit slope angles.

16.5 Mine design

The selected pit shell for the mine design falls between the RF 0.6 and RF 0.7 pit shells and targets a diluted grade in excess of 15% manganese oxide. Five mine stages were created within the designed pit limit, as shown in Figure 16.9.

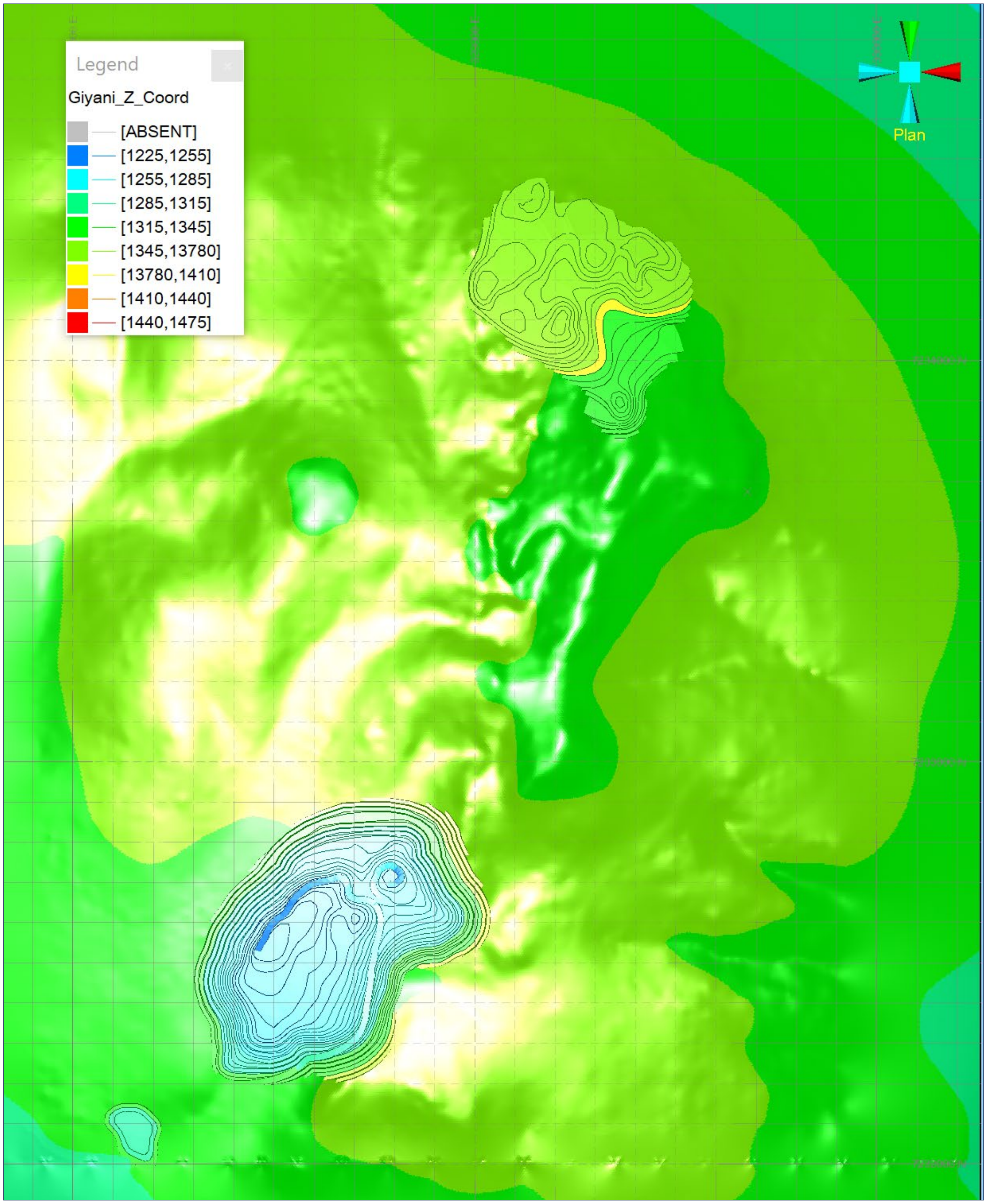
Figure 16.9 Pit limit design, waste rock dump, and site boundary limit



Source: Axe Valley Mining

Designs were initially created for pit stages 1 and 2, as shown on Figure 16.10.

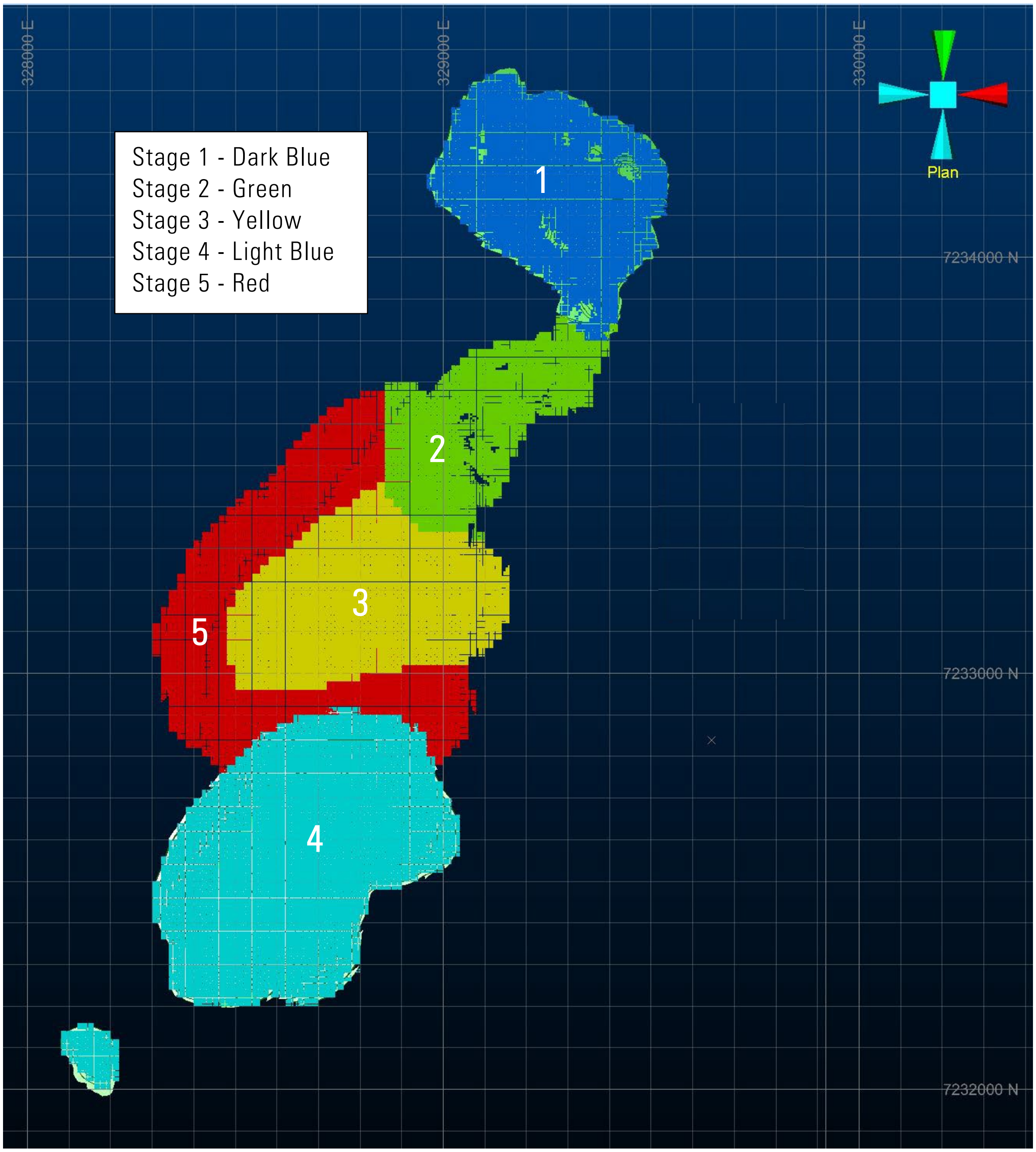
Figure 16.10 Pit stages 1 and 2



Source: Axe Valley Mining

The mine sequence is assumed to start with the north pit area (Stage 1), which is very similar to the pit design proposed in the previous Technical Report (SRK and Tetra Tech 2022). Additional stages were then added in NPV Scheduler between Stages 1 and 2 to match the pit limit design (Stage 5). The final pit staging layout is shown in Figure 16.11.

Figure 16.11 Final pit staging



Source: Axe Valley Mining

The mineable inventory by pit stage is listed in Table 16.4.

Table 16.4 Mineral inventory by pit stage

	Units	Pit Stages					Total
		1	2	3	4	5	
Total Mined	kt	9.363	8,481	30,988	47,207	38.790	134,828
Waste Mined	kt	7,750	7,528	28,199	43,948	36,367	123,792
ROM	kt	1,614	961	2,815	3,289	2,446	11,125
Strip Ratio	-	4.8	7.8	10.0	13.4	14.9	11.1
ROM	MnO (%)	18.18	17.53	15.24	13.79	14.15	15.20
Grades	Fe_2O_3 (%)	14.55	12.19	11.54	9.45	10.37	11.15
	Al_2O_3 (%)	10.62	8.06	8.31	6.39	8.18	8.03
	SiO_2 (%)	40.37	46.35	48.01	55.95	50.16	49.58
	K_2O (%)	2.19	2.16	1.95	1.60	1.34	1.77

Notes:
Modifying factors of 98%mining recovery and 3% dilution have been applied

The pit stages were designed with 10 m benches (mined on 5 m flitches), a batter angle of 70°, and 7.9 m catch berms, for an overall slope angle of approximately 41°.

16.6 Cut-off grade

The calculated economic (mill) cut-off grade of 7.2% manganese oxide is based on the parameters listed in Table 16.1. The cut-off grade was adjusted to 7.4% manganese oxide to provide a minimum profit margin.

Material above the selected cut-off grade was divided into several grade bins, which are used to segregate the ROM material into high-grade, medium-grade, and low-grade material streams.

Table 16.5 Grade bins

Grade bin	Cut-off (MnO %)	Comments
Low grade	7.4	Economic cut-off grade
Medium grade	9.0	Operational cut-off grade
High grade	10.0	Operational cut-off grade

Depending on the operating cut-off grade, these material streams can then be directed either to a long-term (low-grade) stockpile or to the active ROM pad where it will be fed directly to the crusher. Material below the economic cut-off grade will be sent to the WRD.

The grade bins have been selected based on the grade tonnage profiles for the various pit stages, such that up to 2 Mt of lower-grade material can be stockpiled over the LOM. This helps increase revenue in the early years by processing high grade material at >10% manganese oxide.

There has been no attempt in this study to segregate the material by rock type of deleterious elements; this may be necessary once further metallurgical test work has been completed and the blending strategy has been defined.

16.7 Open pit mine schedule

The mine schedule was created in Microsoft® Excel using the mineral inventory by bench for each pit stage. The mining rate then was adjusted to create a smoothed mining sequence that targets a processing plant feed rate of 200 kt/a. Other constraints include:

- a maximum mining rate of 3.5 Mt/a
- a maximum manganese oxide feed grade of 21%
- a minimum processing plant feed grade of 10% manganese oxide
- a maximum sinking rate of nine benches per year per pit stage

To control the feed grade to the processing plant, the reclaim rate from the high-grade stockpile will be controlled such that high-grade material is stockpiled in the first few years to limit the feed grade to 21% manganese oxide. The mining rate in the early years will also increase by applying a high grading strategy, where only high-grade material (≥10% manganese oxide) will be preferentially fed to the processing plant over the first 15 years. The exception to this will be in the first 12 months of plant ramp-up, where lower-grade material will be fed during the commissioning period.

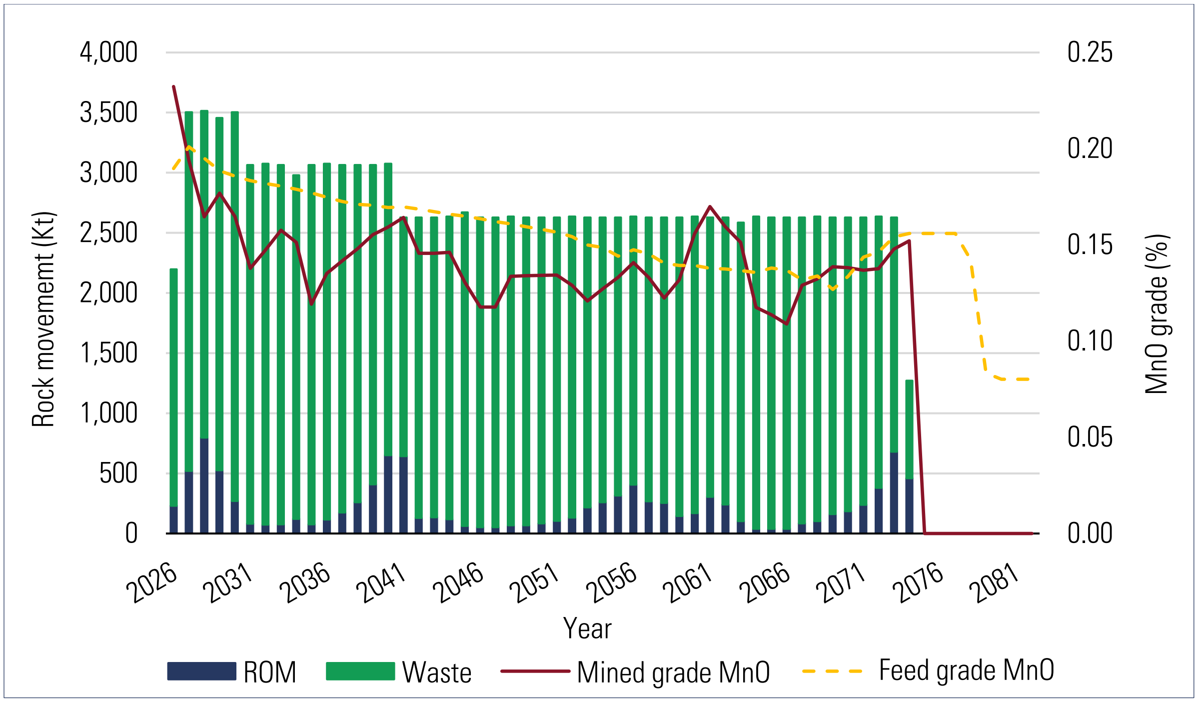
The mining sequence is based on maximising project value by targeting material with a higher profit per tonne and a lower stripping ratio. The mining schedule essentially follows the sequence Stage 1, 2, 3, 4 and 5. However, there are overlaps between mining stages to maintain mineralised material exposure and avoid spikes in waste stripping. This is particularly important with the southern extension area of the open pit (Stages 3, 4 and 5) where the strip ratio generally exceeds 10:1.

The schedule was constructed in quarterly periods for the first 5 years and then in annual periods for the remaining LOM. The period durations ensure that the processing plant ramp up over the first 12 months can be modelled with reasonable accuracy and that variations in grades can be observed. The schedule was then annualised and is summarised in Figure 16.9 and Figure 16.10.

It can be seen in Figure 16.9 in the difference between the grade profile of material mined and the grade profile for material fed from the ROM stockpile to the processing plant, that the high-grading strategy significantly increases the feed grade over the first 30 years of mine life by stockpiling the lower-grade material. The exception is in the first year of operations, where the feed grade to the processing plant is capped at 21% manganese oxide.

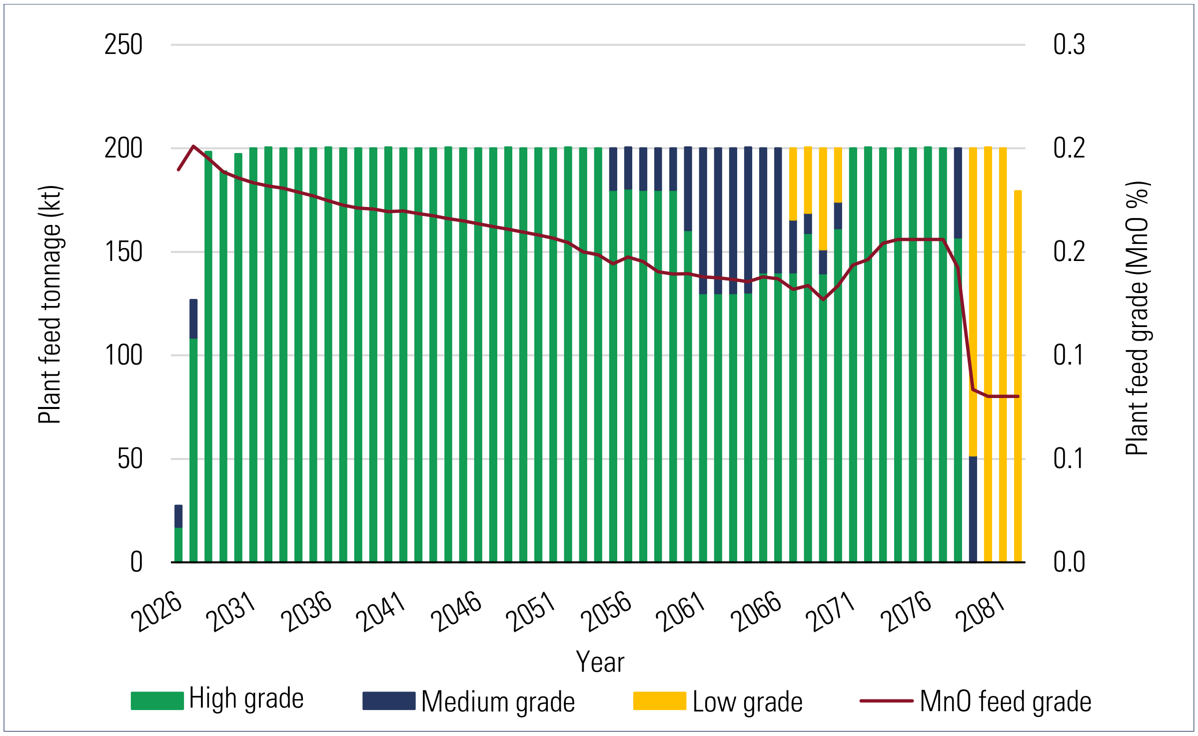
Besides increasing project value, the stockpiling strategy will help provide a consistent quality (grade and blend of material types) to the processing plant. This must be balanced against the cost of managing and reclaiming the stockpile, and an allowance of US$1.00/t has been made in the cost model. The location of the stockpile should be as close as possible to the ROM pad area to reduce costs.

Figure 16.12 Pit optimisation results for a 200 kt/a production target



Source: Axe Valley Mining

Figure 16.13 Plant feed schedule (200 kt/a)



Source: Axe Valley Mining

16.8 Waste disposal

The WRD is assumed to be located east of the pit (Figure 16.1), with a total design capacity of approximately 63 Mm^3 (loose) if built to the RL 420 elevation. This provides sufficient capacity for the LOM schedule (64 Mm^3) after allowance (including 10% contingency) is made for 12 Mm^3 of rock used in the TMF construction. The loose volumes are based on a waste density of 2.48 t/m^3 and an overall swell of 30%.

The required WRD capacity could be further reduced by considering backfilling of pit stage 1 and 2 whilst mining stages 4 and 5. This would also help to bring forward the rehabilitation of the site.

The WRD has been designed with 10 m lifts and a 28° overall slope angle. This compares with the 20° overall slope angle used in the previous Technical Report (SRK and Tetra Tech 2022) and is justified on the basis that the updated pit design will be a mix of weathered material and fresh rock, whereas the previous pit design was primarily weathered material with a pit depth of <90 m. The more durable fresh rock can be used to encapsulate the weathered material and thereby improve the overall slope angle.

The positioning of the WRD, between the open pit and Kanye to the northeast, will help to provide a sound barrier and reduce the visual impact of the mining areas. Strict controls on dust suppression will be important. Allowance has also been made to restrict the WRD by the proposed property boundary limit to the east of the WRD.

16.9 Open pit mining equipment

The location of the Project, close to the South African border and Botswana diamond mines, ensures good access to mining equipment, spares, and contract mining services. There is also a large market of second-hand mining equipment that could be utilised to reduce initial capital expenditure for the mining fleet, should a decision be made to go with an owner-operated mine.

For this exercise, it is assumed that the mining contractor will be responsible for all activities in the open pit and for supplying and maintaining any equipment. The mining contractor will have access to a vehicle workshop for tyre and lubrication services; engine overhauls and remedial maintenance will be completed offsite. Allowances for site establishment and building and maintaining the TMF are not included in the mining equipment requirements.

The assumption for the mining operation is that work will be done 365 days/year, using a single 12-hour shift per day. Preliminary calculations for the load and haul requirements have revealed that up to four hydraulic excavators with a 3.8 m^3 bucket can deal with the total material movement for the LOM. A smaller backhoe will be used as a back-up to the main excavator and for small jobs and mining ROM material while the main excavator is mining waste.

Mining is assumed to be on 10 m benches with 2.5 m or 5 m flitches to ensure the benches can follow the general dip of the deposit and minimise material loss and dilution. The selected mining equipment is capable of mining the waste benches on 10 m intervals, if required, to improve loading efficiency.

As noted in previous studies (SRK 2019), it is assumed that the excavator will be able to initially break at least 50% of the ground at site without the need for blasting. This will be tough to achieve for the breccia, and a ram should be bought to install on the excavator to break the rocky cap before proper mining begins.

A relatively high overall utilisation of 75% is assumed for the excavators based on redundancy in the loading fleet and the flexibility to schedule maintenance during the off shift, as required. A typical productivity of 300 t/hour is assumed.

The assumption for haulage calculations is an average of 1.5 km haulage for waste and a 3.5 km haulage for ROM or stockpile material. The average speeds have been estimated at approximately 20 km/hour for loaded trucks; this means that the operation will require up to 14, 30 t articulated dump trucks, with a spare to ensure continuity of operations during maintenance.

One front-end loader will be used to feed the processing plant at the ROM pad. One track dozer will be used to construct mine roads and level the dumps, and a further track dozer will support the operations within the open pit. Considering the distance between the WRD and the open pit, it is considered impractical to use a single track dozer; therefore, one track dozer will be stationed at the WRD and one at the open pit.

One drill rig (105 mm) and one RC grade control rig have been allowed for based on a proportion of the material near surface being free dig. An additional drill rig may be required as the operations move to the south pit area and the pit become deeper. Further geotechnical studies are essential to establish the rock characteristics for open pit and WRD design.

Provision has also been made for two rubber tyre dozers to support the excavators with clean-up duties and road maintenance. In addition, two graders will be available for road work around the site and at the WRD.

A 23,000 L water bowser will be used for dust suppression, which is important considering the proximity of Kanye. Four light vehicles have been budgeted for, as well as lighting plants, radios, and other light equipment. The equipment requirements in Year 2 are summarised in Table 16.6.

Table 16.6 Equipment requirements

Equipment type	Model/size	Units
Hydraulic excavator	3.8 m^3	4
Haul truck	30 t	15
Wheel loader	Cat 966	1
Drill rig	105 mm	1
Grade control rig	Generic	1
Track dozer	D10	2
Rubber tyre dozer	Cat 844	2
Grader	Cat 14m	2
Water truck	23,000 L	1
Fuel/lube truck	Generic	1
Blast truck	Generic	1
Service truck	Generic	1
Lighting plants	Generic	10
Light-duty vehicle	Toyota	8
Crew bus	Generic	2

16.10 Open pit mining labour

The open pit labour requirements are based on single 12-hour shift, operating 365 days/year, with a two-crew roster working 4-days on and 4-days off. The required mine layout requirements are summarised in Table 16.7.

Table 16.7 Mine labour requirements

Equipment type		Roster	Number	Comments
Mine manager	Owner	Dayshift	1	5-day/week coverage
Mine planner	Owner	Dayshift	1	5-day/week coverage
Geologist	Owner	Dayshift	1	5-day/week coverage
Samplers	Owner	Dayshift	4	5-day/week coverage
Laboratory staff	Owner	Dayshift	4	7-day/week coverage
Surveyor	Contractor	Dayshift	1	5-day/week coverage
Survey assistants	Contractor	Dayshift	2	5-day/week coverage
Foreman	Contractor	Two crews	2	7-day/week coverage
Operators	Contractor	Two crews	65	10% absenteeism
Maintenance supervisor	Contractor	Dayshift	2	7-day/week coverage
Maintainers	Contractor	Two crews	6	7-day/week coverage
Administrator	Contractor	Dayshift	4	5-day/week coverage
Security	Owner	Two crews	4	7-day/week coverage

It is assumed that the owner's technical team will provide the technical support to the mine contractor which will include:

- managing the overall Project
- collecting geological data
- analysing samples
- updating the geological model
- preparing mine plans
- providing site security

To minimise the labour requirements, there will be a mix of 5-day and 7-day coverage, with two crews required for full coverage over the 7 days. The owner will mainly work a 5-day week.

17 RECOVERY METHODS

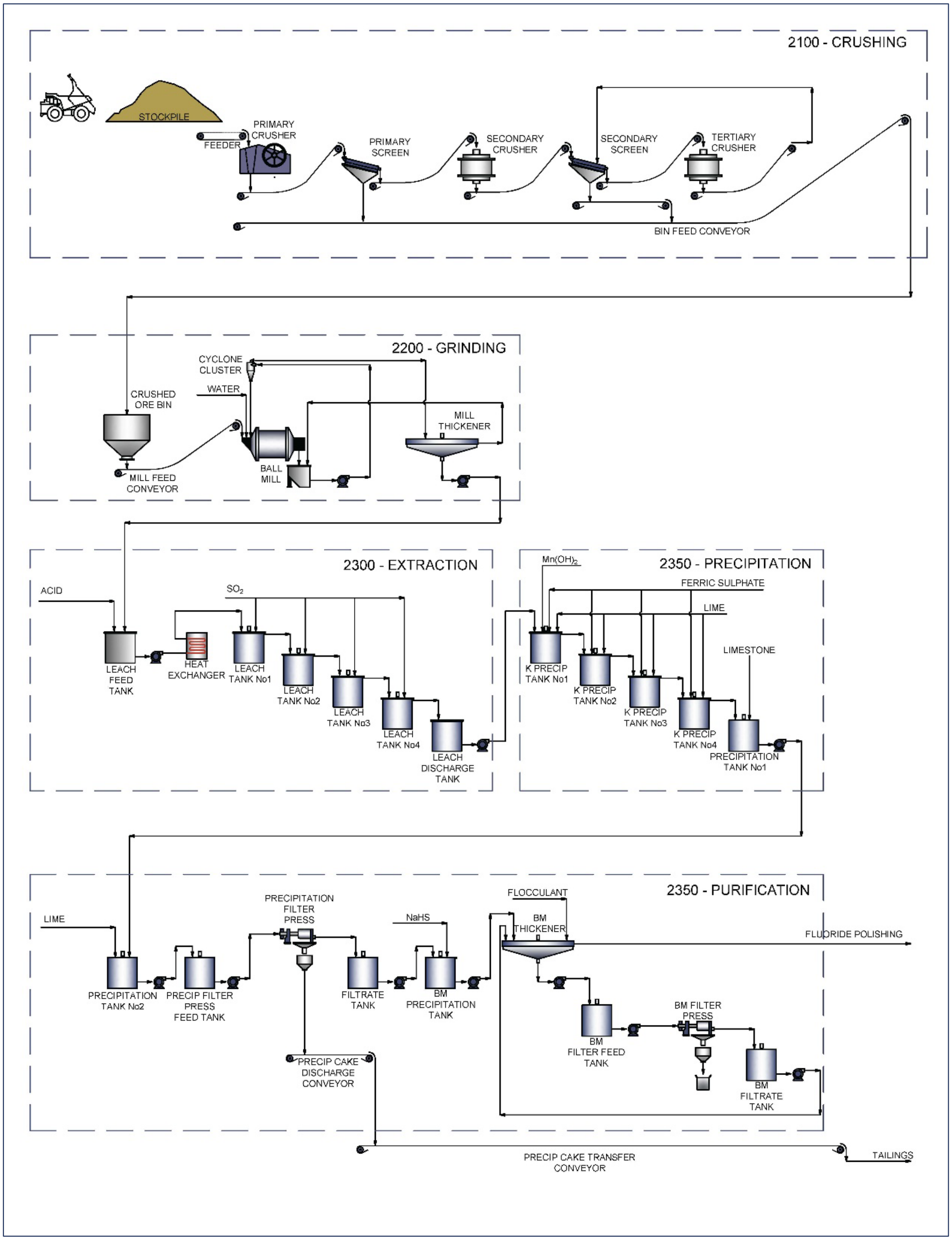
The processing plant for the production of HPMSM will treat 200 kt/a of ROM material from the open pit at an average grade of 15.2% manganese oxide.

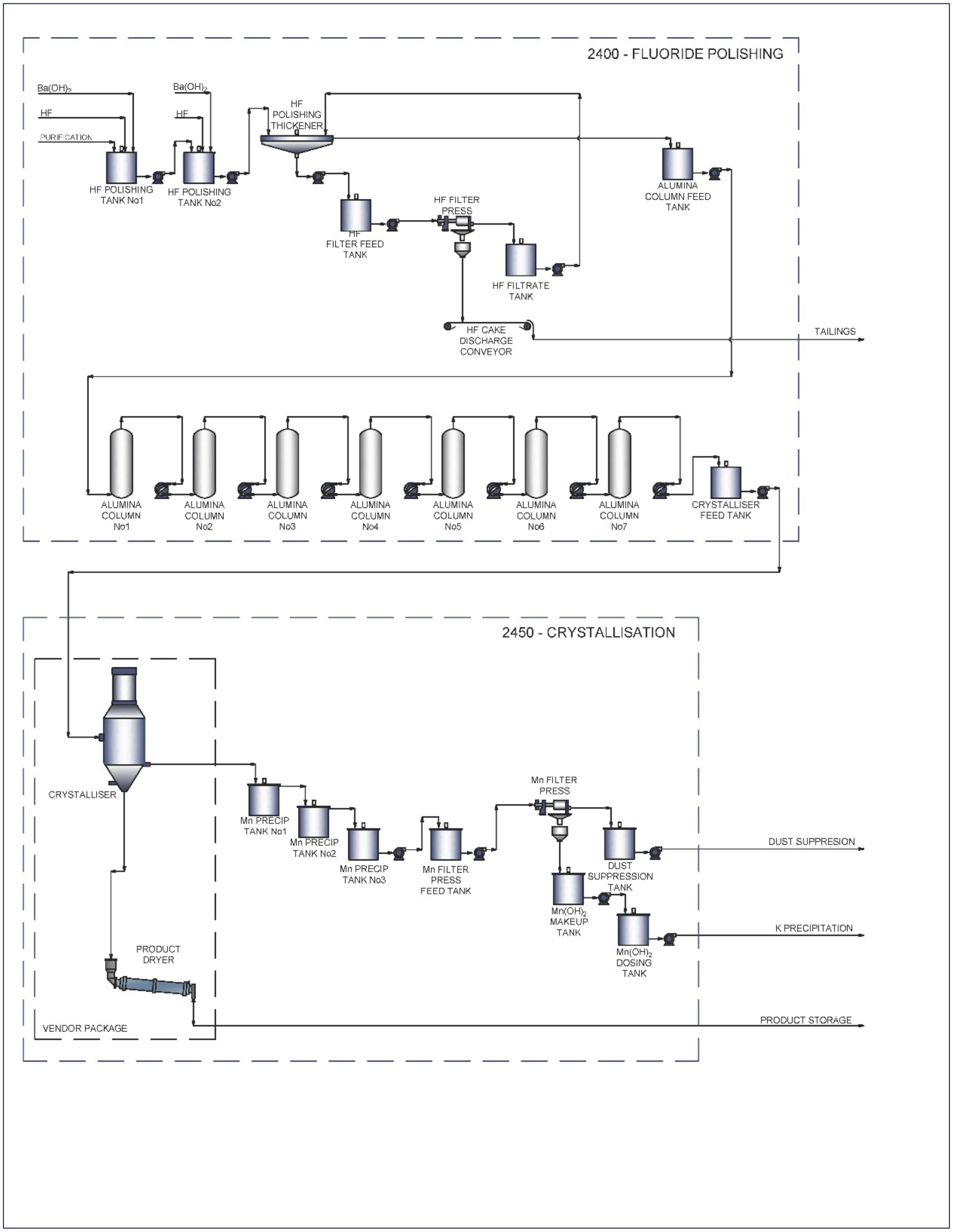
The mineralised material comprises manganese and iron shales, is moderately hard, and is amenable to reductive acid leaching in sulphate media using sulphur dioxide as a reductant. The process comprises crushing and grinding to reduce the ROM material to a characteristic grind (P_{80}) of 150 μm, an acid reductive leach in sulphate media at an elevated temperature using sulphur dioxide as a reductant, followed by sequential precipitation and purification processes for the removal of metal impurities. Fluoride polishing is undertaken to further improve the purity of the solution. The purified solution then undergoes evaporative crystallisation, followed by filtration and drying of the product, to produce an HPMSM final product.

The solids removed during sequential purification and fluoride polishing will be disposed of in the TMF or stored as an intermediate product. All liquors removed in the treatment of the mineralised material will be treated for reuse or used for haul road dust suppression.

A simplified process flow diagram illustrating a single treatment module is presented as a schematic in Figure 17.1.

Figure 17.1 Schematic of the manganese recovery process





Source: Tetra Tech

17.1 Basis of design

Table 17.1 summarises the basis of the process design for the HPMSM recovery process.

Table 17.1 Basis of the process design

Item description	Units	Value/comment
Operating time		
Days per annum	days/year	365
Statutory holidays	days/year	0
Operating days available	days/year	365
Operating hours per day: crusher	hours/day	16
Operating hours per day: mill	hours/day	24
Operating hours per year: crusher	hours/year	3,340
Operating hours per year: mill	hours/year	7,720
Throughput (dry basis)		
Plant throughput	t/a ROM	200,000
Head grade: MnO	w/w (%)	15.2
Overall plant recovery: Mn	%	88.5
Leach dissolution	%	96.8
Precipitation losses	%	0.5
Fluoride polishing precipitation losses	%	4.0
Crystallisation recovery: Mn	%	89.6
Precipitation recovery: Mn	%	97.0

Multiple process stages are used to achieve an overall recovery of 88.5%. Each process is described further in Sections 17.2 to 17.7.

Test work was carried out on the OLOM composite sample, which was selected to represent the K.Hill North orebody, as well as material selected from the B Horizon. Test results from these samples were used to develop the process design criteria, flow sheets, and equipment sizing. Extraction test work carried out on B Horizon samples demonstrated that this material has similar extraction characteristics to K.Hill North orebody.

17.2 Crushing

The crushing circuit will comprise three stages of crushing with interstage screening to reduce the ROM material size to the required size for the grinding section. Because of the relatively small scale of the operation, the off-the-shelf components of the comminution circuit will be able to achieve a throughput capacity of up to 400 kt/a.

17.2.1 Run-of-mine feed and primary crushing

A dump truck will feed the ROM material to a ROM stockpile. Material will be collected by a front-end loader and fed onto the primary crusher feeder. The crusher feeder will discharge onto a primary grizzly feeder. The oversize material from the grizzly feeder will be fed into the primary crusher. The product from the primary crusher and undersize from grizzly feeder will report to the primary crusher discharge conveyor. The conveyor will have an electromagnet at the discharge end of the belt to remove tramp iron.

17.2.2 Secondary and tertiary crushing

The material discharged from the primary crusher discharge conveyor will pass over a primary vibrating screen. The oversize material from the screen will feed into the secondary cone crusher via the secondary crusher feed conveyor.

The tertiary cone crusher will operate in closed circuit with the secondary vibrating screen. The secondary crusher discharge conveyor will transport a combination of the product from the secondary cone crusher and the product from the tertiary cone crusher. This material will be fed onto the secondary vibrating screen. The oversize from the secondary vibrating screen will be fed onto the tertiary crusher feed conveyor, which will discharge into the tertiary cone crusher. The product from the tertiary cone crusher will discharge onto the tertiary crusher discharge conveyor, which will feed onto the secondary crusher discharge conveyor. The undersize material from the secondary vibrating screen will discharge onto the secondary vibrating screen discharge conveyor.

The mill bin feed conveyor will be fed from the undersize material from the primary vibrating screen and the discharge from the secondary vibrating screen discharge conveyor. The mill bin feed conveyor will discharge to the ball mill feed bin.

17.3 Grinding

The grinding circuit will consist of a ball mill in a closed circuit with cyclones. The overflow from the cyclones will feed into a mill thickener. The mill circuit will reduce the material size to a P_{80} of less than 150 µm and ensure that the slurry fed to the extraction area is at the required slurry density.

17.3.1 Milling

The mill bin feed conveyor will be fed with undersize material from the primary vibrating screen and the discharge from the secondary vibrating screen discharge conveyor. The mill bin feed conveyor will then discharge to the ball mill feed bin. The material in the ball mill bin will be removed by the ball mill feed bin feeder and discharged onto the ball mill feed conveyor. The ball mill feed conveyor will be equipped with a weightometer to measure the feed rate to the ball mill. After the weightometer, the required grinding media will be added to the ball mill feed conveyor. The material on the ball mill feed conveyor, cyclone cluster underflow, and dilution water, will then be fed into the ball mill.

17.3.2 Classification and solid liquid separation

Pulp will be discharged from the ball mill via a trommel and gravitate into a cyclone feed pump box. The oversize material discharged by the ball mill trommel will fall by gravity to a scats bunker and periodically removed.

The mill discharge slurry fed into the cyclone feed pump box will be diluted with process water to achieve the correct density for the cyclone feed, which will then be pumped to the cyclone cluster. The cyclone cluster underflow will gravitate to the ball mill for further size reduction. The cyclone cluster overflow will report to the mill thickener feed box. The feed box will be fed from the overflow from the cyclone cluster, as well as flocculant to aid in the settling of the solids.

As part of the requirement to minimise water usage, the overflow from the mill thickener will be pumped to the cyclone feed pump box.

17.4 Extraction

Manganese is associated with higher oxides of manganese such as cryptomelane, hollandite, and bixbyite in which the manganese occurs as either manganese oxide (Mn[III]) or manganese dioxide (Mn[IV]). The manganese must be reduced to manganese oxide to be extracted as a soluble sulphate; this is achieved in an acid sulphate medium at a pH of 1.0 to 1.5, an elevated temperature of 90°C using sulphur dioxide as a reductant (e.g., extraction of manganese from cryptomelane; Equation 17.1).

$$2KMn_8O_{16} + 15SO_2 + 2H_2SO_4 = 16MnSO_4 + K_2SO_4 + 2H_2O \qquad \text{Equation 17.1}$$

At the same time, other base metals such as iron, aluminium, zinc, vanadium, nickel, copper, cobalt, chromium, and arsenic will be partially co-extracted as sulphates during leaching.

17.4.1 Leaching

The underflow from the mill thickener will be pumped to a leach feed tank. The leach feed tank will act as a surge tank to smooth out the amount of feed coming from the mill thickener to the leach tanks. Sulphuric acid will be added to the leach feed tank and the slurry will be heated to 90°. The slurry will then be pumped to the first of four leach tanks. Each of the leach tanks will have sulphur dioxide gas injected into the slurry to maintain the redox potential (E_h) of the slurry at 600 mV to ensure the reduction and leaching of the manganese in the mineralized material. The slurry will flow by gravity, cascading through each of the leach tanks. The discharge from the fourth leach tank will flow by gravity into the leach discharge pump box and will be pumped to the first potassium precipitation tank in the purification area.

17.4.2 Purification

Purification will be undertaken in several stages. The first stage will be a jarosite precipitation process in which potassium and sodium impurities will be removed at an elevated temperature of 90°C and a low pH of 1.0 to 1.5 using ferric iron in solution together with supplementary ferric iron as required. Generally (Equation 17.2):

$$3Fe^{3+} + M^{+} + 2SO_4^{2-} + 6H_2O \rightarrow MFe_3(SO_4)_2(OH)_6 + 6H^{+} \qquad \text{Equation 17.2}$$

Where M is a monovalent cation such as potassium or sodium.

The second stage will be neutralisation, where other metals such as residual iron and aluminium (Al) will be precipitated out at higher pH levels. The leach residue slurry will be neutralised in two stages using limestone ($CaCO_3$) and lime (CaOH): first to an approximate pH of 4 using limestone, then to a pH of 5 to 5.5 using lime, in which iron and aluminium will be precipitated as the hydroxides and other base metals partially co-precipitated as hydroxides (e.g., Equations 17.3 and 17.4).

$$Fe_2(SO_4)_3 + 3CaCO_3 + 9H_2O = 2Fe(OH)_3 + 3CaSO_4.2H_2O + 3CO_2 \qquad \text{Equation 17.3}$$

$$Al_2(SO_4)_3 + 3Ca(OH)_2 + 6H_2O = 2Al(OH)_3 + 3CaSO_4.2H_2O \qquad \text{Equation 17.4}$$

The residue will be a mixed precipitate comprising primarily ferric and aluminium hydroxides, gypsum, Epsom salt, and minerals such as silica and micas. Arsenic will be removed as scorodite. The solids in the slurry will be removed and the solution will move to the base metal precipitation section within the purification area. The precipitate will then be filtered and washed; the solids residue constitutes the tails product, and the solution will move downstream to the next purification step. Residual base metals in the solution will be removed by sulphide precipitation using sodium hydrosulphide (e.g., Equation 17.5).

$$ZnSO_4 + NaHS = ZnS + NaHSO_4 \qquad \text{Equation 17.5}$$

The mixed sulphide by-product will be filtered and washed, and the solution will proceed to the next purification step.

17.4.3 Jarosite precipitation

The leached slurry will be pumped from the extraction area into the first potassium precipitation tank. In the first potassium precipitation tank, ferric sulphate ($Fe_2[SO_4]_3$) will be added to precipitate jarosite. A pH of 1.5 in the jarosite precipitation tanks will be maintained using lime and manganese hydroxide returned from the crystallisation area. The discharge from the fourth potassium precipitation tank will flow by gravity into the first precipitation tank.

17.4.4 Iron and aluminium precipitation

The discharge from the fourth potassium precipitation tank will flow by gravity into the first precipitation tank in the purification area, along with the discharge from the scrubber in the processing plant infrastructure area and limestone for increasing the pH in the first precipitation tank. The carbon dioxide (CO_2) generated from the neutralisation reaction will be vented off. The slurry will then be pumped to the second precipitation tank, where lime and air are added to increase the pH further, oxidise the remaining ferrous iron, and precipitate the iron and aluminium as their insoluble hydroxides. The slurry will then be pumped into the filter press feed tank. The filter press feed tank will act as a buffer for batch filtration, which will use plate and frame filter presses. The slurry will then be pumped into the plate and frame filter presses, which will remove the solids from the slurry, leaving a filter cake that will be conveyed to the tailings handling section. The filtrate will then be collected in the filtrate tank before being pumped to the base metal precipitation tank.

17.4.5 Base metal precipitation

Sodium hydrosulphide will be added to the base metal precipitation tank to precipitate residual base metals. The slurry will then be pumped to the base metal thickener, where the solids will be densified. The thickened underflow slurry from the base metal thickener will be pumped into the base metal filter press feed tank. The base metal filter press feed tank will act as a buffer for batch filtration using plate and frame base metal filter presses. The slurry will then be pumped into the plate and frame base metal filter presses, which will remove the solids from the slurry, leaving a filter cake that will be conveyed to a bulk bag for storage. The filtrate will be collected in the base metal filtrate tank before being pumped back into the base metal thickener. The overflow from the base metal thickener will flow by gravity into the first hydrofluoric acid polishing tank.

17.5 Fluoride polishing

Calcium and magnesium will be removed by fluoride polishing, using hydrofluoric acid and barium hydroxide for pH adjustment. Calcium will be almost completely removed, magnesium will be substantially removed as insoluble fluorides, and the barium used in pH adjustment will be precipitated as an insoluble sulphate (Equation 17.6).

$$CaSO_4 + 2HF + Ba(OH)_2 = CaF_2 + BaSO_4 + 2H_2O \qquad \text{Equation 17.6}$$

The precipitate will be concentrated and removed by filtration, and the solution will be transferred to fluoride removal. Residual fluoride will be removed conventionally using activated alumina in columns. The columns will operate as a carousel, with one of the columns having the alumina stripped of fluoride and regenerated. This is described in further detail using the method described in Sections 17.5.1 and 17.5.2.

17.5.1 Calcium and magnesium precipitation

The overflow from the base metal thickener will flow by gravity into the first hydrofluoric acid polishing tank, along with slurry seed material from the hydrofluoric acid polishing thickener underflow, where hydrofluoric acid and barium hydroxide are added to facilitate the precipitation of calcium and magnesium. The slurry will then flow by gravity from the first hydrofluoric acid polishing tank into the second hydrofluoric acid polishing tank. The slurry will then be pumped from the second hydrofluoric acid polishing tank into the hydrofluoric acid polishing thickener, together with the filtrate from the hydrofluoric acid polishing filtrate tank. The overflow from the hydrofluoric acid polishing thickener will then flow by gravity into the alumina column feed tank. The underflow from the hydrofluoric acid polishing thickener will then be pumped and the slurry stream will split between the return to the first hydrofluoric acid polishing tank and the hydrofluoric acid polishing filter feed tank. The hydrofluoric acid polishing filter press feed tank will act as a buffer for batch filtration, which uses a plate and frame hydrofluoric acid polishing filter press. The slurry will then be pumped into the plate and frame hydrofluoric acid polishing filter press, which removes the solids from the slurry, leaving a filter cake that will be conveyed to tails handling. The filtrate will then discharge into the hydrofluoric acid polishing filtrate tank. The filtrate collected in the hydrofluoric acid polishing filtrate tank will then be pumped back into the hydrofluoric acid polishing thickener.

17.5.2 Fluoride removal

The alumina column feed tank will act as a buffer for the batch removal of fluoride in the alumina adsorption columns. Seven alumina columns will be operated in carousel fashion: six will operate in series, with a seventh column used for stripping of fluoride and regenerating the alumina. The solution that has been contacted with the activated alumina will then then be pumped into the crystalliser feed tank.

After feeding the solution through the six columns for 4 hours, the feed arrangement will change. The column that was being regenerated will become the first column in the sequence, what was the first column will become the second, the second will become the third in the series, and so on. The last column in the sequence will start the regeneration sequence for the column that was number six in the sequence.

The regeneration sequence will start by draining the column and pumping the solution back into the alumina column feed tank. The alumina will then be washed with a caustic solution to strip the fluoride from the alumina. Thereafter, the alumina will then be washed to remove any residual caustic solution, and the solution will be contacted with a dilute sulphuric acid solution to complete the regeneration step. The spent solution will be pumped to the water treatment area for treatment.

The dilute acid will be made by adding demineralised water to the dilute acid tank and then adding the required amount of concentrated sulphuric acid to the tank. Similarly, the diluted caustic solution will be made in the regenerant tank by adding demineralised water to the regenerant tank and adding the concentrated caustic solution to the tank.

17.6 Crystallisation

The preceding purification scheme will result in a high purity stock solution of manganese sulphate; however, it will still contain some minor impurities, notably potassium, sodium, manganese, and residual fluoride. It is anticipated that these constituents will be further removed during crystallisation, resulting in an HPMSM product. The bleed stream from the crude crystalliser will be fed into a manganese hydroxide precipitation circuit for the recovery of manganese oxide from the bleed stream.

The manganese hydroxide precipitation will be undertaken at an elevated pH of 8, to precipitate the manganese as a hydroxide (e.g., Equation 17.7).

$$MnSO_4 + Ca(OH)_2 + 2H_2O = Mn(OH)_2 + CaSO_4.2H_2O \qquad \text{Equation 17.7}$$

The precipitate will be separated and repulped using demineralised water and the pulp transferred back into the process.

17.6.1 Manganese precipitation

The bleed stream from the crystalliser will be fed into the first manganese precipitation tank, along with lime to increase the pH to 8, to promote the precipitation of manganese hydroxide. The solution will then flow by gravity into the second and third manganese precipitation tanks, where lime will be added to maintain a pH of 8. The slurry will then be pumped to the manganese filter feed tank. The filter press feed tank will act as a buffer for batch filtration using plate and frame filter presses. The slurry will be pumped into a plate and frame filter press, which will remove the solids from the slurry, leaving a filter cake that will be conveyed to a manganese hydroxide make-up tank. The filtrate will then be collected in the dust suppression tank, along with bleed from the water treatment, blowdown from cooling water, and the waste stream from the scrubber. The solution in the dust suppression tank will then be used as haul road dust suppression or sent to the TMF and removed from the process.

The manganese hydroxide make-up tank will be fed with filter cake from the manganese hydroxide filter press and diluted down to pump the slurry back into the process. The slurry from the manganese hydroxide make-up will be pumped into the manganese hydroxide dosing tank, which will act as a buffer to smooth out to the batch nature of the filtration. The manganese hydroxide dosing pumps will pump the slurry to jarosite precipitation to be used for pH adjustment and return the impure manganese into the process.

17.7 Product handling

The product handling area will consist of a rotary drier for the removal of moisture from the product and a packaging plant; this will be integral to the crystalliser plant. Section 17.7.1 describes the anticipated process and may not represent the final configuration.

17.7.1 Drying and packaging (vendor package)

Material from the crystallisation plant will be fed into the product rotary drier using the product enclosed screw conveyor, to ensure that there is no external contamination (e.g., dust) and to ensure that very little product is lost off the conveyor. The product rotary drier will heat the product, driving off the free moisture. The product will discharge onto a product chain feeder. The vent from the product rotary drier will flow through a product cyclone to recover larger particles that are entrained in the vent gases. The underflow from the product cyclone will discharge onto the product chain feeder. The overflow from the product cyclone will then go to a product baghouse to recover any of the fine particles that are entrained in the product rotary drier vent gas. The recovered product from the product bag house will discharges onto the product chain feeder. To ensure that there will be sufficient differential pressure across the product bag house, a product bag hose blower will be installed on the discharge. The product bag hose blower will blow the exhaust gasses to the stack.

The dry product from the product rotary drier, the underflow from the product cyclone, and the solids recovered from the product baghouse will be fed onto the product chain feeder, which will transport the material into the product silo. The product silo provides buffer capacity during filling of the product drums. Product in the product silo will be removed from the bottom of the silo using a drum loader to fill the product drums with a dry product. The product drums will then be moved to an enclosed storage area.

17.8 Lime preparation

The lime preparation consists of two lime preparation sections: limestone and slaked lime. Limestone will be milled to produce a product P_{80} of 80% less than 75 µm and dewatered to ensure that the minimum amount of water is added when adding limestone to the process.

The burnt lime is slaked, an exothermic reaction, in the lime slaker before being used in the process. The slaking (hydration) reaction is as shown in Equation 17.8.

$$2CaO + H_2O = 2Ca(OH)_2 \qquad \text{Equation 17.8}$$

The annual consumption of lime and limestone for the processing plant is shown in Table 17.2.

Table 17.2 Annual lime and limestone consumption rates (200 kt/a processing capacity)

Consumable	Annual consumption (t)
Lime	80,590
Limestone	38,380

17.8.1 Limestone preparation

The description in this section is the anticipated process and may not represent the final installation.

Limestone will be delivered by truck and offloaded into a limestone bunker. The limestone bunker will be equipped with a discharge gate to control the limestone feed onto a limestone discharge conveyor, which will feed onto a limestone feed conveyor. The limestone ball mill will be fed by fresh limestone from the limestone feed conveyor, the underflow from the limestone cyclone, water from the process water tank, and steel balls from the lime ball mill ball charger, when required. The discharge from the limestone ball mill will discharge into a limestone cyclone feed pump box where it will be diluted with return water from the limestone dewatering cyclone and additional dilution water from the process water tank to meet the required limestone cyclone feed density. The limestone slurry will then be pumped to the limestone cyclone for size classification, where the oversized limestone returns to the limestone ball mill via the limestone cyclone underflow. The overflow from the limestone cyclone will flow by gravity into a limestone dewatering pump box. The limestone slurry will then be pumped to the limestone dewatering cyclone for dewatering. The thickened limestone dewatering cyclone underflow will feed into a limestone stock tank, and the overflow will flow back into the limestone cyclone feed pump box. The limestone slurry in the limestone stock tank will be pumped to the dosing points in the processing plant with a ring main, returning to the limestone stock tank.

17.8.2 Slaked lime preparation

The description in this section is the anticipated process and may not represent the final installation.

Burnt or quick lime will be delivered by a lime truck and offloaded using a lime unloading blower to transfer the lime to the lime storage silo. The lime will then be drawn down from the lime storage silo using a screw conveyor to feed lime into an agitated lime slaker tank. The lime slaker tank will be fed with treated water from the water treatment area and burnt lime from a lime screw conveyor to achieve the required concentration. After the required hydration time, the slaked lime slurry will be ready for transfer to the slaked lime storage tank. The slurry will be agitated to maintain the suspension of the slaked lime storage tank. Slaked lime will be pumped to the slurry dosing points in the processing plant via a ring main, which returns the slurry that is not used back to the slaked lime storage tank.

17.9 Water treatment

A water treatment area will supply water services throughout the processing plant. Water services include raw water, potable water, fire suppression, process water, and demineralised water, which are produced in and distributed from the water treatment plant.

The processing plant will be a net consumer of water. Water will enter the system from the following sources:

- ROM material in-situ moisture
- water generated in chemical reactions
- potable water
- water used for steam, cooling water, and scrub liquor makeup
- raw water used to balance the water demand

Some water will leave the processing plant as water contained in the deposited tailings and sewage discharge. The remainder of water returned from various streams will be used on site and mine roads for dust suppression. Water lost from venting and evaporation is not recovered.

17.9.1 Potable water

Potable water will be supplied from the municipal water supply into a potable water tank. The potable water tank ensures that if there is an interruption in the water supply from the municipality, there is potable water available for site use. Water from the potable water tank will be pumped into the safety shower ring main for the supply of water to the safety showers and eyewash stations distributed at strategic locations throughout the processing plant. From the ring main, there will be additional take-off points for drinking water and ablutions.

17.9.2 Raw and fire water

Raw water, supplied from boreholes and local dams, will be fed into a raw water tank. The raw water tank will have sufficient capacity for fire water and processing plant requirements. The processing plant raw water requirements will be supplied via raw water pumps, with the take-off nozzle above the fire water take-off.

Three pumps will be used for the supply of fire water. The firewater jockey pump will be a smaller electrical pump that will maintain pressure in the firewater distribution system. The fire water pump will be another electrical pump that will start if required. The third pump will be the fire water diesel pump that will start if there is no electrical supply and is required.

17.9.3 Process water

The process water tank will be filled with raw water from the raw water tank and condensate from the crystallisation process. Process water will then be distributed from the tank throughout the processing plant.

17.9.4 Demineralisation plant

The description in this section is the anticipated process and may not reflect the final installation.

A vendor package ion exchange unit will be fed by the process water distribution system and spent solution from activated alumina regeneration together with limestone or lime to neutralise any acidic streams. It is anticipated that there will be a filter to remove solids in the streams in the form of either a sand filter or a clarifier. The removal of the dissolved solids and minerals in solution will be carried out using an ion-exchange or a reverse-osmosis process. The demineralised water product from an ion exchange will be distributed to the reagents and crystallisation sections. Bleed and backwash streams will discharge into a dust suppression tank in the crystallisation section.

17.10 Area 2800: reagents

The processing plant will use a variety of reagents to perform different parts of the sequential purification and separation processes. This section summarises the offloading, mixing, and dosing requirements of each of the reagents. The reagents that will be used are:

- sodium hydrosulphide
- hydrofluoric acid
- barium hydroxide
- sulphuric acid
- sulphur dioxide gas
- sodium hydroxide (NaOH; caustic)
- flocculant
- ferric sulphate

17.10.1 Sodium hydrosulphide makeup and dosing

Sodium hydrosulphide will be used in the purification area for the precipitation of base metals.

A bulk bag of sodium hydrosulphide will be lowered onto a sodium hydrosulphide bag splitter, to empty the sodium hydrosulphide into a sodium hydrosulphide mixing tank. Demineralised water will be added to the sodium hydrosulphide mixing tank to achieve the required concentration, and the tank will be agitated to ensure complete dissolution. When ready for transfer, the reagent will be pumped into the sodium hydrosulphide dosing tank, from where it will be pumped to the purification section.

17.10.2 Hydrofluoric acid storage and dosing

Hydrofluoric acid will be used in the fluoride polishing area for the precipitation of calcium and magnesium from solution.

ISO containers containing the hydrofluoric acid will be unloaded using an unloading pump, to pump the acid into the hydrogen fluoride storage tank. When required, the acid will be pumped into the hydrogen fluoride dosing tank. From the hydrogen fluoride dosing tank, the acid will then be pumped to the fluoride polishing area for addition.

17.10.3 Barium hydroxide makeup and dosing

Barium hydroxide will be used in the fluoride polishing area for control of pH during the precipitation of calcium and magnesium from solution.

A bulk bag of barium hydroxide will be lowered onto a barium hydroxide bag splitter, to empty the barium hydroxide into a barium hydroxide mixing tank. Demineralised water will be added to the barium hydroxide mixing tank to achieve the required concentration, and the tank will be agitated to ensure suspension of the solids. When ready for transfer, the reagent will be pumped into an agitated barium hydroxide dosing tank.

The barium hydroxide slurry in the barium hydroxide dosing tank will then then be pumped to the dosing point in the purification section for addition via a ring main, then returned to the barium hydroxide dosing tank.

17.10.4 Sulphuric acid storage and dosing

Sulphuric acid will be used for the extraction of metals from the mineralised material in the extraction area.

Sulphuric acid will be delivered by a tanker and offloaded by pumping the acid into any of four acid storage tanks. One of four acid feed pumps will then be used for dosing of sulphuric acid into a leach tank and a dilute acid tank.

17.10.5 Sulphur dioxide

The description in this section is the anticipated process and may not represent the final installation.

Sulphur dioxide will be used as a reducing agent to reduce the manganese to soluble manganese oxide form in the extraction area.

Sulphur will be conveyed from a stockpile into a sulphur melting tank. The liquid sulphur will then be filtered to remove any solids. The molten sulphur will then be pumped into the sulphur burner, along with clean dry air to produce sulphur dioxide gas. The gas will be cooled using steam, which will then be used in the processing plant, and the cooled sulphur dioxide gas will then be stored in a receiver for use in the leach area, as required.

17.10.6 Sodium hydroxide makeup and dosing

Sodium hydroxide will be used in the fluoride polishing area for the regeneration of the activated alumina.

Bulk bags of sodium hydroxide will be lowered onto a caustic bag splitter to empty the sodium hydroxide into the caustic mixing tank. Demineralised water will be added to the caustic mixing tank to achieve the required concentration, and the tank will be agitated to ensure the mixing and dissolution of the sodium hydroxide. When ready for transfer, the reagent will be pumped into the caustic dosing tank.

The sodium hydroxide solution in the caustic dosing tank will then then be pumped to the fluoride polishing section for addition to the regenerant tank.

17.10.7 Flocculant makeup and dosing

The description in this section is the anticipated process and may not reflect the final installation.

Flocculant will be used to improve solid/liquid separation in various separation units throughout the processing plant to improve the solid/liquid separation and reduce the size of the associated equipment.

Flocculant bags and demineralised water will be added to the flocculant mixing tank. The flocculant will be hydrated and then pumped into a storage tank. The flocculant solution will then be pumped to the separation equipment, as required.

17.10.8 Ferric sulphate makeup and dosing

Ferric sulphate will be used in the purification area for the precipitation of jarosite.

Bulk bags of ferric sulphate will be lowered onto a ferric sulphate bag splitter to empty the ferric sulphate into the ferric sulphate mixing tank. Demineralised water will be added to the ferric sulphate mixing tank to achieve the required concentration, and the tank will be agitated to ensure the mixing and dissolution of the ferric sulphate. When ready for transfer, the reagent will be pumped into the ferric sulphate dosing tank.

The ferric sulphate solution in the ferric sulphate dosing tank will then be pumped to the purification area section for addition to the potassium precipitation tanks.

17.11 Plant infrastructure and utilities

The processing plant area infrastructure will comprise a scrubber, to ensure that the gases released comply with emission standards, and plant utilities (steam, cooling water, and plant and instrument air). Each of these utilities will deliver the required service (steam, cooling water, or compressed air) to where they will be needed in the processing plant.

17.12 Control philosophy

The processing plant will be fully automated. Minimal operator supervision will be required to maintain production requirements and manage the interaction between the production plant, utility and reagent areas, or in the event of equipment breakdown.

The processing plant will generally be controlled in the direction of flow between unit operations.

Control will be “PUSH” or “PULL” based.

A PUSH control is defined as source nodes determining outflow to receiving nodes. An example could include a tank level being controlled by the variation of drive speed on the discharge pump. Fluid is moved downstream without direct influence from the downstream system.

A PULL control is defined as receiving nodes determining inflow from source nodes. An example could include reagent addition to a tank, where the flow of reagent is controlled at the receiving tank and reagent supplies must be maintained upstream to meet demand.

18 PROJECT INFRASTRUCTURE

The planned on-site infrastructure and services for the K.Hill Project includes a road network, processing plant, mine facilities, WRD, TMF, and power supply and distribution. Planned off-site infrastructure includes access roads, high-voltage power lines to the site, solar plant, and water pipelines. Construction of some off-site infrastructure will require relocation of some existing infrastructure.

18.1 Site access and location

The major modes of transportation in Botswana are road, air, and rail. As Botswana is a landlocked country, marine transport, while important in terms of transporting goods to European, Asian, and North American markets, is not directly under the control of the Botswana government.

There are 12 airports with paved runways and approximately 80 airstrips with unpaved runways. Of Botswana's 12 airports, Air Botswana uses 5 airports on their normal routes. Three of those five airports regularly serve international transport needs.

As of 2000, Botswana had close to 20,000 km of highway; approximately 8,700 km are paved and approximately 11,200 km are unpaved or graveled road surfaces.

The K.Hill Project area is serviced by paved national roads, namely the A1 and A2 highways. The Project is located within a few kilometres of the A2 highway, which runs from Buitepos at the Namibian border, through Jwaneng, Kanye, and Lobatse to the South African border at Pioneer Gate, near Zeerust, South Africa. The A2 highway is part of the Trans-Kalahari Corridor, which is a highway corridor that provides a direct route from Maputo in Mozambique via Pretoria to central Namibia, Windhoek, and the Port of Walvis Bay.

The K.Hill Project is accessible from the A2 highway via unpaved roads from Kanye. Old mining tracks are largely overgrown and have been degraded by erosion, and access to some areas is only by foot or four-wheel drive vehicle.

18.2 General site layout

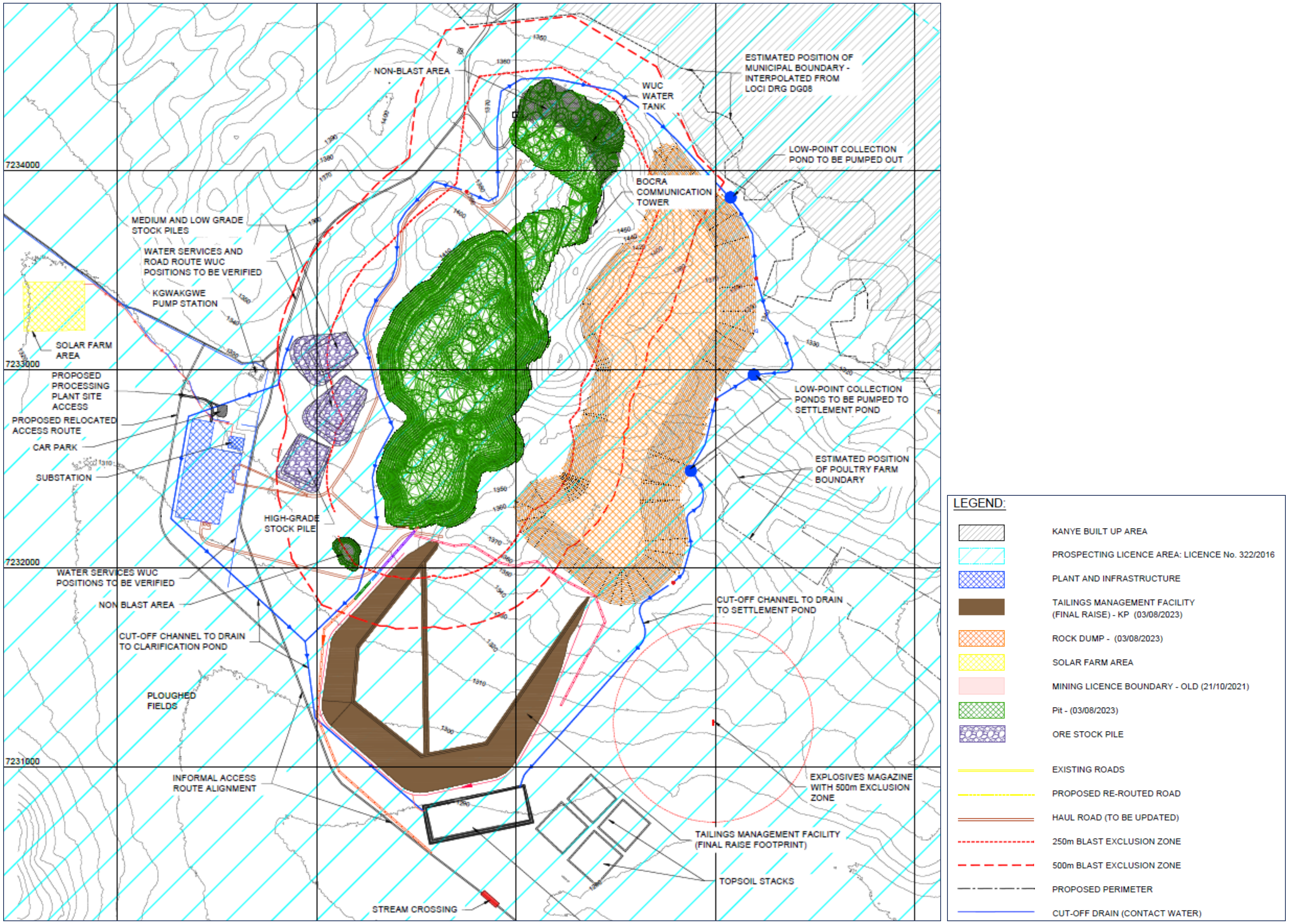
A practical approach was used when designing the site facilities layout, driven largely by the site topography, proximity to Kanye, access to local road structures, and prevailing wind direction.

Based on these factors, the processing plant, on-site infrastructure, and TMF will be located southwest of the pit, a reasonable distance away from the town of Kanye, on relatively flat ground, close to the existing road, and downwind of the town.

Mine site preparation will consist of clearing and grubbing areas to accommodate surface facilities, including an average removal of 100 mm of topsoil. Sites will be levelled and graded only in the areas where construction will take place. Cut-and-fill will be used where large, level areas are required.

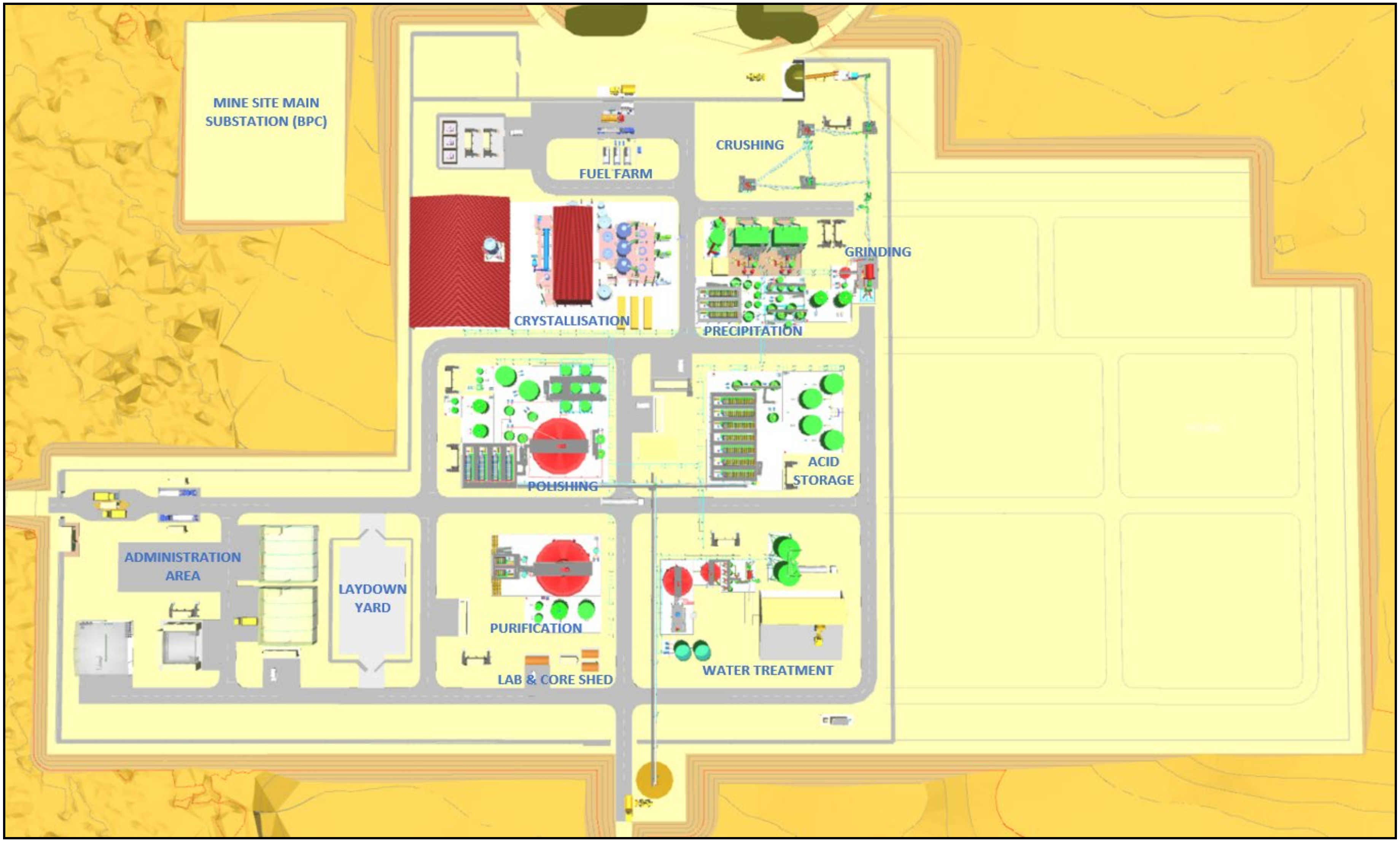
The overall site layout, showing both on-site and off-site infrastructure, is shown on Figure 18.1, and the process plant facility, administration, and maintenance area are shown on Figure 18.2.

Figure 18.1 Overall site layout



Source: Tetra Tech

Figure 18.2 Processing facility, administration, and maintenance area



Source: Tetra Tech

The overall processing plant layout is based on the concepts of maximised operability, maintainability, and general safety. Special design attention has been given to the positioning of process and other equipment in the processing plant layout, to ensure direct and efficient flow of material from the ROM feed point, through the production facility to product storage and dispatch, while minimising conveyor lengths and piping runs.

The processing plant will be located on relatively flat ground, within a short haul distance of the open pit. The TMF will be located down stream of all other mining areas.

Mine infrastructure will be located close to the main access road. The main electrical high-voltage substation will also be located in this area. The solar farm will be located on relatively flat ground, approximately 2 km west of the processing plant along an existing access road. The medium-voltage power line will follow this existing road and tie into the main medium-voltage substation at the processing plant.

18.3 Basis of design

The following subsections provide a description of the rationale, criteria, principles, assumptions, and constraints used in the K.Hill Project site design.

18.3.1 Architectural

The architectural design basis for the site facilities are as follows:

- Modular portacabin-style buildings will be provided for the administration area, with a separate ablution block and a covered outside seating area.
- Modular steel-framed buildings will be provided for all maintenance buildings and the storage warehouses.
- The explosives and detonator stores will be located within bunkered and modular steel framed buildings in line with Section 4 of Botswana Explosives regulations.
- Roll-over metal doors, suitable for vehicle entry, will be provided for the maintenance buildings and storage warehouses.
- External personnel access doors and internal partitions will be provided, as appropriate.
- Liquid storage will be provided by using sloping bunds complete with sumps and pumps.
- Local building materials will be used where possible.

18.3.2 Structural

Structural steel will be modularised, as much as possible, so that construction work on site is minimised. The use of pre-engineered buildings will be maximised to obtain cost-effective optimal designs. It is envisaged that all steelwork structural connections will be bolted.

Operational serviceability and any accidental load cases will be designed using a consistent set of design codes, such as Eurocodes or ISO codes, when different. These dictate the recommended loadings and load combinations, plus consequential allowable stresses and deflections limits.

Typical primary steel will be grade S355 or similar, with metric bolts of grade 8.8 or higher. Typical standard live loads will vary by location from 5 kPa to 15 kPa, with more extreme values considered within the area of the crushers (a minimum of 20 kPa). Typical allowable deflections as a function of member length will vary between 1/90 and 1/800, depending on the application.

18.3.3 Electrical

A high-voltage (33 kV) line from the main Kanye substation will provide electrical power to the K.Hill Project. The line will run from the substation, located north of the town, and run around the town and down the western access road to the main processing plant substation. The BPC will provide the high-voltage line and the main processing plant substation.

18.3.4 Power distribution

Power distribution around the site will be at 11 kV. Cables will run in trenches and trays will deliver power to the process plant, including the mills. Power to areas outside of the processing plant will be delivered via overhead power lines with a cable drop to loads along the line. Electrical rooms will be located close to loads to limit a voltage drop within acceptable tolerances.

The electrical system will be designed using the voltage levels, frequency, and earthing, as listed in Table 18.1.

Table 18.1 Electrical system voltages

Application	Criteria
Medium-voltage level	11 kV, three-phase, 50 Hz, high-resistance grounded
Low-voltage level	400 V, 3+N, 50 Hz high-resistance grounded
Lighting and small power	220 V, 1+N, 50 Hz
Low-voltage motor contactors	220 V, 1+N, 50 Hz
Equipment heaters	220 V, 1+N, 50 Hz
Motor control centre circuits	24 VDC
Plant control system hardware	24 VDC
Electrical field controls	24 VDC
Instrumentation	24 VDC

Notes:
N - neutral
VDC - volt direct current

18.3.5 Mechanical

All process equipment design flow rates are based on the processing plant design feed rates, as stated in the process design criteria and the mass balance (Section 17). All equipment has been selected for compatibility with the process for which they are designed and selected. Equipment material and design has been selected to eliminate corrosion, clogging, and film build up, etc., owing to the process fluids.

The process equipment has been arranged in accordance with current and approved process diagrams.

The design basis for process equipment layout is as follows:

- Gravity and natural properties of material flow have been used, to the maximum extent possible, to reduce energy inputs.
- Arrangements have been designed to provide a smooth process flow and to allow for merging with other process flow streams.
- All material transfer points have been designed to minimise spillage.
- Adequate accessibility and clearance around equipment has been provided for installation, operation, and maintenance.
- Optimal use of the structures and available space within the structures has been included.
- Floors have been suitably sloped and drains and sumps will be provided and positioned at the lowest points to collect spillage and washdown water.

The process equipment selected is robust and fit for heavy-duty applications typically found in a mining environment. All equipment has been conservatively rated and sized to withstand capacity changes owing to process upsets and variations.

All equipment has been selected to meet site conditions, such as altitude, ambient temperatures, seismic activity, wind, rain, humidity, and any locally corrosive atmosphere. The equipment has been selected to meet or exceed the Project-specified production requirements for the expected LOM operation.

Wherever possible, standard "off the shelf" equipment and components have been selected. All equipment has been designed and selected in accordance with the process requirements and site conditions.

As far as possible, equipment has been selected to have a transport weight of less than 10 t per axle or to be able to be broken down into sub-components weighing less than 10 t per axle to meet the transportation requirements to site. Equipment has also been selected to have maximum dimensions of 12 m (length) × 3 m (width) × 2.6 m (height) or to be able to be broken down into sub-components of less 12 m × 3 m × 2.6 m.

18.3.6 Instrumentation, control, and communication systems

Instrumentation and controls will incorporate conventional 4-20 mA analog with highway addressable remote transducer control and 24-volt-direct-current discrete control signaling. Field devices will be connected to field remote input/output panels, which will then connect via industrial Ethernet over single-mode, fibre-optic cable to process control system controller panels located in the electrical room.

The controller panels will contain redundant power supplies and controllers and will connect to redundant control system network core switches and process controller server equipment located in a central control room and adjacent control system server room.

The control system cable network will consist of optical ground wire run on overhead powerlines to off-site locations and conventional fibre cabling distributed throughout the process areas using armoured cable and cable tray. Both modes will be part of the plant-wide integrated fibre backbone network.

Internet communications fibre will be included in a 24-strand optical ground wire cable, to be run with the incoming site power line from the main road to south of the Project.

Industrial Ethernet will be used for control system interfaces with motor starters and variable frequency drives. The central control room will contain three operator human machine interface control stations and two engineering workstations. A remote-control cab provided at the primary crusher will contain a single-operator workstation. Vendor-supplied programmable logic controller control systems will connect to the process control systems via industrial Ethernet fibre cable.

The process control system will be based on a distributed control system platform. This processing plant-wide system will include redundant controller panels, remote input/output panels, human machine interface control stations, peripherals, networks, and complete logic and control screen(s) graphic development.

Processing plant local area network communications racks, including business and process Ethernet network equipment, will be installed in identified electrical rooms and process and office buildings. Fibre distribution panels will be integrated into these racks to provide interconnection of the network switches and dedicated interconnection of various process, business, and fire detection systems. Voice and data systems will be integrated using virtual local area network separation.

Various types of systems will support different operations and business needs:

- Voice over Internet Protocol (VoIP) telephone services and computer networking within buildings
- hand-held radios for remote operations within the processing plant area
- local area network
- wide area network connection to locations outside the processing plant (Internet service)

18.4 Mine infrastructure

18.4.1 Truck shop

The truck shop will consist of a truck wash, tyre store, maintenance bays, warehouse, office, washroom, and truck parking area. The truck shop will be located within the administration area.

The main civil contractor will be responsible for clearing and grubbing earthworks to prepare the pad.

18.4.2 Stockpile

Low-, medium- and high-grade stockpiles will be located northeast of the processing plant and will be managed and operated by the mining contractor. Table 18.2 sets out the minimum stockpile capacity required for each material grade.

The preparation for these stockpiles will consist of clearing and grubbing; topsoil removal; unsuitable soil removal; low-permeable soil placement; and ripping, moisture conditioning, and compacting. Once all organic material, topsoil, and unsuitable material has been removed and any low points have been filled with low-permeability material, the foundation will be shaped to promote adequate drainage toward designated low points.

Table 18.2 Stockpile minimum capacity

Stockpile	Minimum capacity (m^3)
High grade	800,000
Medium grade	600,000
Low grade	400,000

18.4.3 Waste rock dump

The mining contractor will manage and operate the WRD that will be located approximately 500 m east of the open pit as shown in the site layout in Figure 18.1.

The WRD has been designed to accommodate waste rock according to the production and mining schedule. The preparation for the WRD will consist of clearing and grubbing, topsoil removal, unsuitable soil removal, low-permeable soil placement in low-lying areas, and ripping and compacting to create a firm and dense platform to dump the waste rock.

All waste generated from open pit operations will be stored in an environmentally sound, safe, and secure manner in permanent storage facilities. The WRD will be developed systematically from the north end of the dump as this offers the most efficient mine waste haul. Cut-off drainage and collection ponds have been strategically positioned around the WRD to capture all runoff from the WRD and divert it to the settlement pond.

18.4.4 Explosive magazine

An explosives storage facility will need to be constructed on site to meet the annual explosives and initiation product requirements. The explosives store will be constructed to the specification and under the supervision of the explosive's supplier in line with Section 4 of Botswana Explosives Regulations (Government of Botswana 1979). The store is expected to have a separate shed for storing bagged ammonium nitrate, bunkered magazines, office, workshop, and change room facilities.

The explosives store will be contained within a high-security fence and a 500 m exclusion zone. The access will be via boom gate at the entrance that will be manned by security guards. Only authorised personnel will be permitted to enter the explosives storage facility. The components of the storage area will be connected by a maintained graded road.

18.5 Plant infrastructure

The processing plant will be provided with the following utilities:

- water:
  - municipal potable water sourced off site
  - municipal raw water sourced off site
  - process water reclaimed and recycled on site
  - dust suppression water reclaimed from excess process water
  - soft water produced on site for steam production
  - demineralised water produced on site for high-purity process users
- medium pressure steam (approximately 6 bar[g]):
  - generated on site in dedicated boiler units at the sulphur burning plant
- compressed air:
  - high-pressure processing plant air will be reticulated to users
  - a separate supply of dried and filtered air will be provided for instrument use and for any locations that require higher-quality process air
- scrubber:
  - off gases from the process streams will be withdrawn from the vessels and scrubbed such that the vapour is free of pollutants and appropriate for discharge to atmosphere

The processing facility, administration, and maintenance area layout is shown in Figure 18.2.

18.6 Additional on-site infrastructure and services

18.6.1 Laboratory

An assay laboratory will be located near the processing plant and will contain a simple preparation area, a chemical laboratory for standard minerals analysis, and an environmental analytical facility.

Additional laboratory facilities will also be provided to support the mining operation and grade control activities.

18.6.2 Fire station

The fire station will consist of an office, storage facility for all firefighting equipment, and designated parking space for the fire truck. It will be located on the pad to have unimpeded access in all directions.

18.6.3 Medical facility

The medical facility will consist of a reception area, a consulting room, an emergency room, storage facility for medical equipment and medical waste, and pharmaceutical drugs store. The ambulance will have a designated parking space with unimpeded access in all directions.

18.6.4 Warehouse

There will be two warehouses on site: a processing plant warehouse and a product warehouse. There will also be a fenced laydown area.

18.6.5 Workshop

There will be two workshops on site, both located near the administrative area. There will be one workshop for processing plant maintenance and another for heavy- and light-vehicle maintenance.

18.6.6 Site gatehouse

Vehicular access to the site will be controlled to provide security for personnel and property and to manage the risk to the mining and processing operations.

The gatehouse will consist of a four-bay truck parking area, a guard house, multiple weighbridges, and control room and site gate.

18.6.7 Site administration

The site administration building will be located at the southern end of the processing plant. The building will be sized to accommodate on-site administration requirements and will provide offices, clerical space, meeting rooms, a boardroom, a kitchenette, and toilet facilities for the site-based general and administration staff. The building will be constructed from repurposed 40 ft shipping containers, will be insulated with a steel truss roof, and heated and cooled to provide a comfortable working environment. The toilet facilities for the processing plant will be located in the administration building. A septic tank will be buried next to the building and will be emptied as necessary by mobile bowser to take sewage and wastewater to an off-site sewage plant for treatment.

Offices will occupy the first floor of containers, and the lower floor will comprise meeting rooms, boardroom, toilets, kitchenette, and some offices. The open space between the containers will be used as a breakout and dining area.

18.6.8 Fuel farm

The fuel farm will comprise a tank farm, holding 75,000 L of diesel, located on a bunded concrete pad. The bund will be designed to take 110% of the volume of the largest tank on the pad. The tank farm will be made up of three, 20 ft ISO container tanks.

Fuel will be delivered by tanker to a receiving pump station located adjacent to the fuel farm and pumped up into the fuel farm. A facility to fuel site vehicles will be located alongside the pumping station.

18.6.9 Fire detection and protection

Fire protection facilities will incorporate both passive and active systems. Passive systems are features that, by nature of design, resist heat damage, facilitate safe evacuation of people, and aid fire suppression operations. Active systems involve the use of systems and equipment specifically intended to extinguish or control fires, protect people or surrounding property from fire, and warn of a fire emergency. Examples of both types of systems include:

- passive:
  - spatial separation
  - drainage
  - fire separation
  - materials of construction
  - grounding
- active:
  - fire detection (heat/smoke)
  - fire water systems, hoses, hydrants, sprinklers, monitors
  - carbon dioxide gas suppression
  - fire alarms

General design features include the following:

- Smoke detectors and carbon dioxide hand-held fire extinguishers will be installed in all electrical rooms, variable frequency drive rooms, and control rooms. Fire protection for "mission critical" electrical rooms will use clean agent (gaseous) fire suppressant room flooding.
- Electrical rooms will have 2-hour fire separations.
- Hand-held, all-purpose standard ABC fire extinguishers will be provided in all buildings for local emergency firefighting.
- Smoke and heat detectors will be installed in all occupied areas not equipped with sprinklers.
- Duct smoke detectors will be installed in all air-handling units. Once a duct smoke detector is activated, the associated air-handling unit will shut down.

Firewater will be available at facilities and buildings via wet standpipes, sprinklers, and yard hydrants connected to the firewater loop, so that all areas of the facility are within reach of a hose stream. Monitors mounted on hydrants will allow water to be directed to specific hazards, such as the transformers. The firewater loop will be designed so that water can be provided from both directions.

18.7 Solar plant

The proposed location of the K.Hill Project solar plant is 1.2 km west-northwest of the Kayne municipal borehole.

The 4.5 MWpdc solar plant block will cover 7.6 ha, which includes space for an interconnection corridor along the southern edge of the solar plant, with space provision for the central inverter option, alternating current and high-voltage switchgears, communications and weather station, step-up transformer, 11 kV feeder pole interconnection, and clearance path for incoming high-voltage feeders for eventual future expansions, allowing those expansion blocks to share the initial interconnection step-up transformer and main feeder to the mine's 11 kV main electrical bus.

The solar plant location was selected to take advantage of the relatively flat topography west of the mine and to position the solar plant outside of any dust plume coming from K.Hill open pit (dominant wind direction and speeds were accounted for when selecting the location).

This location is also served by a pre-existing road, facilitating installation of the solar plant and the 11 kV feeder routing to the mine. The ground cover of the solar plant is 49.4%, to prevent unacceptable shading losses caused by rows of solar panels casing shadows on each other.

18.8 Tailings management facility

The TMF will consist of a fully lined tailings storage area, to target zero discharge of process-affected fluids, and a settlement pond, in which surface runoff and seepage water will be collected.

The TMF will be located southeast of the processing plant. The final stage of processing uses a filter press, which will produce highly thickened tailings. Based on a reported solids content of 63%, Knight Piésold has assumed that the filtered tailings will form a highly thickened material, such that a "dry stack" facility is not considered feasible, with a geotechnical moisture content (59%) that significantly exceeds its liquid limit and optimum moisture content. Therefore, a conventional impoundment has been adopted for the PEA.

Tailings material will leave the processing plant in the form of highly thickened tailings at an estimated 60% to 70% solids ratio by mass. The tailings will be trucked in tankers to the filtered TMF.

Knight Piésold developed the TMF design criteria, presented in Table 18.4, using data provided by Giyani, including completed historical technical studies. Where no information was available, assumptions were made based on industry standards and Knight Piésold's experience. A TMF plan view is included in Figure 18.3.

Table 18.3 Tailings management facility design criteria

Description	Unit	Criteria	Source
Total LOM throughput	Mt	23.7	Tetra Tech
LOP	years	57	Axe Valley Mining
Annual production rate	t/a	416,880	Tetra Tech
Total required capacity	Mm^3	16.5	Calculated
Tailings transportation method	-	Trucked in tankers	Knight Piésold design
Tailings deposition method	-	Direct discharge into impoundment	Knight Piésold design
Tailings density	t/m^3	1.44	Adopted from SRK and Tetra Tech (2022)
Crest width	m	8	Knight Piésold design
Embankment slope	-	1V:3H	Knight Piésold design
Tailings storage method	-	Within waste rock impoundment	Knight Piésold design

Notes:
V - vertical
H - horizontal

An initial starter embankment (Figure 18.3) has been allowed for the initial 5-year capacity, followed by downstream raises during operations to increase the capacity to final extents as shown on Figure 18.4.

Figure 18.3 Tailings management facility plan view at end of Year 5

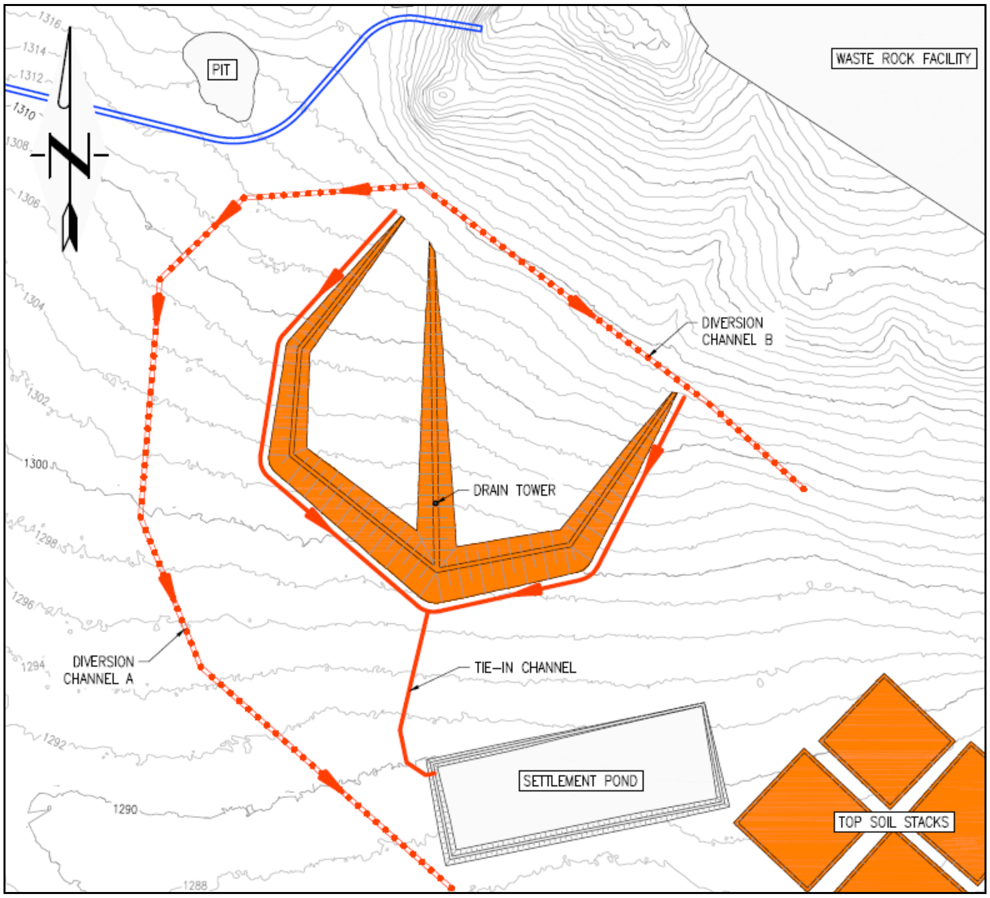
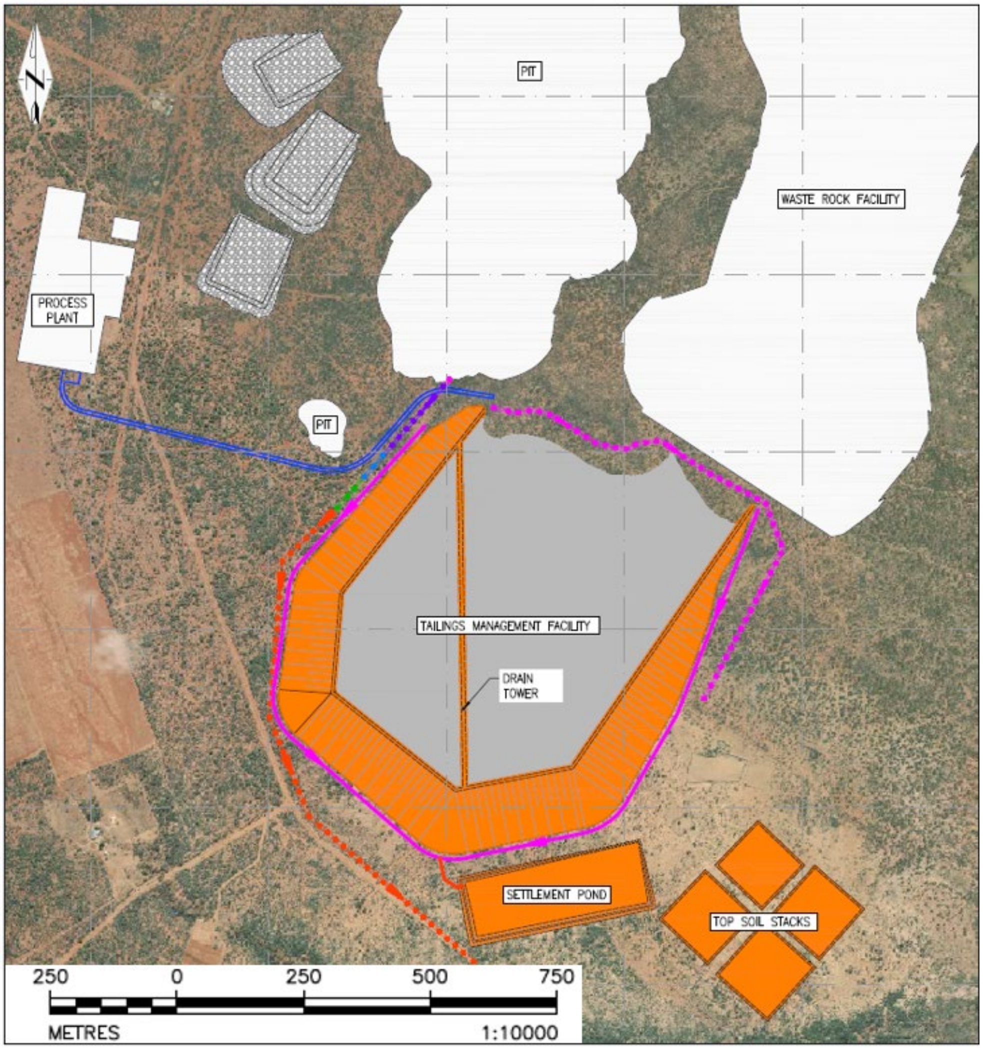


Figure 18.4 Tailings management facility plan view at end of the life of project



The TMF embankment will be constructed in five TMF phases using the downstream construction method. TMF embankment phases 1 to 3 will consist of 10 m lifts, TMF embankment Phase 4 will be a 7 m lift, and TMF embankment Phase 5 will be a 3 m lift. The embankment will be constructed of pre-strip and mine waste rock. An intercell bund will be constructed to facilitate tailings deposition and access to a seepage water drain tower.

The TMF will be a fully lined internally to prevent the release of seepage into the environment. All contact water will be collected and returned to the processing plant for reuse or treatment prior to discharge.

18.8.1 Operating methodology

Tailings will be delivered to the tailings storage area in the form of highly thickened tailings at a 60% to 70% solids ratio. A tanker will transport the tailings from the processing plant and the tailings will be deposited into the paddock facility along the perimeter embankment and intercell bund.

As the thickened tailings has a higher angle of repose than conventional tailings, depositing from the intercell bund will increase the operational flexibility of the TMF by allowing tailings to be deposited from the inside of the paddock.

Dust control by a water truck may be required in the driest months.

18.8.2 Closure planning

The TMF will be closed by installing a suitable cover that will be vegetated to minimise water ingress and erosion, while decreasing the potential for dust generation once the final raise has been constructed.

18.9 Site water management systems

Management of water, within the boundaries of the Project's site surface infrastructure footprint, is based around current industry best management practices to:

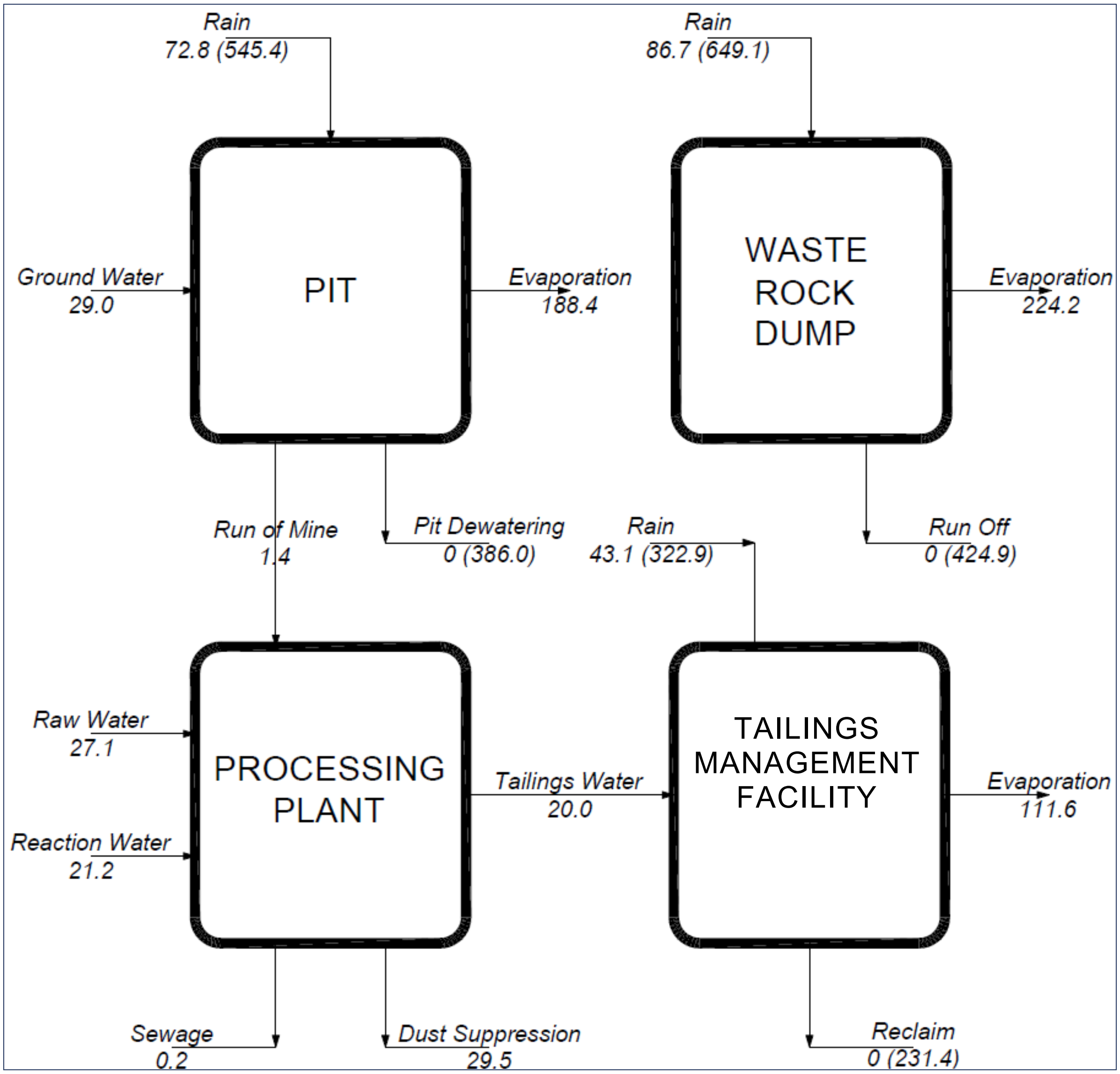
- control surface water runoff to prevent pollution of clean or non-impacted water resources
- control erosion of the site to limit sediment runoff that may impact receiving waters

Water will be collected at various points on the mine site and classified based on the infrastructure or surface from which it has made contact. All contact water will be diverted via ditches to the settlement pond that will be pumped to the water treatment plant (WTP) at the processing plant facility. The average annual evaporation is estimated to be 1,345 mm, indicating a significant deficit between rainfall and evaporation losses all year around.

Total precipitation at the site was derived from regional stations captured from US National Oceanic and Atmospheric Administration database and compared with the satellite information available from ERA5-Land Model. The average annual precipitation at the site is approximately 520 mm. Over a 24-hour duration, the 2-year return period storm event rainfall depth is estimated to be 64 mm. However, for a 100-year return period storm event, the rainfall depth is estimated to reach 260 mm over a 24-hour duration.

Figure 18.5 is a graphical representation of the site-wide water balance, showing nominal and 2-year storm event values for rainfall and evaporation. This graphical representation demonstrates that during nominal weather patterns, with minimal rainfall and significant evaporation, pit dewatering and water reclaim from the TMF will not be required. However, during the 24-hour duration of the 2-year return period storm event, the significant rainfall will be markedly more than the evaporation rate; therefore, pit dewatering and reclaim from the TMF will be required, as well as diversion of any runoff for containment in the settlement pond.

Figure 18.5 Site-wide water balance for the nominal and 2-year return period storm event values (m³/hour)



Source: Tetra Tech

Notes: Amounts in brackets indicate the 2-year return period storm event value, amounts not in brackets indicate the nominal value.

18.9.1 Water supply

A key component of any mining operation is a reliable supply of water that is available year-round. The K.Hill Project water supply is expected to come from a combination of two sources. The first source comprises water supplied by the WUC, which currently supplies water to Kanye from an existing wellfield and the local Mmamakhasi Dam. The second source comprises abstracting groundwater from a local aquifer via a newly proposed wellfield. A third source, comprising treated water from the Kanye sewage treatment plant, has also been evaluated; however, this option was disregarded due to various challenges.

Giyani has held discussions with the WUC regarding the need to supply the Project with raw water. While the WUC cannot supply the Project with its full water requirement (expected to be approximately 50 m^3/hour), the WUC can supply 20 m^3/hour. Giyani will need to source the balance of raw water via new groundwater supply boreholes, which the mine would need to install. The water supply from the WUC can also not be fully guaranteed, as the WUC is obliged to supply Kanye, rather than the mine, in case of water shortages which might occur during severe droughts.

With the WUC unable to meet the mine's full water demand, the option of sourcing water from the surrounding aquifers was evaluated. Geophysical surveys were conducted to identify preferential target areas, borehole drilling, and 72-hour pump tests to determine the sustainable yields available from a number of different aquifers. The results of the investigation found that boreholes drilled into the Transvaal Dolomitic aquifer are capable of sustainably yielding 50 m^3/hour of water, which could meet the full demand for the mine operations. These boreholes are situated approximately 20 km from the proposed mine area.

A study on the water supply coming from aquifers outside the PL area is currently underway. From the studies undertaken to date for the EIA, no environmental or social red flags/fatal flaws have been identified. It is recommended that Giyani drill a network of monitoring boreholes around the proposed production boreholes, in order to monitor any drawdown interference encountered from the nearby existing WUC production boreholes and vice versa.

Groundwater and stormwater extracted from the open pit can be used for dust suppression and as preferential processing plant raw water to reduce the water demand from other sources. As discussed in Section 16.3.3, groundwater inflows into the excavated main pit are expected to range from approximately 430 m^3/day to 690 m^3/day, while groundwater inflows into the southern satellite pit are expected to range between 40 m^3/day and 50 m^3/day. Surface runoff in the open pit is calculated to be 85,850 m^3/day in the event of a 1:100-year return period storm event.

Potable water will be supplied directly from the municipal water supply into a new potable water tank situated in the processing plant area. The potable water tank will ensure that in the event there is an interruption in water supply from the municipality, potable water will be available for users.

Process water will also be supplied from the municipal water supply to limit the volume of impurities entering the process.

Water supply options for the K.Hill Project are well documented and understood and considered to be appropriate for this level of study.

Water demand for the mining operations, estimated to be 50 m^3/hour, is based on a conceptual plant water balance model developed as part of the previously completed Technical Report (SRK and Tetra Tech 2022). With the proposed expansion of the open pit (based on the updated MRE [Section 14]), it is recommended that Giyani update the existing water balance and re-evaluate the water demand. It is not expected that the updated water demand will differ significantly from the demand calculated as part of the previously completed Technical Report. The groundwater contributions flowing into the open pit (which will need to be pumped out) should also be incorporated into an updated water balance calculation.

18.9.2 Water infrastructure requirements

A number of new water-related pieces of infrastructure will be required for the K.Hill Project, including potable water storage tanks and systems required to treat and distribute processing plant water, fire water, and potable water.

Open pit dewatering

With the updated open pit expected to breach the groundwater level, infrastructure to manage open pit dewatering will be required.

Pit dewatering can be undertaken in-pit using a sump or out-of-pit using dewatering boreholes. Based on the expected relatively low aquifer transmissivity limiting the zone of influence of out-of-pit dewatering wells and the relatively low volume of groundwater inflows expected in the open pit, it is recommended that dewatering take place using an in-pit sump.

It is expected that the water pumped from the open pit will be routed to the processing plant for use. Therefore, it is expected that water pumped from the open pit will be pumped into a pollution control dam/return water dam where it will be absorbed into the water management system and pumped to the processing plant.

Potable water tanks

The village of Kanye receives its water from the Mmamakhasi Dam, situated approximately 5 km from the Project site, which is pumped into two existing water reservoirs that are located adjacent to the open pit area. These reservoirs, and the associated water pipelines, belong to the WUC and are part of the water reticulation for Kanye. Due to the close proximity to the open pit area, the water reservoirs will need to be relocated prior to the start of mining. It is understood that Giyani will engage a local Botswana contractor to relocate the water reservoirs and associated piping.

Fire water and processing plant water management

The processing plant will receive raw water makeup from the local WUC pump station via a new 250 m^3 raw water makeup tank. The proposed raw water tank will have sufficient capacity for fire water and processing plant raw water requirements. The tank will be split into two sections through the placement of suction nozzles; the upper section will be used for raw water and the lower section will be used for fire water.

For the processing plant, a process water tank will be constructed, which will be filled with water from the raw water tank and condensate from the crystallisation process. The process water will then be distributed throughout the processing plant. The following utility streams will be managed in the processing plant:

- raw water provided by the WUC fed into the processing plant raw water tank
- process water reclaimed from the TMF settlement pond
- excess water reclaimed from process water and water treatment effluent used for dust suppression
- soft water produced on site for steam production
- demineralised water produced on site for high-purity process uses

Pumps drawing water from the lowest section of the raw water tank will distribute fire water through a fire water ring main that will run through the processing plant, reagent mixing area, and crystalliser. Three pumps will be used to supply fire water: two electrical pumps and one diesel pump. A fire water jockey pump is a small electrical pump that will be used to maintain pressure of the fire water distribution. In the event of a fire fighting event, the electric fire water pump will engage to supply the required flow of fire water. A diesel pump will serve as a backup pump if there is no electrical supply and fire water is required.

The fire water supply and pump flow will be provided per applicable fire protection codes and standards, and the pump head will be calculated based on the maximum pressure requirements for the farthest hydrant/user.

Water treatment plant

A vendor packaged WTP will be required to supply water services throughout the processing plant. Details surrounding the WTP design and capacity have not been provided at this stage; however, it is envisaged that the WTP will be fed from a number of sources that will contribute toward the process water distribution tank and the spent solution from activated alumina regeneration. The contact water generated on site will also be pumped to the WTP and added into the processing plant water system for general processing.

18.9.3 Water Management Plan

A Water Management Plan for the K.Hill Project has been developed based on current industry best practices, which aim to:

- control surface water runoff to prevent pollution of clean or non-impacted water resources
- control erosion of the Project site to limit sediment runoff that may impact receiving waters

As no perennial watercourses or water bodies are located within the PL area. Surface water management during the LOM will primarily comprise the management of stormwater, with a focus on the separating clean and dirty water streams, preventing contaminated runoff from escaping the site, and recycling water pumped from the open pit.

Separation of clean and dirty water

A Surface Water Management Plan has been developed to mitigate potential surface water risks in order to comply with relevant legislation. As part of the Stormwater Management Plan, potential clean and dirty water areas have been delineated, based on topography and infrastructure plans, and measures to separate these areas have been proposed.

For the purposes of the Surface Water Management Plan, the Project site has been broken up into four dirty water areas: the processing plant, the open pit, the TMF, and the WRD. These four dirty water areas have different drainage characteristics; therefore, separate Stormwater Management Plans have been developed for each area.

All contact water will be diverted via ditches to a new settlement pond constructed at the TMF. The designs are such that the settlement pond will have sufficient capacity to divert the estimated volume of contact water generated in each area. The contact water diverted to the settlement pond will then be pumped to the WTP at the processing plant.

Delineated clean water areas include greenfield areas upstream of the TMF and the WRD. Runoff emanating from these areas will need to be diverted to prevent mixing with potentially dirty water areas.

Processing plant water management

The entire processing plant area has been classified as a dirty water area, owing to its proposed infrastructure and activities that are expected to include a ROM pad, stockpile, manganese extraction using sulphuric acid leach, etc. Therefore, all runoff from the processing plant area, including the stockpile and the processing plant site, will be directed into ditches that will drain to a settlement pond to be constructed at the TMF. Water collected in areas outside of the processing plant, but that may have come into contact with contaminants such as hydrocarbons and industrial chemicals, will also be diverted to settlement pond.

It is understood that the processing plant water treatment has not allowed for the treatment of any return water from the settlement pond at the TMF, as the treatment of impurities could not reliably be estimated at the time. This creates an opportunity to further evaluate the possibility of treating this water to reduce the overall water demand.

Tailings management facility water management

In order to estimate the peak flow of surface runoff from contributing catchments and to size the surface water diversion structures, various storm events need to be taken into consideration.

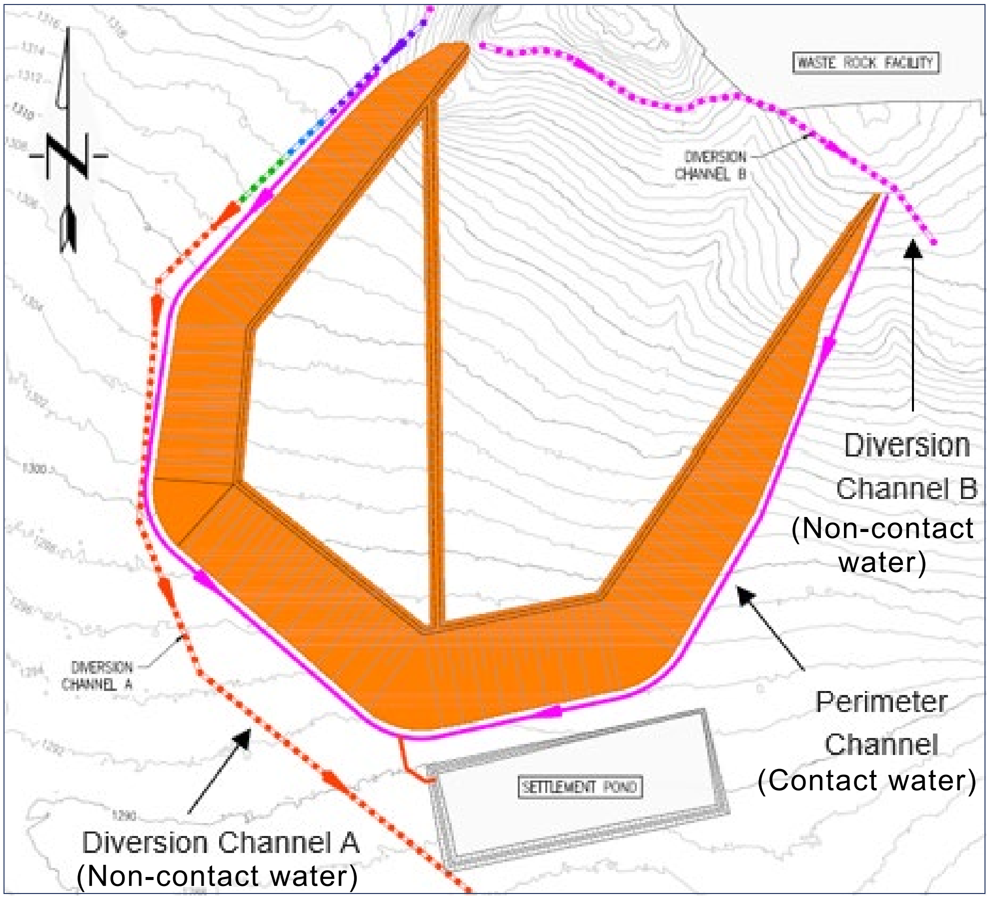
A 1:100-year return period 24-hour storm event was used to estimate the peak flow of surface runoff from contributing catchments and to size the surface water diversion structures.

The preliminary design parameters used for TMF diversion include the following:

- design storm event: 1:100-year return period 24-hour storm event
- rainfall intensity: 260 mm/day (equivalent to 10.83 mm/hour)
- 100-year return period flow rate = 1.7 m^3/second/km^2
- 5-year return period flow rate = 0.63 m^3/second/km^2

Water that comes into contact with the TMF is considered contact water, and water outside of the TMF, such as catchment surface water, is referred to as non-contact water. Diversion channels have been designed to intercept and divert non-contact water before it reaches the TMF, and perimeter channels and a drain tower within the TMF impoundment have been included to manage contact water and direct it to the settlement pond.

Figure 18.6 Location of diversion and perimeter channels



The non-contact water diversion channels will be constructed in Year -1 to 0. A tie-in section of the channel will be developed prior to the commencement of deposition at each stage.

Contact water diverted into the settlement pond will be pumped to the WTP at the processing plant and added into the process water system.

The perimeter contact water channels were designed following the same methodology used for the non-contact water diversion channels. The contributing catchment for the perimeter channels is the downstream face of the waste rock embankments at each stage.

The excavated perimeter channel for TMF Phase 1 will be backfilled and compacted once the perimeter channel for TMF Phase 2 has been constructed. This methodology will continue for all subsequent stages until the final perimeter channel for TMF Phase 5 has been constructed and connected to the settlement pond. A tie-in channel will connect the channels to the settlement pond for pumping back to the processing plant during the intermediate stages.

Water-return infrastructure

A climate and hydrological assessment of the area was completed in the previously completed Technical Report (SRK and Tetra Tech 2022) for the K.Hill Project area, and a resultant flow rate of 1.7 $m^3/s/km^2$ for a 100-year return period was determined. This flow rate was adopted in the sizing of diversion channels, perimeter channels, and the settlement pond.

The required storage volume of the settlement pond is based on contact water being pumped from the vertical drain tower within the TMF basin and contact water from the TMF embankments. An additional 20% of capacity was allowed for as a contingency.

The water-return infrastructure will consist of a static pump system installed at the settlement pond drawing directly from the pond via a suction line and connecting to the processing plant.

Waste rock dump water management

Runoff and seepage emanating from the WRD area have been classified as potentially dirty areas, and as such collection sumps/ponds will be required to settle runoff and prevent excessive suspended solids and elevated salt concentration in runoff water draining toward the Mmamokhasi Dam.

It is assumed that WRD runoff and seepage will follow the topography of the existing terrain, draining in an east-to-southeast direction. Therefore, collection channels are recommended to intercept the drained water along the southern and eastern boundaries of the WRD. This water will be channelled to the low topographic points south and east of the WRD, where a sedimentation pond in each location will control the settling of suspended solids and water quality.

The volume of water expected to emanate from the WRD and daylight as surface water that either ponds near to or flows away from the WRD was calculated using analytical methods based on the available revised WRD layout. Using the final footprint of the WRD and climatic data, and taking into consideration the expected groundwater recharge portion of rain that will fall onto the WRD as well as a runoff factor of 0.55, it was calculated that under a 1:100-year return period 24-hour storm event, approximately 204,000 m^3/day (8,500 m^3/hour) will go toward immediate surface runoff.

A volume of 5,080 m^3/day is expected to emanate from the base of the WRD onto surface, in the form of toe drainage, over a period of time until storage is depleted.

These calculations are based on the following assumptions:

- The calculation is based on the final WRD footprint area; no allowance has been made for expansion over the life of the Project, as this information is not available.
- No rehabilitation (sloping, capping, vegetation) of the WRD will take place over the LOM.
- No backfilling of the open pit using waste rock will take place over the life of the Project.
- The calculated volume of a daily average over the LOM; in reality the seepage volumes will be seasonally dependent, with increasing outflows during the rainy season and decreasing flows during the dry season.

The quality of the water emanating from the WRD is unknown at this stage, and it is recommended Giyani undertake an analysis of the expected water quality, as the final designs of the drainage ponds will depend on the water quality. For example, if the water is found to be “non-contaminated” and fulfills local environmental requirements, it can be released into the natural watercourse; however, if the water is found to be contaminated, it will require treatment (or use if possible) to avoid the impact on downstream environment. The appropriate water management approach (and related infrastructure) should be selected based on the findings of the EIA currently underway, which should consider waste rock geochemistry and related risk of acid rock drainage and metal leaching.

Pit water management

The existing Water Management Plan for the open pit only considers and makes provisions for surface water runoff contributions into the open pit, and does not consider any groundwater inflows, as the open pit area would not intersect any groundwater (SRK and Tetra Tech 2022). However, based on the updated MRE (Section 14) and the updated mining schedule (Section 16), groundwater is likely to be intersected during the LOM, and the Water Management Plan should be updated to include the management of groundwater inflows. As previously discussed in Section 16.2.3, groundwater inflow volumes into the excavated main pit at the end of the LOM are expected to range between 430 m^3/day to 690 m^3/day. Groundwater inflows into the southern satellite pit are expected to range between 40 m^3/day to 50 m^3/day it is recommended that Giyani develop or update a numerical groundwater model to predict the groundwater inflow volumes into the open pit.

Water that is pumped from the open pit should be stored in a water storage facility at the processing plant and should be used for dust suppression and for the processing plant operations to reduce water demand. The water storage facility should be designed such that it has sufficient storage capacity to not only accommodate the volumes being abstracted from the open pit but also to have excess storage available in order to reduce the risk of water shortage for the processing plant in case of breakdown/issue in the water supply system. Such a facility has not yet been designed or accounted for in any of the Water Management Plans or water balances to date.

In terms of managing the volume of stormwater runoff into the pit from the surrounding areas, recommendations have been made in the Water Management Plan to construct berms around the open pit perimeter to direct stormwater away from the pit. However, given that the open pit is situated in the uppermost part of the catchment, the volume of stormwater to be diverted around the open pit is considered to be negligible.

Based on a review of available information, it is evident that preliminary plans are in place to control surface water at the Project site and accompanied by appropriate water management designs. The available information and studies undertaken for water management is considered to be at an appropriate level of confidence for a PEA-level study.

Runoff calculations

Surface water runoff volumes for the open pit, WRD, and stream diversions around the TMF were estimated based on the engineering and mine designs for the previously completed Technical Report (SRK and Tetra Tech 2022). The Rational Method was used in the estimation of peak flows, as this method makes use of drainage catchment and runoff coefficient.

For existing calculations, the following runoff conditions were considered in the estimation to provide a range of potential inflows into the pit:

- 70% runoff coefficient on the open pit wall
- 24-hour, 48-hour, and 72-hour runoff rates after extreme storm event rainfall

Inflows into the pit are likely to be limited to storm events, which usually occur for short durations, given the arid climate of the K.Hill Project area. However, storm runoff can generate high inflows into the open pit in case of rare storm events such as 50- or 100-year return period events. It is noted that water runoff volumes generated by storm events do not have to be pumped out at the same rate that they flow in, as they can be stored in the open pit base up to the capacity of the open pit sump and then pumped out over a period using pumps dedicated to pumping such short-term inflows (the “standby” pumping infrastructure).

As part of the previously completed Technical Report, the anticipated volume of surface water runoff into the open pit was calculated (as detailed in Section 16.3.3) based on the original mining schedule (SRK and Tetra Tech 2022) and prior to the updated MRE (Section 14). The surface water runoff into the open pit was recalculated based on the available updated open pit layout. The open pit extent for the first year of operation is not available; therefore, the surface water runoff in the pit during first year of operations cannot be updated for comparison. However, using the open pit layout at the end of the LOM (Year 50), the calculated surface water runoff into the open pit area is expected to be approximately 85,850 m^3/day in the event of a 1:100-year return period storm event (equivalent to 3,577 m^3/hour). These calculations do not take into consideration possible backfilling of the pit. It should be noted that this calculation was completed to a PEA level of accuracy, and it is recommended that Giyani undertake a full surface runoff calculation as part of further studies.

For the WRD catchments and the diversion around the TMF, the same simple Rational Method was used for runoff estimations. The size of the diversion structures required around the TMF were designed using estimated runoff volumes from a 100-year return period 24-hour storm. To size the water collection ditch around the WRD, 50-year return period storm depths were used.

While preliminary conceptual engineering designs and drawings for the diversion ditches and channels were provided for review, and are deemed to be more than adequate for a PEA-level study, it is noted that all runoff estimations calculated to date are based on the previously completed Technical Report (SRK and Tetra Tech 2022). Analytical calculations of runoff from the WRD (previously discussed starting on page 18-20) and surface runoff into the open pit (Section 16.3.3) have been completed. However, while these numbers meet the requirements of a PEA, it is recommended that Giyani update the calculations to a higher level of confidence in later phases of the Project. If necessary, the conceptual channel designs, in terms of channel locations and sizes, should be updated.

19 MARKET STUDIES AND CONTRACTS

CPM completed a HPMSM product market outlook study in July 2022 for Giyani. CPM is an independent research and consultancy company based in New York and has advised clients on precious and speciality metals markets since 1986.

The study concludes the following:

- The demand for lithium-ion batteries used in electric vehicles is expected to grow by 25% annually between 2021 and 2031 and at a slightly slower rate, around 10% annually, between 2031 and 2041.
- The demand for high-purity manganese (HPMSM and high-purity electrolytic manganese metal [HPEMM]) from the battery sector will grow nearly 30 times between 2021 and 2036, reaching 1.8 Mt on a contained metal basis and may reach 4.5 Mt by 2050.
- Global high-purity manganese production capacity in 2021 was approximately 127 kt/a, and an identified new project pipeline is expected to contribute only an additional 221 kt/a by 2031[1], resulting in a projected supply deficit of 726 kt/a over this period.
- Between 2023 and 2035, the realised price of HPMSM for the K.Hill Project is expected to range from US$2,993 to US$5,499/t (free carrier [FCA] Durban).

19.1 High-purity manganese sulphate monohydrate demand

19.1.1 Battery technology

Manganese is used in the production of batteries in the form of high-purity manganese sulphate solution (HPMSS). Most cathode makers buy HPMSM as a dry crystalline powder and dissolve it to make HPMSS, but some produce it in-house through the metal route by purchasing HPEMM.

Presently, there are two main groups of lithium-ion battery chemistries that use high-purity manganese (either HPMSM or HPEMM) to produce their cathodes: nickel-manganese-cobalt (NMC) and lithium-nickel-manganese oxide (LNMO).

- NMC is the dominant lithium-ion battery chemistry, currently claiming approximately 44% of the rechargeable battery market, and is likely to remain the dominant technology, contributing about 47% to 50% to the market by 2031 and beyond.
- LNMO's use in battery manufacturing is expected to grow rapidly—about 30 times between 2021 and 2031—although it would still account for less than 1% of all manganese-using batteries. LNMO battery chemistry is one of the most manganese-intensive battery chemistries, requiring over one kilogram of manganese per kilowatt hour of battery capacity.

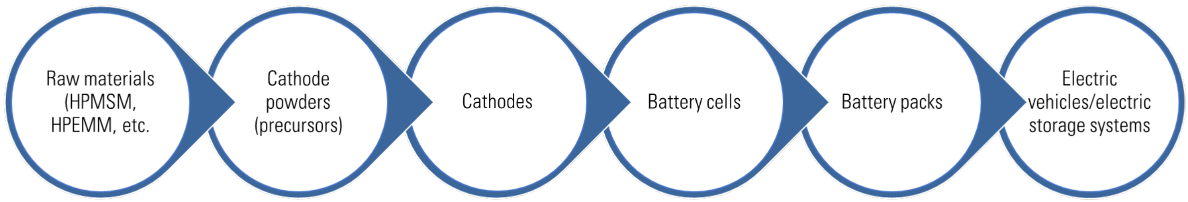
19.1.2 Battery supply chain

The lithium-ion battery industry has its own structure and supply chain with many specialised manufacturers. HPMSM can be sold to different manufacturers, depending on the level of supply chain

[1] Does not include potential 300 kt/a of HPMSM produced from selenium-containing 997 electrolytic manganese metal not meeting the specification of Tier 1 cathode producers.

integration by the various battery and electric vehicle manufacturers; some make just cathode powders or cathodes and others (e.g., Tesla) have many stages of battery production within their manufacturing operations. The ultimate product is a battery pack sold to or assembled by an electric vehicle manufacturer (Figure 19.1).

Figure 19.1 Lithium-ion battery supply chain

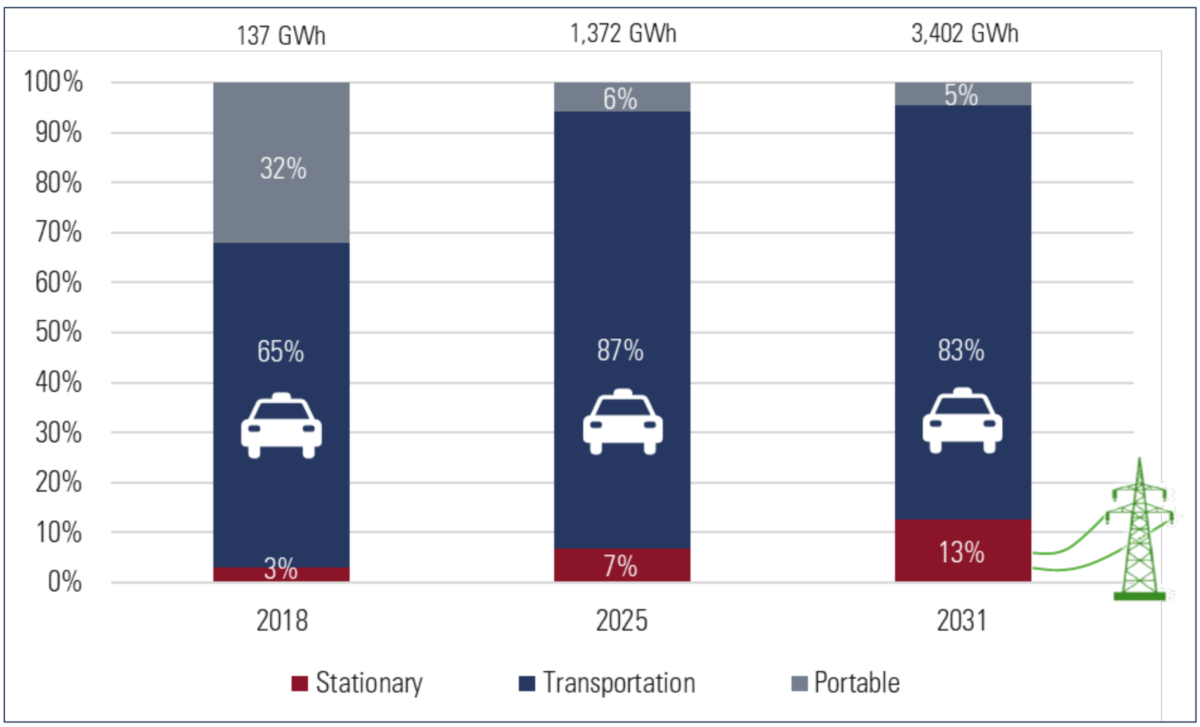


Source: CPM (2022)

19.1.3 Demand for batteries

CPM focused its high-purity manganese sulphate market analysis on its use in the cathodes of rechargeable (secondary) batteries, specifically from electric vehicles. The demand for batteries used in electric vehicles is expected to grow at a compound annual growth rate (CAGR) of 25% between 2021 and 2031 and at a slightly slower rate (around 10% CAGR) between 2031 and 2041 (Figure 19.2).

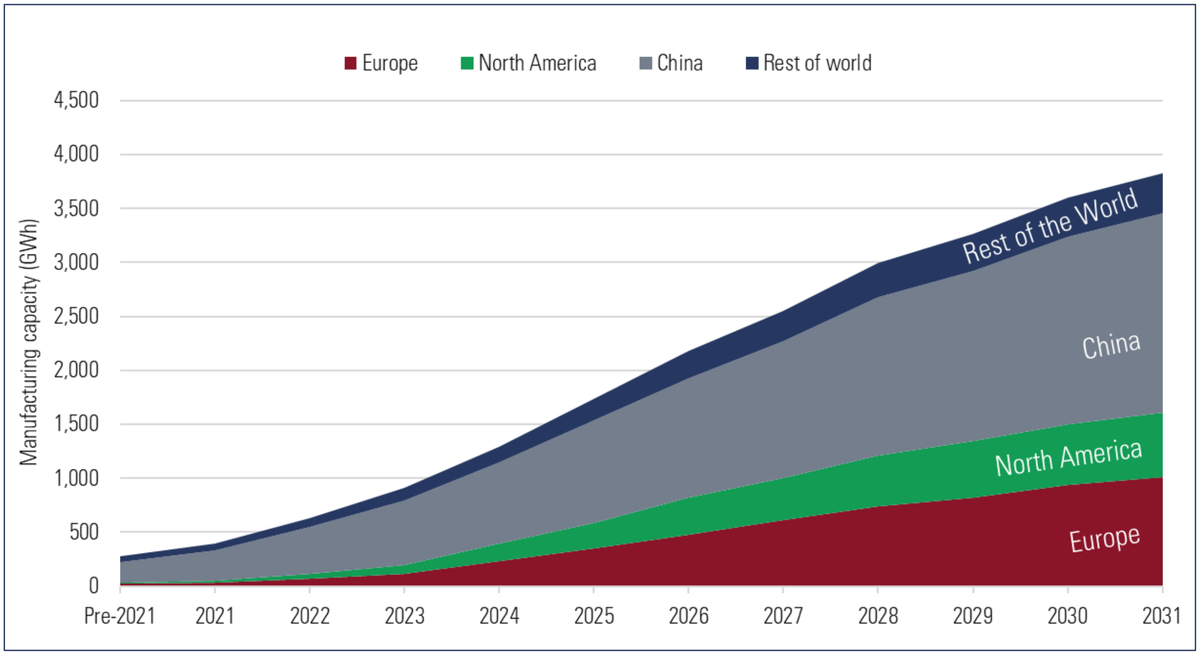
Figure 19.2 Lithium-ion batteries by end use (2018, 2025, 2031)



Source: CPM (2022)

Another method of calculating the demand for battery raw materials is to add up the capacity of the present and future (announced) battery factories who are the clients for the cathode active materials producers. CPM's high-purity manganese demand forecast is based on E Source's battery demand forecast figure of 3.4 TWh in 2031 and, in the context of battery factory capacity, can be considered conservative (Figure 19.3).

Figure 19.3 Battery factories: cumulative effective manufacturing capacity by region to 2031



Source: CPM (2022)

19.1.4 Demand for manganese in lithium-ion batteries

Manganese-based chemistries are likely to dominate the rechargeable battery market over the next 10 to 20 years, partly owing to cobalt supply chain problems and partly because of the technical merits of manganese as a battery metal. CPM, and many battery experts like E Source, expect the demand for high-purity manganese from the battery sector to grow nearly 30 times between 2021 and 2036, reaching 1.8 Mt. The global production would need to rise 15 times to satisfy this demand.

If the battery chemistry mix remains unchanged after 2035 and the demand for batteries grows between 6% and 11% per year, by 2050 the total demand for high-purity manganese from the battery industry could reach 4.5 Mt, compared to 0.13 Mt in 2021. The use of manganese in lithium-iron-phosphate batteries can potentially boost the demand for high-purity manganese by an additional 35% to 70%, according to CPM's calculations; however, these lithium-manganese-iron-phosphate batteries are not yet in commercial circulation.

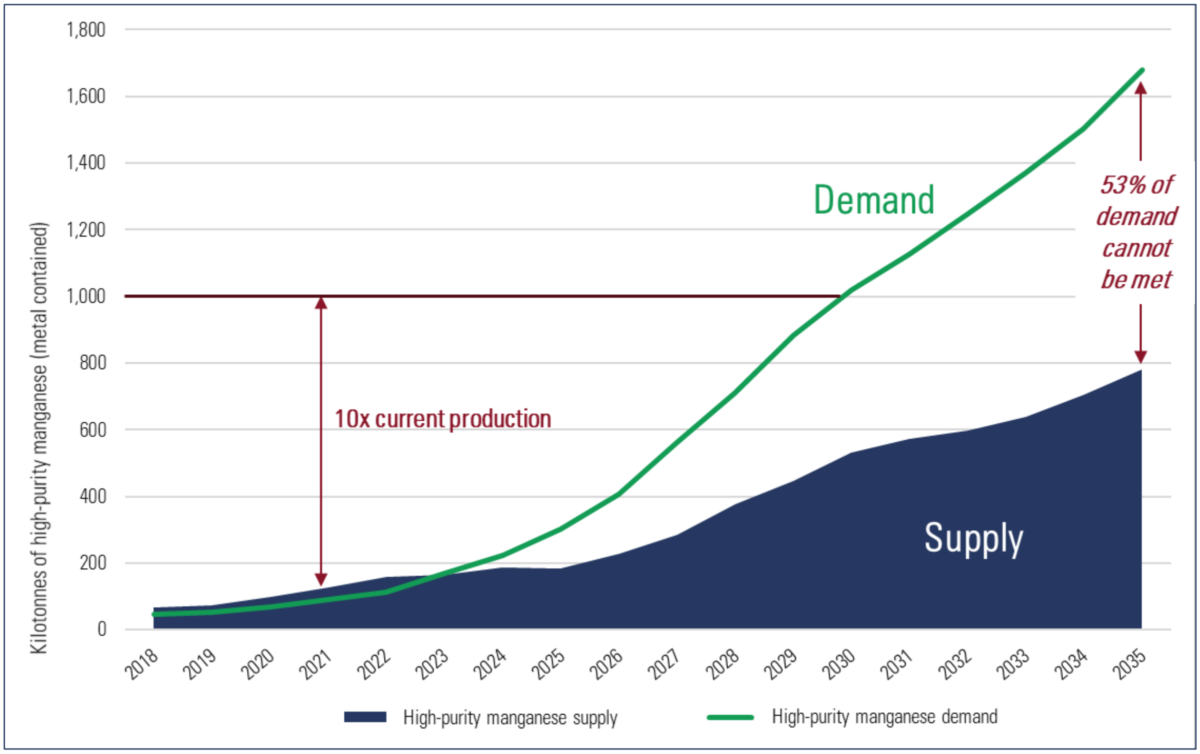
19.2 High-purity manganese sulphate monohydrate supply-demand balance

CPM's assessment of the global high-purity manganese industry indicates that there are only six non-Chinese high-purity manganese projects likely to come on stream before 2031, producing 221 kt/a of new supply of high-purity manganese. An additional 100 kt/a (metal units) may be coming from China by 2025, but there are doubts about the purity of this material because of the use of selenium-containing electrolytic manganese metal as a feedstock. CPM also considered new high-purity manganese supply from the recycling of spent electric vehicle batteries. Assuming a 50%

recycling rate and 100% manganese recovery (unlikely), this supply stream could satisfy up to 6% of 2031 high-purity manganese demand.

Combined with the current declared (but not fully utilised) production capacity of up to 180 kt/a, the total capacity available in 2031 will be 401 kt/a of metal contained. Meanwhile, 2031 projected high-purity manganese demand from the battery sector alone stands at 1,094 kt/a (1,127 kt/a when metallurgical uses are included). This creates a supply deficit of 726 kt. Correcting for nascent projects and recycling, the 2031 deficit comes down from 726 kt to 475 kt. The global supply-demand balance as projected by CPM is shown in Figure 19.4.

Figure 19.4 High-purity manganese demand to 2035

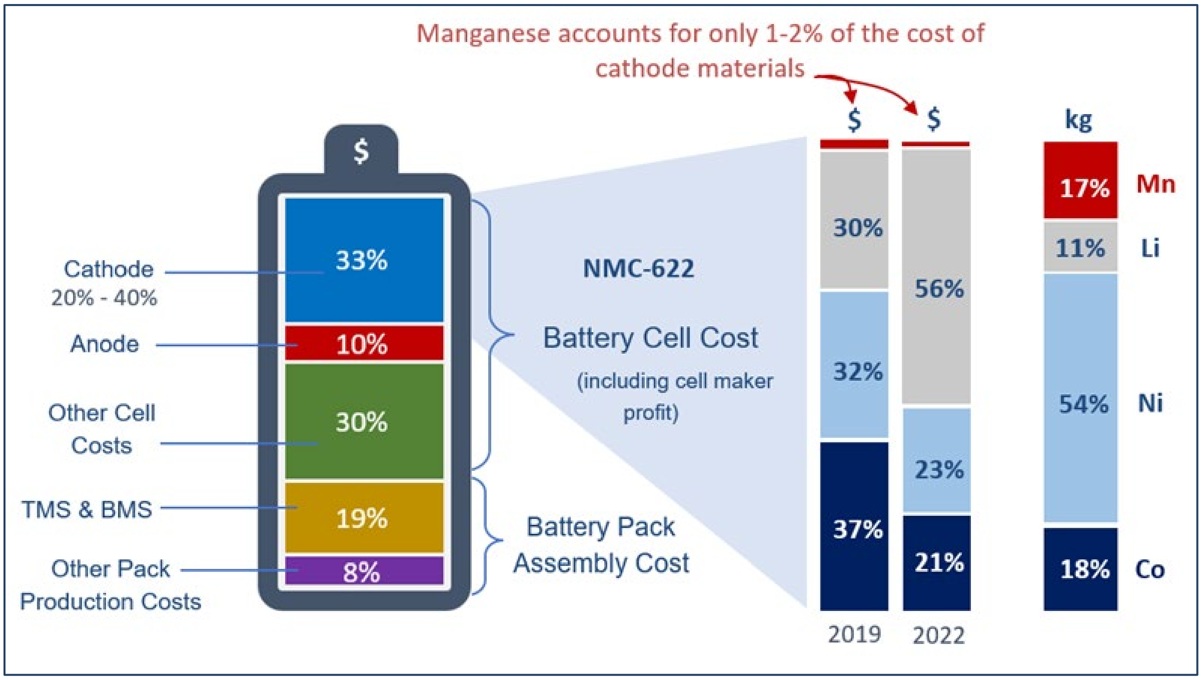


Note: In metal units, included HPEMM and HPMSM
Source: CPM (2022)

19.3 High-purity manganese sulphate monohydrate pricing

Prices set in bilateral agreements between the sellers and the buyers of HPMSM are, to a large extent, divorced from prices of other manganese products. Manganese is the cheapest of all battery metals and, despite contributing a similar weight (in kilograms) to cobalt (in the example shown in Figure 19.5), it accounts for only 1% to 2% of the cost of the cathode materials. This makes manganese virtually price insensitive for cathode makers; they will buy it even if the price doubles or triples, as long as they can secure the right purity (Figure 19.5).

Figure 19.5 Cost and weight of cathode materials in NMC-622 battery pack (2019, 2022)



Notes:
Prices used (per kilogram of battery-grade material, metal contained):
January 2019: lithium = $85, nickel = $18, manganese = $3.2, cobalt = $62
May 2022: lithium = $386, nickel = $31, manganese = $3.5, cobalt = $84
TMS - Thermal Management System (cooling)
BMS - Battery Management Systems (electronics)
Source: CPM (2022)

19.3.1 CPM price projection assumptions

CPM used ex-warehouse prices in China as a basis for their price forecasting. Freight cost, duties, and price premia for high quality, low carbon non-Chinese product were added to arrive at the European and North American prices. The European price is assumed to be delivered duty paid (DDP) Berlin because of its proxy to numerous Central and Eastern European battery factories. Similarly, the North American price is DDP Detroit as a hub for battery factories.

To calculate the price for its HPMSM product received by Giyani, Giyani has assumed that its sales will be made on a FCA basis in Durban, South Africa; therefore, the realised price shall be the prevailing benchmark price in Europe or North America, as estimated by CPM, less the cost of land and sea freight from South Africa and any applicable import tariffs. Table 19.1 illustrates the net realised aggregate prices for HPMSM shipped from the K.Hill Project to the ports of Rotterdam and Baltimore in Europe or North America, respectively, assuming an even split of sales between European and North American markets.

Table 19.1 Net realised high-purity manganese sulphate monohydrate (32% manganese) price projections

Year end	Average realised price net of transport and sales (US$/t of HPMSM)
2026	2,901
2027	3,146
2028	3,401
2029	3,648
2030	3,792
2031	4,021
2032	4,298
2033	4,601
2034	4,924
2035	5,370
Average 2026-35	4,010

Notes: Projected annual average prices in US$/t of HPMSM.
Net realised prices at K. Hill Project's gate assuming 50% of sales to the European Union (Berlin) and 50% sales to the USA (Detroit).
Real price base: 2021.
Source: CPM's (2022) calculations based on supply-demand assessment and historical prices reported by Bloomberg, AM, Argus, SMM, and industry sources.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

Giyani, through it's wholly owned subsidiary Menzi Battery Metals, engaged Loci to undertake an EIA for the K.Hill Project, in accordance with the Environmental Assessment (Amendment) Act, 2020 and the Environmental Assessment (Amendment) Regulations, 2021 of Botswana. Loci prepared an EIS, based on the work completed for the EIA, which was submitted to the Botswana DEA in July 2023.

The EIA includes a series of specialist studies covering archaeology and cultural heritage, biodiversity, hydrogeology and geochemistry, hydrology, traffic, noise, air quality, landscape and visual amenity, waste management, health, social, and rehabilitation/mine closure.

Public consultation was initiated during the EIA scoping stage, in line with the Environmental Assessment (Amendment) Act, 2020 and its associated guidelines for public consultation methodology. Consultation continued throughout the EIA process as the Project scope was refined and additional information became available from key informants.

The EIA assesses only the first 11 years of mine operations. Impact assessments beyond the 11-year LOM will need to be assessed by an independent environmental consultant. It is anticipated that any changes would not be material in terms of the EIA process and could be done through amendments on the existing EIA. The process of amending the EIA is well stipulated by the DEA.

In accordance with NI 43-101 Standards of Disclosure, this section includes:

- a summary of environmental baseline results and known environmental issues
- requirements and plans for waste and tailings disposal, site monitoring, and water management during mine operations and post-mine closure
- project permitting requirements, status of applications, and requirements for performance or reclamation bonds
- potential social- or community-related requirements and plans for the Project and the status of any negotiations
- mine closure requirements

Please see Figure 18.1 for an overall site layout map.

20.2 Policy, legal, and administrative framework

The K.Hill Project will comply with Botswana legislation and current policies, legal requirements, regional conventions, and international obligations to which Botswana is a signatory. Giyani has opted to conform with the IFC *Performance Standards on Environmental and Social Sustainability* (IFC 2012) and the *Equator Principles EP4* (Equator Principles Association 2020).

Table 1.1 outlines the permitting required for the K.Hill Project.

Table 20.1 Required permits and approvals for the K.Hill Project

Detail	Authority	Relevant legislation	Status	Comments
Mining licence	Botswana Department of Mines	Mines and Minerals Act (1999)	• Before Project construction begins.	• Application for a mining licence requires EIA approval, parent company guarantee, certificate of incorporation, identities of directors and shareholding, feasibility study report, rehabilitation program, surface rights, Department of National Museum and Monuments approval, and a valid prospecting licence. • Review time is 30 working days.
EIA approval	Botswana Department of Environmental Affairs	Environmental Assessment (Amendment) Act (2020)	• Prior to application for the mining licence. • Currently, an EIS has been submitted to the DEA for approval. The scope of the current EIS is only for the initial 11-year LOM. The EIS will have to be amended in future to address any changes made on the Project.	• A key document upon which other approvals rest. • Review time is 14 working days followed by a 10-working-day public review period.
Archaeological Impact Assessment approval and development licence	Botswana Department of National Museum and Monuments	Monuments and Relics Act (2001)	• Prior to EIA approval.	• Required for EIA approval and mining licence application. • Approval has been obtained and was included in the EIS submission.
Water borehole registration	Botswana Department of Water Affairs	Boreholes Act (1956)	• When drilling boreholes for the Project.	• Required that all boreholes be registered.
Water rights	Botswana Department of Water Affairs	Water Act (1968)	• Before Project construction begins.	
Construction working conditions	Botswana Department of Labour and Botswana Department of Mines	Factories Act (1979); Mines, Quarries, Works and Machinery Act (1978)	• Before Project construction begins.	• Once the mining licence has been issued, no other permit regarding construction is required for the Project.

Detail	Authority	Relevant legislation	Status	Comments
Operations working conditions	Botswana Department of Labour and Botswana Department of Mines	Employment Act (2001); Mines, Quarries, Works and Machinery Act (1978)	• Before Project operations begin.	
Work permits and residence permits	Botswana Department of Labour	Employment Act (2001)	• As and when needed.	• Permit applications take 3 months to process.
Transportation and storage of explosives	Botswana Department of Mines	Explosives Act (1968)	• Before any explosives are brought to the Project site.	
Surface rights	Ngwaketse Land Board	Tribal Land Act (1968)	• Before application for mining licence.	• Application review time is 1 month.
Permission to generate power for own use by solar plant	Botswana Department of Energy	Electricity Supply Act (1973)	• Before operation of solar plant.	
Permission to connect access roads to existing roads	District Council, Roads Department	Road Traffic Act (1981)	• Before access road construction.	
Licences (mining and EIA approval) for borrow pits	Botswana Department of Mines and Botswana Department of Environmental Affairs	Mines and Minerals Act (1999), Environmental Assessment (Amendment) Act (2020)	• Before excavation of any materials for construction.	
Extension of PLs	Botswana Department of Mines	Mines and Minerals Act (1999)	• Before the PL expires.	• Extension review time is 1 month.
Waste storage facilities (design) and transport	Botswana Department of Waste Management and Pollution Control	Waste Management Act (1998)	• Before Project construction begins.	

20.3 Baseline environment

20.3.1 Topography

The land surface of Botswana is mostly flat or gently undulating, with the greatest topographical relief in the southern parts of the country. Kanye lies at an elevation of approximately 1,300 masl, while the K.Hill Project site sits partially on Kgwakgwe Hill, forming the highest topographical feature alongside Kanye, which is the nearest village. Kgwakgwe Hill reaches nearly 1,500 masl.

20.3.2 Climate

Southeast Botswana's climate is sub-tropical and semi-arid with summer rainfall. The summer season in Kanye is long, warm, and partly cloudy, and the winter season is short, cold, dry, and clear. Temperatures vary from 4°C to 30°C and are rarely below 0°C or above 34°C. The warmest months of the year are January and December, with the hot season ranging from September to March. The coldest months are June and July, with the cool season ranging from May to August.

Winds in the study area blow predominantly from the northeast and east-northeast. The higher velocity winds (greater than 8 m/s) also blow predominantly from the northeast. Calm conditions (winds below 1 m/s) are noted only 7.8% of the time.

The K.Hill Project site receives most of its rainfall during the summer months (December to February). The site experiences the lowest rainfall levels during the winter months (June to August). Total annual rainfall for 2017, 2018, and 2019 was 971.1 mm, 824.6 mm, and 947.7 mm, respectively (based on modelled Fifth-Generation Penn State/National Center for Atmospheric Research Mesoscale Model [MM5] data).

20.3.3 Land use

The K.Hill Project site has been disturbed by previous mining activities dating back 50 years, livestock grazing, construction of the WUC reservoirs and the Botswana Communications Regulatory Authority (BOCRA) communications tower, and illegal waste dumping.

During the EIA, no ploughed fields were recorded within the K.Hill Project site; however, cattle were observed grazing on the hill. Two farms were recorded outside the Project site: a ploughing field approximately 3 km south-southwest of the Project site and a chicken farm (not in use) approximately 2.5 km southeast of the Project site. An abandoned cattle post was noted 1.4 km southwest of the Project site.

Other land users within the proposed mining licence boundary are livestock owners who have erected huts and cattle kraals. Cattle owners mostly occupy these huts, but some are occupied by cattle herders. These land users have no formal land rights but have been allowed by local tribal leaders to use the land under the condition that they do not develop permanent structures. Fifteen such land users were identified. Although the land belongs to the Ngwaketse Land Board, adequate relocation procedures will need to be followed to move these land users in collaboration with the Ngwaketse Land Board, who have set procedures for such situations.

A surface rights application has been lodged with the Ngwaketse Land Board and at the time of this report it was under review.

20.3.4 Groundwater

The aquifer systems in the study area consist mainly of low-permeability fractured rock systems associated with shales. The area further north and east of the Project site consists of fissured aquifer systems associated with dolomites. The area south of the Project site comprises mainly dolomitic aquifers, and higher yields can be expected. The aquifer transmissivity is approximately 700 m^2/d, and hydraulic conductivity is approximately 10 m/d. Data suggests a high-yielding aquifer with high aquifer flow characteristics for the dolomitic system west and south of the Project site. Borehole depths are variable, and it is assumed that major fracturing has intersected at deeper horizons. The indicative groundwater depth contour map suggests that groundwater depths vary between approximately 60 m and less than 100 m in the area surrounding the Project site.

20.3.5 Surface water

The Project site is partially located at the top of Kgwakgwe Hill, and runoff water drains east and west into local streams. Runoff from the western slopes flows west, approximately 3 km into the Nneneke River (catchment area of 35 km^2), which then converges from the eastern slopes in the Matlhapise River (catchment area of 32 km^2). The Matlhapise River turns into the Mokape River near the Lobatse road and then into the Masinyetse River. The Masinyetse River joins the Taung River 30 km west at Mogobane and then joins the Notwane River, which flows 150 km north into the Limpopo River and eventually the Indian Ocean. The Matlhapise River, 2 km east of the Project site, drains the southern slopes of Kanye. The catchment area is 26 km^2 and drains into the Mmamokhasi Dam, which catches some of the runoff from the Project site. It should be noted that there are no perennial streams near the Project site; the previously mentioned streams are all ephemeral. Mean annual evaporation from open water bodies was recorded for the Gaborone Dam by the Institute of Hydrology and the Department of Water and Sanitation and is estimated at an average of 1,600 mm/a. Evaporation losses exceed annual rainfall, creating an annual water deficit.

20.3.6 Soils

Soils are mostly Lithic Leptosols, which are rocky, shallow, sandy on the surface, and reddish in colour (based on observations made by EIA specialists during site visits). Soil fertility is low in the higher areas. Given the presence of sandy materials, dust control will be required throughout all mining phases.

20.3.7 Traffic

Traffic counts were undertaken at 11 count locations within Kanye to establish baselines. The general observation was that traffic in Kanye is low and all intersections function at good service levels. Traffic observed at all count locations primarily consisted of light vehicles.

20.3.8 Waste management

Waste management service delivery in Kanye (waste collection and disposal) is currently at a reasonable standard. Kanye has a relatively well-operated general waste disposal site and a health care risk waste treatment facility (despite not being equipped with scrubbers). Downtime on waste collection vehicles is problematic, as the fleet is old and limited finances are available to upgrade the fleet. The Kanye landfill also suffers from equipment breakdown and lack of environmental monitoring (air quality and groundwater). Illegal dumping of waste occurs in multiple locations on and around the Project site. Waste from the processing plant will be classified as either hazardous or non-hazardous;

hazardous waste will need to be disposed of at a licenced hazardous waste facility, which will be identified during the final waste classification. Botswana and South Africa are a signatories to the Basel Convention on the international movement of hazardous waste.

Mining waste for the K.Hill Project will comprise waste rock and tailings, which will be disposed of in the designated WRD and TMF within the Project site.

20.4 Biodiversity

20.4.1 Flora

Kanye falls within the tree savanna vegetation type and subtype Arid Sweet Bushveld. The Arid Sweet Bushveld is mainly undulating, with many rocky outcrops, valleys, and sandy plains. In the central area of this subtype (which is where Kanye is located) the dominant tree species are Silver Cluster-leaf (*Terminalia sericea*), Wild Seringa (*Burkea africana*), Shepherd's Tree (*Boscia albitrunca*), Large Fruited Bushwillow (*Combretum zeyheri*), Russet Bushwillow (*C. hereroense*), Marula (*Sclerocarya birrea subsp caffra*), and Karee (*Searsia lancea*), with shrubs including Umbrella Thorn (*Vachellia tortillis*), Plate Thorn (*Senegalia cinerea*), Sickle Bush (*Dichrostachys cinerea*), Common Gaurri (*Euclea crispa*), and Velvet Raisin (*Grewia flava*).

The hilly areas generally support a thick cover of trees and shrubs, mainly Buffalo Thorn (*Ziziphus mucronata*), Red Bushwillow (*C. apiculatum*), Leadwood (*C. imberbe*), Umbrealla Thorn (*V. tortillis*), Karee (*S. lancea*), and Sickle Bush (*D. cinerea*). In riverine areas or along drainage lines, dense thorn shrub is common, dominated by Vachellia and Senegalia species. Common grass species associated with the vegetation subtype include Common Finger Grass (*Digiteria eriantha*), Silky Bushman Grass (*Stipagrostis uniplumis*), Lehmann's Love Grass (*Eragrostis lehmanniana*), Herringbone Grass (*Pogonarthria squarrosa*), Sand Quick (*Schimidtia pappophoroides*), and Curly Leaf Grass (*E. rigidior*).

20.4.2 Ecosystem services

Among the woody and herbaceous species, several species provide valuable veld products that local communities can utilize. Aside from grazing resources, provisioning ecosystem services comprise plant-based food sources, insect-based food sources (typically those hosted by flora species), and traditional medicine uses. Of the flora species observed by Loci ecologists, 14 are known to provide provisioning ecosystem services in the form of veld products.

20.4.3 Alien and invasive species

Some alien and invasive flora species were observed in the study area. South African Daisy (*Verbesina enceloides*), Jimson Weed (*Datura stramonium*), and Lantana (*Lantana camara*) were observed during the site survey at the top of the hill and in areas around Kanye. South African Daisy (*V. enceloides*) and Jimson Weed (Datura stramonium) are commonly found invading areas that are disturbed; these species are often established on roadside reserves, sandy watercourses, and cultivated fields and are mildly toxic to small mammals and livestock. Invasive plants can suppress native vegetation, which in turn results in natural habitat degradation and loss of biodiversity.

20.4.4 Fauna

During the EIA site surveys, five mammal species were observed: Rock Dassie (*Procavia capensis*), Baboon (*Papio ursinus*), Common Slender Mongoose (*Galerella sanguinea*), an unidentified rodent, and

Southern African Ground Squirrel (*Xerus inauris*). All four mammal species identified are classified as least concern on the International Union for Conservation of Nature red list.

20.4.5 Herpetofauna

A desktop review found 44 reptile species whose distribution overlaps with the Kanye area, including 20 snakes, 7 lizards, 5 chelonians, 3 skinks, 3 geckos, 3 agamas, 2 rock monitors, and 1 chameleon. During the EIA field survey, a skink was recorded; however, several other reptile species are likely to occur at site. Communication with drilling teams working on Kgwakgwe Hill has revealed that snakes are common during warm months. Species cannot be confirmed, but are probably puffadders, cobras, and black mamba.

20.4.6 Avifauna

The Project site falls within the range distribution of 285 bird species. The bird diversity in Kanye is average to high, due to the variety of habitats in the greater area ranging from plains, hills, rocky outcrops, rivers, dams, and pans, all with varying vegetation types. All the habitats offer different qualities in terms of food sources and nesting and roosting spots to suit a variety of species. Kgwakgwe Hill offers a rocky habitat that suits various birds. The dams in Kanye are home to many water birds and attract other species. During the site surveys, 24 bird species were observed: 16 were within the Project site and 8 were observed at Mmamokhasi and Bathoen dams near Kgwakgwe Hill. The species noted on Kgwakgwe Hill include Crowned Lapwing, Double-Banded Sandgrouse, Rock Dove, Ring-Necked Dove, Laughing Dove, Burchell’s Coucal, African Hoopoe, Southern Yellow-Billed Hornbill, Crimson-Breasted Shrike, Magpie Shrike, African Red-Eyed Bulbul, Rattling, Cisticola, Tinkling Cisticola, and Mocking Cliff Chat.

20.4.7 Endangered species and species of conservation concern

Although desktop studies have shown that endangered flora and fauna and species of conservation concern may exist in the study area, none were observed on site during the EIA field surveys.

20.5 Socio-economic setting

Kanye is the administrative headquarters of the Southern District, located 83 km southwest of Gaborone, the capital city of Botswana. It is home to the Bangwaketse people, who settled in the area during the mid-19th century.

The administrative setting for Southern District is like other districts in Botswana and is managed by four local institutions:

- Southern District Council
- Ngwaketse Tribal Administration
- Ngwaketse Land Board
- Office of the District Commissioner in Kanye

20.5.1 Population

Kanye has a population of 48,028, comprising 22,273 males and 25,755 females (Statistics Botswana 2022).

20.5.2 Education

Pre-school facilities offer classes to pre-school aged children to prepare them for primary school. The pre-schools are either owned by individuals or run by communities through the Village Development Committee, as is the case in most pre-schools in the rural areas of the Southern District. The Southern District has a total of 58 daycare centres.

The mandate of the Department of Primary Education is to ensure that all children of school-going age (5 to 12 years) have access to education. The department currently faces a backlog for the provision of educational facilities. In 2019, Kanye had 273 classrooms to accommodate 8,027 pupils, or 29 pupils per classroom.

The Botswana Department of Secondary Education is responsible for quality assurance in all the secondary schools in a district. The Southern District has 24 community junior secondary schools and 3 senior secondary schools.

20.5.3 Existing health facilities

Currently, there are 15 health clinics in the Southern District, with an estimated 13 new clinics required to keep pace with population growth. A new clinic was recently commissioned in Letlhakane village, located approximately 15 km from Kanye. There are 53 health posts in the Southern District.

The Seventh Day Adventist Hospital serves as the Southern District hospital for the Kanye sub-district. The Seventh Day Adventist Hospital is a large hospital with 165 beds and an accident and emergency department with 10 stations. The hospital has 11 doctors, 2 dentists, and 152 nurses. There is a specialist surgeon and radiography and laboratory staff.

The main reasons for outpatient consultations comprise hypertension, diabetes, eye diseases, respiratory tract infections, and musculoskeletal conditions. Theatre work undertaken includes caesarean sections (10 to 20 per month), minor orthopaedics, minor abdominal surgery, incision, and drainage. Referrals are to the Princess Marina Hospital in Gaborone. Both hospitals have digital x-ray facilities and laboratories.

20.5.4 Economy

Kanye is the largest centre for economic activity in the Southern District. The Jwaneng Diamond Mine (operated by Debswana) is located in the Southern District and is one of the major sources of mining revenue for Botswana. Aside from the mine at Jwaneng, there are other smaller mining and quarrying activities in the Southern District. Residents of the Southern District rely on a mix of agriculture, trade, and formal employment for household livelihoods. Arable agriculture is more viable in the eastern part of the Southern District, with better soils and higher and more reliable rainfall. Arable agriculture covers approximately 10% of the total land area, while rangeland comprises more than half of the total land area (Ministry of Local Government 2019).

20.5.5 Literacy

The literacy rate for the population aged 15 to 65 years in Botswana is 90%. According to the National Literacy Survey 2014 by Statistics Botswana (2016), the Francistown, Orapa, and Sowa districts have the highest literacy rate of 98.1%, followed by Gaborone with 97.5%. The Southern District literacy rate is 80.8%, consisting of 63,961 literate people. Women are noted to be more literate than men, with 35,765 women recorded compared to only 28,196 men (Statistics Botswana 2016).

20.5.6 Crime and safety

The area's policing zone falls under the Sejelo Police Station. The main criminal activity in this policing area is common theft committed by youths aged between 20 to 35 years old. The statistics show that more crimes are committed by males than females. As crime remains a concern, various measures have been implemented to tackle it, including undertaking public education to sensitize the community; the stop, question, and search approach; and task teams/special operations.

20.5.7 Vulnerable groups and livelihood strategies

Vulnerable groups are people who by virtue of gender, ethnicity, age, physical or mental disability, economic disadvantage, or social status may be more adversely affected by project impacts. As such, the Botswana government supports the most vulnerable groups in all the districts. The destitute, needy, elderly, and orphans receive government support in the form of food baskets and other social grants to sustain livelihoods and improve their quality of life. Other government initiatives include the Ipelegeng Poverty Eradication Programme, which employs numerous people in the villages, and the Youth Development Fund, which targets the economic empowerment of youths in rural communities. In some of these settlements, underprivileged households receive livestock grants through the Rural Area Development Program. The elderly (65 years old and above), receive the old-age pension allowance every month to sustain their livelihoods and become less impoverished. Botswana has no specific laws on Indigenous peoples' rights in Botswana nor is the concept of Indigenous peoples recognised, rather the country maintains that all citizens of the country are indigenous.

20.5.8 Energy use

In the Southern District, most households (43%) use electricity, followed by those reported to be using paraffin at 39%. A further 11.1% of the households use candles for lighting, while firewood usage is low (less than 5%).

In Kanye, 40% of households use wood for cooking, followed by liquefied petroleum gas at 35%, electricity from the national grid at 20.9%, and paraffin at only 1.4% of households.

20.5.9 Archaeology and cultural heritage

The Kgwakgwe Hill area is rich in archaeological resources; however, no archaeologically significant materials were recorded during EIA surveys of the Project site. All staff involved in excavation and construction activities will be trained by a qualified archaeologist regarding chance-finds procedures.

Guided tours revealed that the Kgwakgwe Hills are visited by spiritual groups who also use the mine adits as cave shelters for rituals purposes. One cultural and historical component of the Project site is the Solomon's Temple Church for spiritual and healing practices. The church is located at the foot of Kgwakgwe Hill on historic mine structures. Project activities will affect the church, and consultations have been ongoing with the healer to reach an amicable and culturally appropriate solution. An alternative location has been identified for the temple, and the resettlement is in accordance with the IFC Performance Standard 5. The resettlement will be completed before any construction will commence.

20.6 Environmental management plan

A Project EMP was developed to manage the environmental and social impacts predicted during the EIA. An EMP provides a framework for mitigating environmental impacts associated with the Project and its activities. This includes a summary of all potential environmental impacts expected during all phases of the Project (i.e., construction, operations, and rehabilitation/closure) and mitigation measures that address each impact and assign roles and responsibilities for personnel who will implement the EMP.

The DEA created the structure of the EMP through guidance within the Environmental Assessment (Amendment) Act, 2020. The use and implementation of the EMP will be, primarily, the responsibility of Menzi Battery Metals, who will oversee contractors for all phases (design, construction, operations, decommissioning) of the Project and implement the EMP as a tool to avoid, reduce, and remedy negative impacts (and enhance positive impacts), where applicable. It is most likely that other management plans, such as biodiversity management plan and a critical habitat assessment needs to be developed.

20.7 Monitoring

A Monitoring Plan will be put in place to make sure the Project complies with environmental requirements, as per legislation and standards. For each possible impact, monitoring includes the parameter to be monitored, monitoring locations, key performance indicators, the agent responsible for monitoring, monitoring methodology, frequency, reporting mechanism, threshold, and recommended action when the threshold is exceeded. Monitoring will also give an indication of the effectiveness of mitigation measures and provide for improvements where required. This will include biodiversity monitoring and the biodiversity action plan.

20.8 Closure and decommissioning

A Mine Closure Plan needs to be developed for the Project, with the primary goal of leaving behind an enduring and positive legacy after mining, in which closure planning has been adequately resourced and integrated to facilitate effective transition of the mine lease area back to land authorities or other third parties. Post-Project land use and landscaping will form part of the consultation exercise for closure.

Decommissioning and closure objectives will ensure that the site is:

- safe for people and wildlife
- stable (rates of change for geochemical, geotechnical, and geomorphic [erosion and deposition] processes are acceptable)
- non-polluting (long-term performance meets the Proponents commitments to protect environmental values)
- able to sustain the agreed upon post-mining land use (agreed to by authorities and those involved with developing post-closure land use)
- socially and visually acceptable
- adequately provisioned with financial resources for closure
- possible to relinquish back to the Ngwaketse Land Board after mining

21 CAPITAL AND OPERATING COST ESTIMATES

21.1 Capital costs

21.1.1 Summary

Initial, sustaining, and closure cost capital estimates were prepared for the Project. The estimate is a Class 5 estimate prepared in accordance with the AACE® International Cost Estimate Classification System with an accuracy estimate of -20% to +30%.

The total initial capital cost for the K.Hill Project base case is US$282.6M, including a contingency of US$62.5M. It is expected that the initial capital program will be managed on an engineering, procurement, construction management (EPCM) basis with support from the Giyani owner's team. In addition, the operating cost for the mine pre-strip is capitalised, totalling US$1.1M.

The total estimated capital cost for the K.Hill Project base case is US$579.1M, as presented in Table 21.1. This total estimated capital cost also includes sustaining capital of US$288.0M and closure cost of US$8.4M.

Table 21.1 K.Hill Project capital cost summary

Cost area	Cost (US$M)
Mining	2.0
Processing	94.1
Infrastructure and Services	37.1
TMF	9.2
Indirect costs	48.8
Construction overheads	20.7
Owner's costs	8.2
Total initial capital cost (excluding contingency)	**220.1**
Contingency (28%)	62.5
Total initial capital cost (including contingency)	**282.6**
Sustaining capital	288.0
Closure cost	8.4
Total K.Hill Project capital costs	**579.1**

Note: The sum of costs may differ from the total due to rounding.

21.1.2 Responsibilities

Various professionals were responsible for estimating the capital cost for the different components of the PEA. These parties are identified in Table 21.2.

Table 21.2 K.Hill Project Preliminary Economic Assessment capital cost estimate responsibilities

Area	Company
Mining	Axe Valley Mining
Processing	Tetra Tech
Infrastructure and Services	Tetra Tech
TMF	Knight Piésold
Indirect costs	CSA Global
Sustaining capital	Axe Valley Mining Tetra Tech Knight Piésold CSA Global
Construction overheads	Tetra Tech
Owner's costs	CSA Global
Closure cost	Knight Piésold
Contingency	CSA Global

21.1.3 Validity and scope

The following items are excluded from the estimate:

- escalation
- allowance for foreign exchange rate fluctuations
- schedule delays and associated costs, which may be related to unexpected ground conditions, extraordinary climate events, and labor disputes
- delays associated with receipt of information beyond the control of the EPCM team
- financing costs associated with the project

21.1.4 Direct costs

Mining

The mining operations are assumed to be conducted using a mining contractor who will be responsible for purchase, operation, and replacement of the equipment used at the mine. The mobilisation of the mining contractor is estimated at US$2.0M.

Processing

Conceptual process mass balance and process design criteria were used to generate the requirements for process equipment. Budget estimates from ongoing and completed projects comparable to the K.Hill Project were used to determine the costs for mechanical equipment and building supplies, which were then scaled for size. The estimated processing plant capital costs are presented in Table 21.3.

Table 21.3 K.Hill Project processing plant capital cost estimate

Cost area	Cost (US$M)
Mechanical equipment	69.4
Piping and valves	4.5
Instrumentation and controls	1.3
Electrical	2.9
Civil and buildings	3.3
Structural steel	3.7
Platework	9.0
Total processing plant capital costs	**94.1**

Note: The sum of costs may differ from the total due to rounding.

The processing plant capital costs are based on priced items from recognised mining equipment suppliers that were developed from to a mechanical equipment list. The equipment list items were sized and selected based on the high-level mass balance that was completed along with the process flow sheets.

Infrastructure and services

Capital costs for infrastructure and services include all powerlines, roads, concrete foundations, and earthworks required for the Project. The estimated infrastructure and services capital costs are presented in Table 21.4.

Table 21.4 K.Hill Project infrastructure and services capital cost estimate

Cost area	On-site Infrastructure (US$M)	Off-site Infrastructure (US$M)
Civil and buildings	12.5	9.1
Electrical	4.1	0.1
Instrumentation and controls	6.7	0.1
Mechanical equipment	1.6	-
Mobile equipment	1.5	-
Piping	0.2	-
Structural Steel	1.2	-
Total infrastructure and services capital costs	**27.8**	**9.3**

Note: The sum of costs may differ from the total due to rounding.

Construction overheads

Construction overheads are the preliminary and general costs of contractors operating and executing work on site, but that are not directly associated with the quantity of material, and have been estimated at US$20.7M.

Tailings management facility

The capital costs associated with the construction of the TMF include:

- site clearance
- mass excavation (removal and stockpile of topsoil, subsoil, and unsuitable foundation material)
- TMF embankment and intercell bund construction
- settlement pond embankment construction
- geotextiles, geo-composites, and geomembrane liners
- vertical drain construction
- diversion ditch excavation
- perimeter channel excavation and lining
- monitoring instrumentation
- tailings transport and deposition
- capping TMF at closure

Allowances for engineering and contractor costs include:

- contractor mobilisation/demobilisation: included in 15% of capital cost (preliminary and general)
- construction management: lump sum spread over construction periods within LOP
- construction quality assurance supervision: lump sum spread over construction periods within LOP

Construction requirements that have not been addressed in the TMF capital costs include:

- recruiting and training costs
- site communications
- accommodation and messing
- human resources and safety resources and equipment
- environmental monitoring
- office, stores, workshops, refuelling, servicing, and maintenance facilities
- utilities for fixed infrastructure, such as power, water, sewerage, communications, etc.
- contingency items

The estimated TMF capital costs are summarized in Table 21.5.

Table 21.5 K.Hill Project tailings management facility capital cost estimate

TMF Phase	Initial Capital Cost (US$M)	Sustaining Capital Cost (US$M)
Construction	$9.2	-
TMF Phase 1	-	$10.7
TMF Phase 2	-	$14.0
TMF Phase 3	-	$22.0
TMF Phase 4	-	$11.4
TMF Phase 5	-	$1.3
Total TMF capital costs		**$68.5**

Note: The sum of costs may differ from the total due to rounding.

21.1.5 Indirect costs

Indirect capital costs, estimated at US$48.8M, include:

- commissioning, including vendor support, processing plant operational costs, and commissioning spares
- engineering and management, including EPCM contractor, basic and detailed engineering, specialist consultants, and an allowance for value engineering
- first-fill and spares, including provision for the initial stocking of stores with consumables and general spares, which represent approximately 2 weeks of inventory
- operational readiness, including overhead, an operational team, and training for the start of production
- transport and logistics, including the cost of logistics of major equipment and transport costs

The estimated infrastructure and services capital costs are presented in Table 21.6.

Table 21.6 K.Hill Project indirect cost estimate

Cost area	Cost (US$M)
Commissioning	2.6
Engineering and management	30.4
First fills	6.9
Spares	4.6
Training	0.2
Transport and logistics	4.0
Total indirect costs	**48.8**

Note: The sum of costs may differ from the total due to rounding.

21.1.6 Owners costs

The owner's costs include provisions for the following items:

- cost of the owner's Project team during execution; the team will comprise both full-time Giyani personnel and part-time external consultants
- cost of the application for licences and permits
- insurance
- owner's G&A cost

The estimated owner's costs are presented in Table 21.7.

Table 21.7 K.Hill Project owner's cost estimate

Cost area	Cost (US$M)
Owner's team	3.1
Licences	0.5
Insurance	2.0
Owner's G&A	1.0
Miscellaneous	1.6
Total owner's costs	**8.2**

Note: The sum of costs may differ from the total due to rounding.

21.1.7 Contingency

A 28% contingency, based on the total direct and indirect costs, has been included for items, conditions, or events for which the state, occurrence, or effect is uncertain.

The contingency percentage for each discipline was individually assessed on the accuracy of quantity measurement, type, scope of work, and price information. The contingency for the Project has been calculated to be US$62.5M.

21.1.8 Sustaining capital

An allowance for sustaining capital has been provided for as a proportion of the capital expenditure for each of the major capital cost areas. Sustaining capital has been defined as capital required to maintain the nameplate capacity of the operation and may include the replacement or refurbishment of existing assets as required. The estimated sustaining capital is estimated at US$287.7M.

Table 21.8 Sustaining capital value per cost area

Cost area	Sustaining capital cost assumption	Value (%)
Mining	Mining annual sustaining capital cost allowance	1.00
Processing	Processing annual sustaining capital cost allowance	2.60
Infrastructure and Services	Infrastructure and services annual sustaining capital cost allowance	1.00
TMF	TMF annual sustaining capital cost allowance	14.40
Offsite infrastructure	Offsite infrastructure annual sustaining capital cost allowance	1.00
Indirect costs	Indirect costs annual sustaining capital cost allowance	1.00

21.1.9 Closure costs

TMF closure costs comprise US$3.8M and the current EIA closure and general rehabilitation plan funding requirements comprise US$4.6M. The closure cost is scheduled 12 months prior to the end of LOM.

21.2 Operating costs

21.2.1 Summary

The operating costs for the K.Hill Project consist of mining, processing (including water treatment), G&A, and tailings management.

The total average LOP unit operating cost, excluding royalties, is estimated at US$579/t of material processed (Table 21.9). Mining, processing, and G&A costs constitute 7.4%, 89.4%, and 3.0% of the total unit operating cost, respectively. The average unit operating costs may differ slightly from the nominal values calculated for each cost area (Sections 21.2.2 to 21.2.5) because they represent a weighted average over the LOP.

Table 21.9 K.Hill Project average unit operating cost summary

Cost area	Unit cost (US$/t feed)	Contribution to operating cost (%)
Mining	43	7.4
Processing	517	89.4
G&A	18	3.0
TMF	1	0.2
Total unit operating cost(1)	**579**	**100**

Note:
(1) Total operating costs, excluding royalties.
The sum of costs may differ from the total due to rounding.

21.2.2 Mining

The LOM operating cost of US$2.50/t of waste mined used in the open pit optimisation is based on factoring estimates from previous studies (SRK 2019, 2020) and benchmarking against quotes for other similar contract mining operations in Africa. These quotes mainly came from contract miners based in South Africa, so they are considered appropriate for this exercise.

The reference mining cost for waste is US$2.50/t, and a mining cost adjustment factor of 1.2 has been applied to the mineralized material (US$3.00/t) to account for the longer haul.

Low-grade material reclaimed from stockpiles has an additional operating cost of US$1.00/t mined. This includes the cost of stockpile management and reclaiming the material to the ROM pad, which is assumed to be within 500 m of the stockpile.

The mine infrastructure (workshops, administration building, fuel/lube facilities, etc.) will be supplied by the owner.

21.2.3 Processing

The Project development plan considers a production rate of 200 kt/a. Process operating costs for are presented in Table 21.10.

Table 21.10 K.Hill Project process operating cost estimate

Cost area	Unit operating cost (US$/t)	
	Years 1 to 4	Years 5 to 57
Energy	64.0	64.0
Labour	8.8	8.8
Laboratory/quality control	0.7	0.7
Water treatment	18.7	18.7
Maintenance	76.5	76.5
Tailings management	1.8	1.8
Solar plant	0.1	0.1
Raw material	423.4	339.8
Total process operating costs	**594.0**	**510.4**

Note: The sum of costs may differ from the total due to rounding.

The estimated processing operating cost is based on the following information:

- process design criteria, where quantities are based on historical metallurgical test work
- projected reagent usage forecasts based on the mineralogy grade indicated in the mine schedule for manganese, iron, aluminium, and potassium
- process mass balance model for estimation of the reagent consumptions using the mining schedule data
- reagent costs provided to Giyani by quantity surveyors and cost accounting company VDDB.

Processing plant labour

The annual cost of labour (by category) is presented in Table 21.11.

Table 21.11 K.Hill Project annual cost of processing plan labour

Labour Type	Annual Cost (US$)
Assayer	15,900
Assistant	4,300
Boiler maker	10,800
Instrumentation and controls technician	15,900
Electrician	10,800
Fitter	10,800
General/plant manager	103,400
General hand	3,400
Laboratory clerk	10,800
Laboratory general hand	3,400
Laboratory technicians	5,200
Maintenance clerk	4,300
Maintenance foreman	15,900

Labour Type	Annual Cost (US$)
Metallurgical clerk	10,800
Metallurgists	25,900
Operator general hand	3,400
Operators	5,200
Plant foreman	15,900
Plant superintendent	32,800
Resident engineer	32,800
Services foreman	15,900
Services general hand	3,400
Services operator	5,200
Shift boss comminution	10,800
Shift boss extraction	10,800
Shift boss leach	10,800
Shift boss services	10,800
Tailings foreman	15,900
Tailings general hand	3,400
Tailings operator	5,200
Tailings superintendent	32,800
Warehouse clerk	4,300
Warehousing/logistics manager	15,900

Fuel cost

Diesel and petrol will be supplied at the mine site by one of the three main suppliers in Botswana: Oryx, Total, or Shell. Indicated fuel prices are:

- diesel: P8.09/L or US$0.65/L
- petrol: P9.03/L or US$0.72/L

Costs of bulk deliveries (freight), storage, and handling are included in the fuel prices.

Power cost

BPC will supply electricity to the Project site. The estimated power cost that has been used in the PEA is US$0.12/kWh.

21.2.4 Tailings management facility

The TMF operating costs are presented in Table 21.12.

Table 21.12 K.Hill Project tailings management facility operating cost estimate

Phase	Cost (US$M)
Construction	-
TMF Phase 1	$0.7
TMF Phase 2	$1.3
TMF Phase 3	$2.1
TMF Phase 4	$1.8
TMF Phase 5	$1.5

Phase	Cost (US$M)
Closure	$3.8
Total TMF operating costs	**$11.2**

Note: The sum of costs may differ from the total due to rounding.

All costs associated with TMF pumping, lighting, and transportation are included in the processing operating costs. The unit TMF operating cost is US$1/t feed to plant (or $0.45/m^3 of tailings deposited).

21.2.5 General and administrative

The G&A cost estimate basis includes the following:

- Labour costs (personnel rates) were derived from *D271 GA Cost Estimate Report* by VDDB, provided by Giyani.
- All other costs, including equipment costs, facilities, and laboratory services, were derived from *D271 GA Cost Estimate Report* by VDDB, provided by Giyani.
- The total estimated G&A cost is US$3.52M/a, or US$18.00/t processed, as shown in Table 21.13.

Table 21.13 General and administrative operating cost estimate

Area	Cost (US$/a)
Labour	613,196
Equipment	53,068
Facilities	199,609
Utilities	37,350
Services	1,136,153
Botswana corporate	667,248
Overheads	301,242
Laboratory services	52,708
Contingency	462,081
Total G&A operating costs	**3,522,655**

Note: The sum of costs may differ from the total due to rounding.

22 ECONOMIC ANALYSIS

22.1 Introduction

CSA Global performed an economic analysis for the K.Hill Project PEA study. Cash inflows are based on annual production and revenue projections of the HPMSM product, while cash outflows consist of capital costs (mining, processing, infrastructure, and TMF), sustaining capital costs (mining, processing, infrastructure, and TMF), operating costs, royalties, and taxes. Royalties referred to here are tax royalties levied by the Botswana authorities and not financing royalties (Table 22.1).

The modelling period covers the 57-year LOP, which includes a 2-year construction period, a 2-year ramp-up period to full production following processing plant commissioning, and an 8-year processing period after mining has ceased in Year 49, in which stockpiled material will be fed to the processing plant until Year 57.

The K.Hill Project PEA includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. The results of the K.Hill Project PEA represent forward-looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the PEA. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially, are presented in the body of this report under each relevant section.

Table 22.1 Key Project metrics

Metric	Unit	Base case		
Project economics				
NPV (8% discount rate)	US$M	984		
IRR	%	29%		
Cumulative cash flow, undiscounted	US$M	5,283		
Project production		Year 1-5	Year 1-25	LOP
Total material mined	Mt	2.3	5.8	11.1
Average annual material processed	kt/a	170.0	194.0	196.0
Average plant feed grade	% MnO	19.1	17.3	15.2
Total HPMSM produced	kt	341	1,767	3,561
LOP	years	5	25	57
Net pricing assumptions				
Average realized HPMSM price (Years 1 to 5)	US$/t	3,559		
Average realized HPMSM price (Year 6 onwards)	US$/t	3,780		
Capital expenditure				
Total initial capital expenditure	US$M	282.6		
Total sustaining capital	US$M	18	142	288
Project cash flows				
Total revenue	US$M	1,214	6,620	13,387

Metric	Unit	Base case		
Total operating costs (including royalty)	US$M	579	2,905	6,458
Total EBITDA	US$M	635	3,715	6,929

22.2 Pre-tax model

CSA Global reviewed and analysed a 100% equity financial model in real terms based on the mining and processing schedules. A summary of the capital and operating costs is detailed in Table 22.2 and Table 22.3. The operating cost for the mine pre-strip is capitalised, totalling US$1.1M.

Table 22.2 K.Hill Project capital cost summary

Cost area	Cost (US$M)
Mining	2.0
Processing	94.1
Infrastructure and services	37.1
TMF	9.2
Indirect costs	48.8
Construction overheads	20.7
Owner's costs	8.2
Total initial capital costs, excluding contingency	**220.1**
Contingency on initial capital cost	62.5
Total initial capital cost, including contingency	**282.6**
Sustaining capital, including contingency	288.0
Closure cost	8.4
Total	**579.1**

Note: The sum of costs may differ from the total due to rounding.

Table 22.3 K.Hill Project average unit operating cost summary

Cost area	Unit cost (US$/t feed)	Contribution to operating cost (%)
Mining	43	7.4
Processing	517	89.4
G&A	18	3.0
TMF	1	0.2
Total(1)	**579**	**100**

Note:
(1) Total operating costs, excluding royalties.
The sum of costs may differ from the total due to rounding.

A contingency cost has been calculated, equal to an allowance of 28% of the total initial capital cost, including direct capital costs, indirect capital costs, construction overheads, and owner's costs. The sustaining capital costs include mining, processing, and TMF sustaining capital costs over the LOP.

An initial capital cost of US$282.6M will be expended over the first 2 years of the Project schedule. All revenue and costs were modelled annually to match the mining and processing schedules. The schedule uses HPMSM forecast prices from CPM (2022) prepared for Giyani (Section 19).

22.2.1 Working capital

The working capital allowance is limited to debtor and creditor adjustments. The following working capital assumptions are included in the financial model:

- debtors: 90 days
- creditors: 30 days

22.2.2 Commissioning and ramp up

It is assumed in the financial model that the processing plant will be constructed over 24 months, followed by a 2-year ramp-up schedule to reflect the complex nature of the K. Hill Project processing plant. Metallurgical recoveries are assumed to start at 75.2% in the first year of processing plant production and ramp up to meet nameplate recovery of 88.50% in the second year.

Over the LOP, processing costs average 21% fixed and 79% variable, with fixed costs being mainly labour costs.

Mining costs average 28% fixed and 72% variable over the LOM, while tailings average 32% fixed and 68% variables over the LOM.

Due to the nature of the orebody, access to material will be immediate upon the commencement of production. As such, mining production will be an immediate need for stockpiling material.

A summary of the LOP pre-tax net free cash flow (NFCF) is presented in Figure 22.1. The cash flow summary is shown in Table 22.4.

Figure 22.1 Net free cash flow (pre-tax)

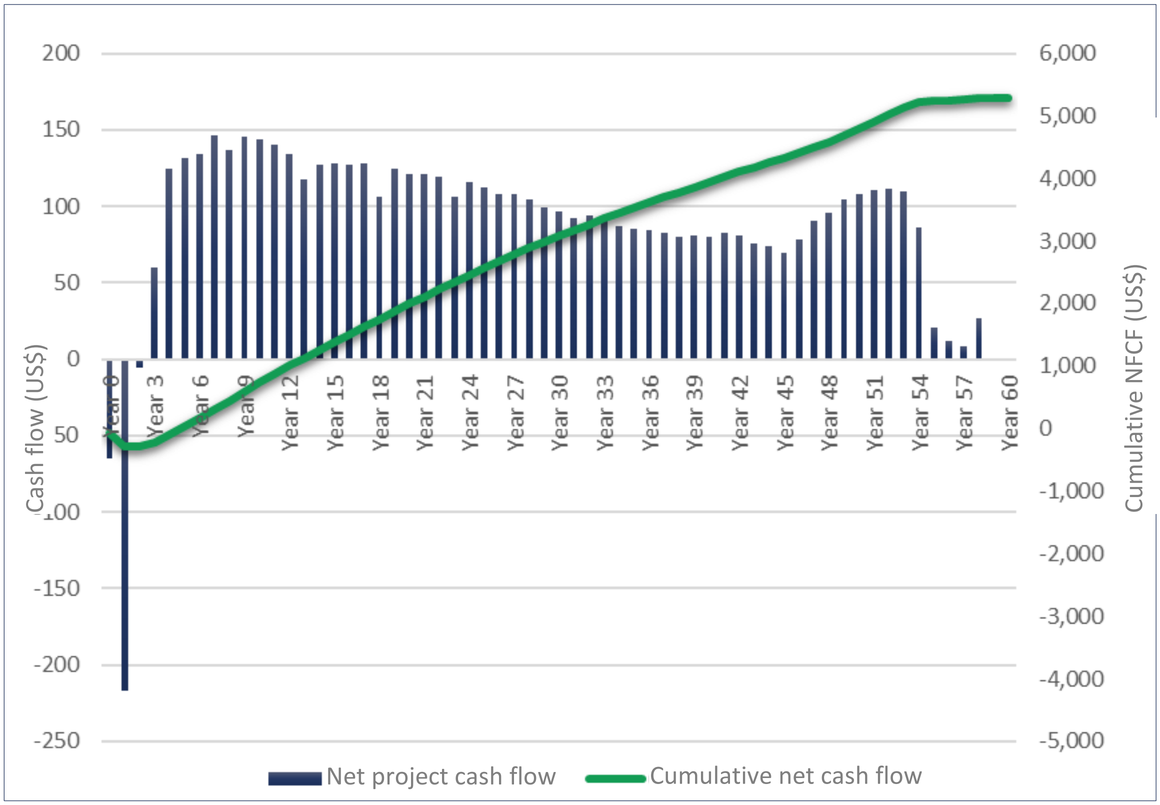


Table 22.4 Cash flow summary

	Unit	Total/Average	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Years 11-20	Years 21-30	Years 31-40	Years 41-50	Years 51-58
Production																		
Waste mined	t	123,702,990	-	64,358	2,140,963	2,984,767	2,712,120	2,959,592	3,219,531	2,987,072	3,000,677	2,978,984	2,869,860	26,188,963	25,208,889	24,034,053	22,353,162	-
ROM mined	t	11,125,080	-	7,642	273,437	522,433	789,740	505,489	251,969	79,628	73,023	79,849	117,296	2,710,001	1,123,613	2,225,889	2,365,070	-
Stripping ratio (waste to mineralized material)	t:t	11.12	-	-	7.83	5.71	3.43	5.85	12.78	37.51	41.09	37.31	24.47	9.66	22.44	10.80	9.45	-
Total material moved	t	134,828,070	-	72,000	2,414,400	3,507,200	3,501,860	3,465,081	3,471,500	3,066,700	3,073,700	3,058,832	2,987,156	28,898,964	26,332,503	26,259,942	24,718,232	-
Processing																		
Material processed	t	11,125,080	-	-	56,000	160,667	200,000	200,000	200,000	200,000	200,000	200,000	200,000	2,000,000	2,000,000	2,000,000	2,000,000	1,508,413
MnO grade processed	%	15.20	0	0	20	20	19	19	18	18	18	18	18	17	16	14	14	12
MnO content	t	1,690,765	-	-	11,004	32,213	38,310	37,460	36,931	36,554	36,274	35,994	35,634	340,451	314,135	280,276	281,471	174,058
MnO process recovery	%	88.49	0.00	0.00	86.73	88.50	88.50	88.50	88.50	88.50	88.50	88.50	88.50	88.50	88.50	88.50	88.50	88.50
MnO recovered	t	1,496,131	-	-	9,543	28,509	33,905	33,152	32,684	32,350	32,102	31,854	31,536	301,299	278,010	248,044	249,102	154,041
HPMSM tonnes produced	t	3,560,793	-	-	22,712	67,851	80,693	78,902	77,788	76,994	76,404	75,813	75,056	717,092	661,663	590,345	592,862	366,618
Macro economics																		
Average realised HPMSM price	US$/t	3,760		2,710	3,120	3,365	3,621	3,868	4,011	4,011	4,011	4,011	4,011	4,011	4,000	3,993	3,993	3,993
Revenue																		
Total revenue	US$M	14,169	-	-	75	240	304	309	311	309	306	304	301	2,876	2,647	2,357	2,367	1,464
Cost of sales	US$M	(782)	-	-	(5)	(15)	(18)	(17)	(17)	(17)	(17)	(17)	(16)	(157)	(145)	(130)	(130)	(18)
Net revenue	US$M	13,387	-	-	70	225	287	291	294	292	290	287	285	2,719	2,502	2,227	2,237	1,383
Operating costs																		
Mining	US$M	(475)	-	-	(8)	(11)	(11)	(11)	(11)	(12)	(10)	(10)	(10)	(101)	(93)	(95)	(90)	(2)
Processing	US$M	(5,756)	-	-	(34)	(96)	(119)	(119)	(119)	(104)	(102)	(102)	(102)	(1,022)	(1,022)	(1,022)	(1,022)	(771)
TMF	US$M	(11)	-	-	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(2)	(2)	(2)	(2)	(1)
G&A	US$M	(199)	-	-	(3)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(35)	(35)	(35)	(35)	(27)
Mining royalties	US$M	(18)	-	-	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(4)	(3)	(3)	(3)	(2)
Total operating costs (including royalties)	US$M	(6,440)	-	-	(45)	(111)	(134)	(134)	(134)	(120)	(116)	(116)	(116)	(1,160)	(1,152)	(1,154)	(1,150)	(801)
Capital costs																		
Mining division - pre-operations	US$M	(1)	-	(0)	(1)	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing division - pre-operations	US$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mining	US$M	(2)	(0)	(2)	(0)	-	-	-	-	-	-	-	-	0	-	-	-	-
Processing	US$M	(94)	(22)	(72)	(0)	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure	US$M	(28)	(6)	(21)	(0)	-	-	-	-	-	-	-	-	-	-	-	-	-
TMF	US$M	(9)	(2)	(7)	(0)	-	-	-	-	-	-	-	-	-	-	-	-	-
Offsite infrastructure	US$M	(9)	(2)	(7)	(0)	-	-	-	-	-	-	-	-	-	-	-	-	-
Indirect costs	US$M	(49)	(11)	(37)	(0)	-	-	-	-	-	-	-	-	-	-	-	-	-
Construction overhead costs	US$M	(21)	(5)	(16)	(0)	-	-	-	-	-	-	-	-	-	-	-	-	-
Owner's cost	US$M	(8)	(2)	(6)	(0)	-	-	-	-	-	-	-	-	-	-	-	-	-
Working capital	US$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contingency	US$M	(63)	(14)	(48)	(0)	-	-	-	-	-	-	-	-	-	-	-	-	-
Total sustaining capital	US$M	(288)	-	-	(4)	(3)	(3)	(3)	(3)	(3)	(13)	(3)	(3)	(68)	(52)	(46)	(46)	(35)
Closure cost	US$M	(8)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(8)
Net working capital movement	US$M	-	-	-	(24)	(36)	(4)	(1)	(1)	1	(1)	1	1	5	9	3	(8)	57
Total Project capital	US$M	(580)	(65)	(217)	(30)	(39)	(8)	(4)	(4)	(2)	(14)	(3)	(3)	(63)	(43)	(43)	(55)	13
Economics																		
Operating profit - EBITDA	US$M	6,929	70	225	242	180	160	158	156	168	168	165	161	1,519	1,287	1,039	1,147	85
Net free cash flow pre-tax	US$M	6,349	4	8	212	141	152	153	151	165	154	162	159	1,457	1,244	996	1,092	98
Corporate tax liability	US$M	(1,066)	-	-	(1)	(14)	(20)	(21)	(21)	(23)	(23)	(23)	(22)	(239)	(210)	(169)	(171)	(108)
Net free cash flow post-tax	US$M	5,283	4	8	211	127	132	132	130	142	131	139	136	1,218	1,034	827	921	(10)

The pre-tax NPV was calculated for the base case using discount rates ranging from 0% to 15% (Table 22.5).

Table 22.5 Net present value pre-tax summary

	Unit	Discount Rate					
		0%	5%	8%	10%	12%	15%
NPV	US$M	6,349	2,014	1,210	899	681	460

At an 8% discount rate, the pre-tax NPV for K.Hill Project is US$1,210M ,with an IRR of 33%, and a payback period of 2 years (Table 22.6).

Table 22.6 Pre-tax financial performance

Project economics	Unit	Value
NPV[1] (8% real discount rate)	US$M	1,210
IRR	%	33
Payback period, from the start of processing	years	2

Notes:
(1) Percent real discount rate.

22.3 Sensitivity Analysis

Figure 22.2 shows the sensitivity of the pre-tax NPV at an 8% discount rate to product price, operating cost, and capital cost. The Project's NPV is most sensitive to the product price over a range of -10% to +10%, which can also be a proxy for grade and recovery. The Project is less sensitive to operating and capital costs, as shown in Figure 22.2.

Figure 22.2 Pre-tax net present value sensitivity

22.4 Post-tax analysis

The tax regime applied in the financial model is per SRK's assumption and consultations on the Botswana tax regime and the business model that Giyani intends to utilise for the K.Hill Project from the previously completed Technical Report (SRK and Tetra Tech 2022).

From discussions with local stakeholders, Giyani intends to split the K.Hill Project operation into two business units: mining and manufacturing. The mining operation will sell material to the manufacturing operation at an assumed long-term manganese material price.

The mining unit will be taxed according to the Botswana mining company tax formula, which has a minimum level of 22% and will be based on the sale of manganese material to the manufacturing unit (PwC 2022). The mining tax formula is as follows: a mining royalty of 3% will be applied to the revenue on the sale of the manganese material to the manufacturing unit. Capital investments will depreciate immediately for the mining unit, and unredeemed capital will be carried forward indefinitely, as allowed for mining projects in Botswana, as detailed in the previously completed Technical Report (SRK and Tetra Tech 2022).

The manufacturing unit will be taxed on the manufacturing tax rate, assuming a manufacturing development order will be received, resulting in a tax rate of 15% (Deloitte 2021). For the manufacturing unit, initial capital investments will depreciate at 10% per year on a straight-line basis, and sustaining capital costs will depreciate at 20% per year. Table 22.8 presents the post-tax analysis results.

Table 22.7 Post-tax financial performance

Parameter	Unit	Value
Revenue (excluding cost of sales)	US$M	14,169
Operating cost (including royalties)	US$M	6,458
Operating profit	US$M	6,929
Tax liability	US$M	1,066
Capital cost	US$M	579
Post-tax cashflow	US$M	5,283
NPV (8% discount rate)	US$M	984
IRR	%	29.4%
Payback period (from the start of processing)	years	2

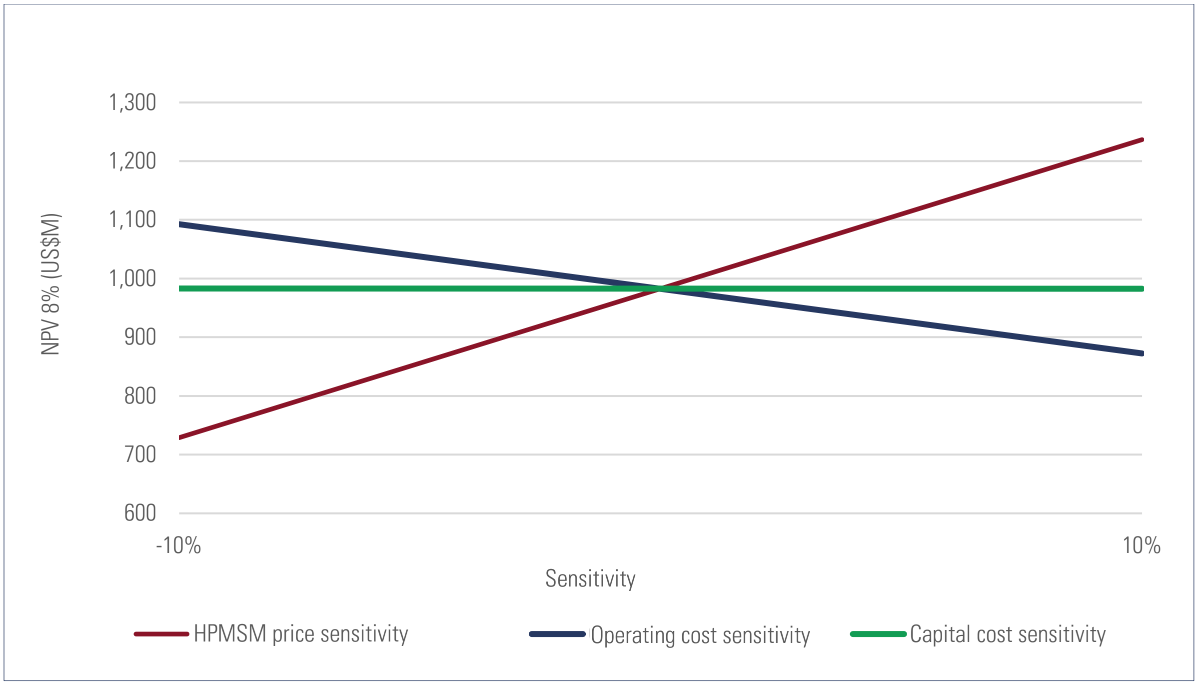
Table 22.9 presents the post-tax NPV at discount rates ranging from 0% to 15%

Table 22.8 Net present value post-tax summary

	Unit	Discount Rate					
		0%	5%	8%	10%	12%	15%
NPV	US$M	5,283	1,660	984	722	539	352

Figure 22.3 shows the sensitivity of the post-tax NPV at an 8% discount rate to product price, operating cost, and capital cost. The K.Hill Project's NPV is most sensitive to the product price, which can also be a proxy for grade and recovery. The Project is less sensitive to operating and capital costs.

Figure 22.3 Post-tax net present value sensitivity



22.5 Upside case financial results

An upside case was designed at a higher processing production rate to advance revenue and decrease the unit cost of production (Section 24.1). The earlier and increased cashflow would improve the Project's NPV and IRR.

The upside case is characterised by three strategic choices:

- The processing plant will double in capacity (200 kt/a to 400 kt/a)
- The increase in capacity will be achieved by duplicating the base case processing plant to create a twin-stream processing plant.
- The capacity increase will start after 5 years to limit initial capital costs and allow Giyani to conclude product qualification before committing more capital. Additionally, ramp-up and operational data from Phase 1 could be used to inform any design improvements in Phase 2.

For all other aspects of the Project, including processing technology, the upside case and base case are the same.

Therefore, the upside case mine schedule supports a processing plant production schedule that ramps up to 200 kt/a in Years 1 to 4 (Phase 1), followed by a ramp up to 400 kt/a during Year 5 of operations (Phase 2).

The upside case financial results, modelled on a 100% equity basis, are summarised in Table 22.10.

The K.Hill Project PEA includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral

Reserves as they do not have demonstrated economic viability. The results of the K.Hill Project PEA represent forward-looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the PEA. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially, are presented in the body of this report under each relevant section.

Table 22.9 Upside case financial results

Metric	Unit	Upside case		
Post-tax project economics				
NPV (8% discount rate)	US$M	1,528		
IRR	%	33		
Undiscounted cumulative cashflow	US$M	5,463		
Project production		Years 1-5	Years 1-25	LOP
Total material mined	Mt	2.4	9.3	11.1
Average annual material processed	kt/a	203	360	364
Average processing plant feed grade	MnO (%)	19.0	15.6	15.2
Total HPMSM produced	kt	402	2,958	3,559
LOP	years	5	25	31
Net pricing assumptions				
Average realized HPMSM price (Years 1 to 5)	US$/t	3,656		
Average realized HPMSM price (Years 6 and onward)	US$/t	3,787		
Capital cost				
Total initial capital cost	US$M	284		
Total additional capital expenditure	US$M	208		
Total sustaining capital cost	US$M	23	205	245
Project cash flow				
Total revenue	US$M	1,472	11,160	13,426
Total operating costs (including royalty)	US$M	677	5,024	6,120
Total EBITDA	US$M	795	6,136	7,306

23 ADJACENT PROPERTIES

There are no adjacent properties.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Alternative development plan for the K.Hill Project

Giyani intends to develop the K.Hill Project as an integrated mining and processing operation to manufacture a refined lithium-ion battery cathode precursor material, HPMSM, used in battery recipes that are deployed in electric vehicles. CPM (2022) reports that demand for lithium-ion batteries used in electric vehicles is projected to grow by 25% annually between 2021 and 2031 (Section 19). CPM demonstrates that the limited number of new sources of HPMSM supply forecast to come onstream during this period will result in a projected supply deficit of 726 kt/a on a contained metal basis by 2031. Furthermore, demand for HPMSM in lithium-ion batteries will result in a nearly 30-fold increase between 2021 and 2036, reaching 1.8 Mt on a contained metal basis, and up to 4.5 Mt by 2050.

As a result, Giyani investigated an upside case, designed to exploit this growing supply gap, through a higher processing production rate to advance revenue and decrease the unit cost of production. The earlier and increased cashflow would improve the Project's NPV and IRR.

A base case sensitivity analysis indicates that the K.Hill Project is relatively insensitive to capital cost (Section 22) and offers a compelling return on investment.

The upside case is characterised by three strategic choices:

- The capacity of the processing plant will double.
- The increase in capacity will be archived by duplicating the base case processing plant to create a twin-stream processing plant.
- The capacity will increase after 5 years. This approach will limit initial capital costs and allow Giyani to conclude product qualification before committing more capital. Additionally, ramp-up and operational data from Phase 1 could be used to inform any design improvements in Phase 2.

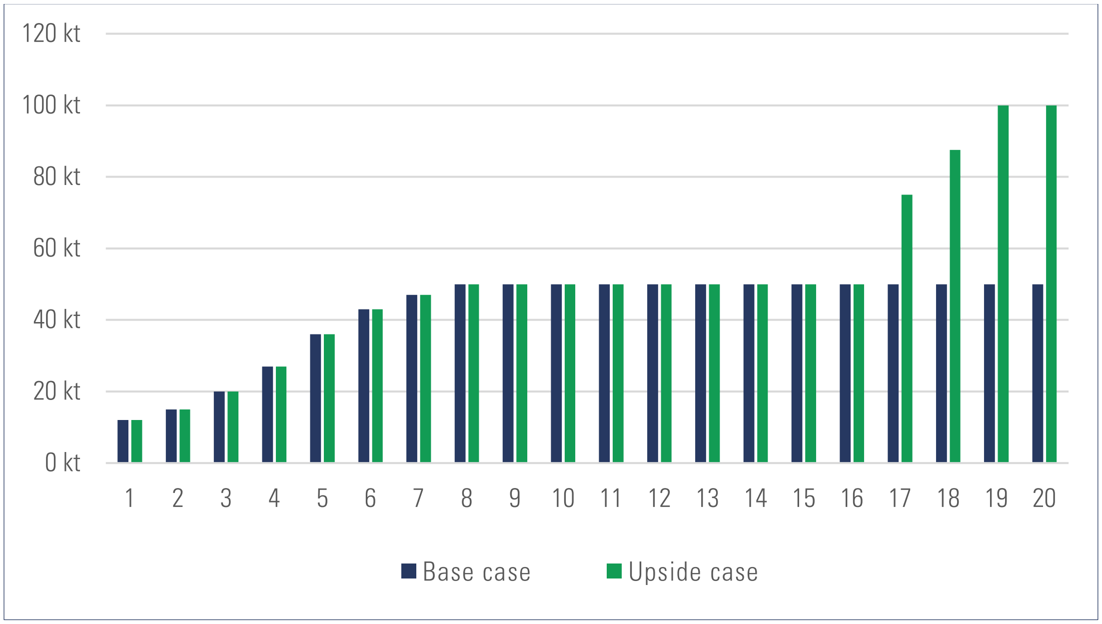
For all other aspects of the Project, including processing technology, the upside case and base case are the same.

The following subsections provide a summary of the upside case mining schedule and describe the implication on the Project components, including mining, processing, infrastructure, tailings management, environmental and social impact, and cost estimates.

24.1.1 Mine plan

An upside case mine schedule was prepared using the base case open pit design and mineral inventory reported in Section 16. This mine schedule supports a processing plant production schedule that ramps up to 200 kt/a in Years 1 to 4 (Phase 1), followed by a ramp up to 400 kt/a during Year 5 of operations (Phase 2). The ramp-up schedule for Phases 1 and 2 are shown in Figure 24.1.

Figure 24.1 Processing plant ramp-up schedules for the base case and upside case (Years 1 to 5, by quarter)



The upside case mine schedule total rock movement will peak around 5.5 Mt/a in Year 10. This rate will be maintained for 11 years before it starts to decline over the last 9 years of operations. The total LOM will be 29 years.

The processing plant feed grade profile will be similar to that of the base case, with a gradual decline over the LOM. As with the base case, the high grading strategy will help to raise the feed grade over the first half of the LOM, at which point lower-grade material will be reclaimed, as required, to maintain a processing plant feed of 400 kt/a.

Fluctuations in material exposure are controlled in the schedule by maintaining a large low-grade stockpile. It may be possible to further optimise this strategy by modifying the operational cut-off grade or changing the design of the open pit stages; however, at this preliminary stage, it is reasonable to assume that a high grading strategy in conjunction with a ramp up in production rate to 400 kt/a would indicate an improved project valuation.

The peak equipment requirements (Year 10) were kept similar to those of the base case by increasing the articulated dump truck size from 30 t to 40 t and increasing the excavator bucket size for waste from 3.8 m^3 to 5.5 m^3. Therefore, the total number of installed units required during the first 4 years of production will be less than that of the base case, and it can be expected that there will be some operating unit cost savings. For this evaluation, the unit mining costs were assumed to be the same as those of the base case.

It is also assumed that no further mining capital will be required during Phase 2.

Figure 24.2 Upside case life-of-mine schedule

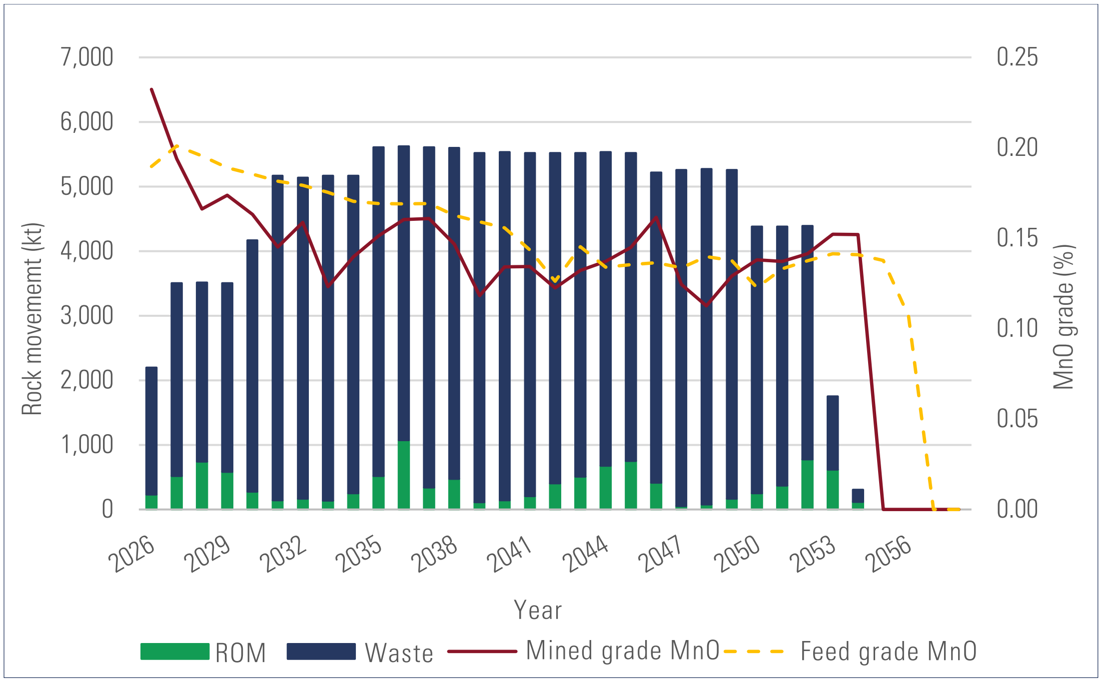
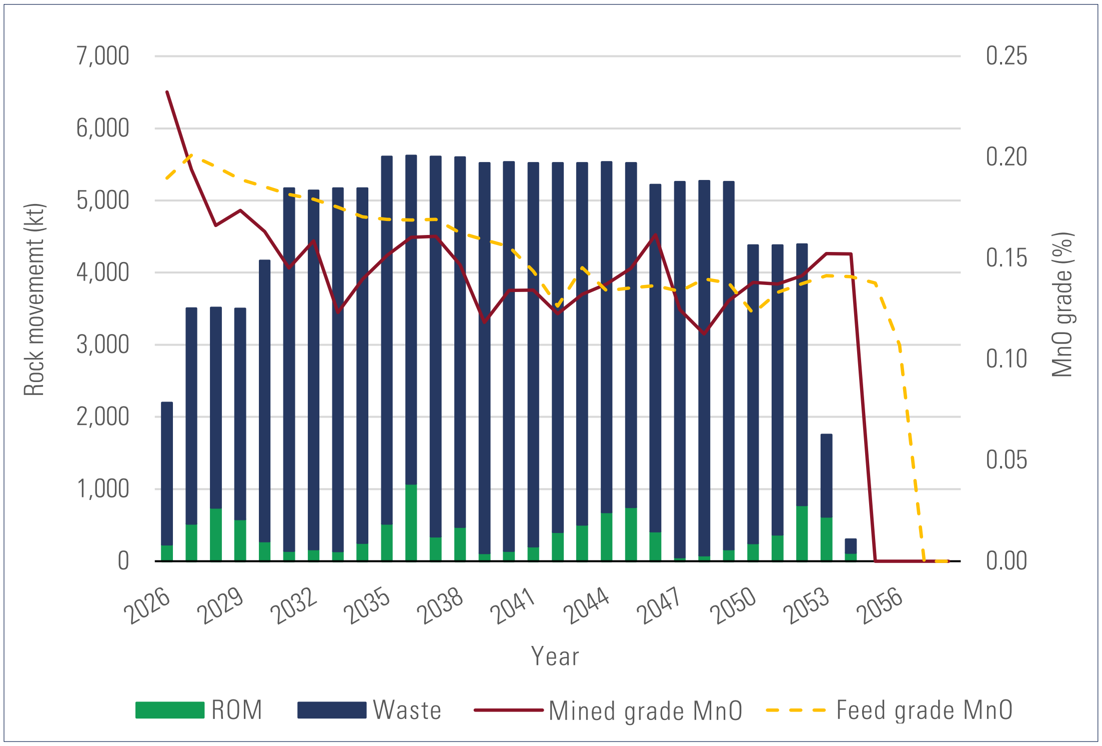


Figure 24.3 Upside case processing plant feed schedule



24.1.2 Processing and infrastructure

A duplicate processing plant for Phase 2 will be constructed alongside the initial processing plant, in such a way that they can share the crushing plant. The crusher day-shift-only operation is slightly oversized for Phase 1; therefore, the Phase 2 operation can easily be handled with minor modifications to the crushing plant and moving to a 2-shift operation. Each of the two mill plants will be fed from a dedicated crushed material bin, providing a buffer to account for the difference in availability of the dry and wet plants. All processing plant infrastructure and facilities, including reagent storage and handling, will be duplicated.

The Phase 2 capital cost required for processing will be approximately equal to that of the Phase 1 capital cost amount, except for the use of a common crushing plant. Existing infrastructure, such as the administration building, changerooms, workshops, warehouses, and gatehouse, will be mostly sufficient to accommodate the increase in capacity. The upside case mining equipment will be similar to the base case but will use slightly larger equipment, and identical processing streams will offer the opportunity for operational rationalisation.

The direct capital cost for processing and infrastructure for Phase 1 and Phase 2 will be US$122 M and US$103 M, respectively.

Base case unit process operating costs were used to estimate the upside case equivalent. Unit process operating cost mostly vary with production rate, leading to constant or almost constant unit process operating costs, which include energy, labour, laboratory, and water treatment cost areas. With an increase in the production rate, larger proportions of fixed costs for maintenance, solar plant, and tailings management lead to a reduction in unit process operating costs.

Unlike the rest of the operating cost areas, the raw materials unit cost is independent of the production rate and only a function of the mineralogy of the material being treated. In general, higher grade and associated contaminants will lead to an increase in reagent consumption. For this reason, the average raw material unit operating cost is higher for Phase 1 than for Phase 2. The average upside case unit process operating cost is US$594/t HPMSM for Phase 1 and US$489/t HPMSM for Phase 2, as presented in Table 24.1.

Table 24.1 Upside case process unit operating cost (US$/t processed)

Cost area	Phase 1 200 kt/a	Phase 2 400 kt/a
Energy	64.0	64.0
Labour	8.8	8.7
Laboratory/quality control	0.7	0.7
Water treatment	18.7	18.7
Maintenance	76.5	55.2
Tailings management	1.8	1.4
Solar plant	0.1	0.0
Raw material	423.4	339.8
Total	**594**	**489**

24.1.3 Tailings management

The final size and total capital cost of the TMF for the upside case is mainly a function of the mining inventory and therefore very similar to the base case. The upside case will require a higher frequency of downstream raises to accommodate the larger tailings production rate. This will compress the capital costs over the shorter LOM.

Table 24.2 Tailings management facility design criteria

Description	Unit	Criteria	Source
Total LOM throughput	Mt	23.3	Tetra Tech
LOM	years	Base case: Years 1 to 5 Upside case: Years 6 to 31	Axe Valley Mining
Annual production rate	t/a	Base case: 461,880 Upside case: 826,040	Tetra Tech
Total required capacity	Mm^3	16.2	Calculated
Tailings transportation method	-	Trucked in tankers	Knight Piésold design
Tailings deposition method	-	Direct discharge into impoundment	Knight Piésold design
Tailings density	t/m^3	1.44	Adopted from SRK and Tetra Tech (2022)
Crest width	m	8	Knight Piésold design
Embankment slope	-	1V:3H	Knight Piésold design
Tailings storage method	-	Within waste rock impoundment	Knight Piésold design

24.1.4 Capital cost estimate

The total capital cost for the upside case of US$490M (Table 24.3) consists of the Phase 1 initial capital cost, which is identical to the base case initial capital cost, followed by a lower Phase 2 expenditure to establish a second process stream.

An allowance of US$2M has been made for off-site infrastructure in Phase 2, for items such as new access roads, road upgrades, and upgrading of any off-site logistics facilities to handle the larger operation. Project indirect costs and construction overhead costs will be reduced from the base case costs in relation to the total capital cost. The owner's cost is assumed to be the same as the base case.

Table 24.3 Upside case capital cost (US$M)

Cost area	Phase 1 (200 kt/a)	Phase 2 (400 kt/a)
Mining	2.0	0.0
Processing	94.1	91.9
Infrastructure and services	27.8	11.5
TMF	9.2	0.0(1)
Off-site infrastructure	9.3	2.0
Indirect costs	48.8	34.1
Construction overhead costs	20.7	12.1
Owner's cost	8.2	8.2
Total initial capital cost (excluding contingency)	**220.1**	**159.8**
Contingency (28%)	62.5	47.9
Total initial capital cost (including contingency)	**282.7**	**207.7**

Note:
(1) The TMF capital cost is spread over the LOM after the initial expenditure.

24.1.5 Operating cost estimate

A breakdown of the upside case unit operating cost is presented in Table 24.4.

Table 24.4 Upside case unit operating cost

Cost area	Unit	Phase 1 (200 kt/a)	Phase 2 (400 kt/a)
Mining - overheads	US$/month	200,000	200,000
Mining - mineralised material	US$/t mined	3.00	3.00
Mining - waste	US$/t mined	2.50	2.50
Mining - rehandling	US$/t moved	1.00	1.00
Processing	US$/t processed	594	489
TMF	US$/t processed	1.36(1)	1.36(1)
G&A	US$/t processed	12.61	8.44

Notes:
(1) Based on US$0.45/m^3 of tailings, a tailings density of 1.437 t/m^3, and an average tailings to processing plant feed ratio of 2:1.

24.1.6 Financial results

The upside case financial results, modelled on a 100% equity basis, are summarised in Table 24.5.

The K.Hill Project PEA includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. The results of the K.Hill Project PEA represent forward-looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the PEA. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially, are presented in the body of this report under each relevant section.

Table 24.5 Upside case financial results

Metric	Unit	Upside case		
Post-tax Project economics				
NPV (8% discount rate)	US$M	1,528		
IRR	%	33		
Undiscounted cumulative cashflow	US$M	5,463		
Project production		Years 1-5	Years 1-25	LOP
Total material mined	Mt	2.4	9.3	11.1
Average annual material processed	kt/a	203	360	364
Average processing plant feed grade	MnO (%)	19.0	15.6	15.2
Total HPMSM produced	kt	402	2,958	3,559
LOP	years	5	25	31
Net pricing assumptions				
Average realized HPMSM price (Years 1 to 5)	US$/t	3,656		
Average realized HPMSM price (Years 6 and onward)	US$/t	3,787		
Capital cost				
Total initial capital cost	US$M	283		
Total additional capital expenditure	US$M	208		
Total sustaining capital cost	US$M	23	205	245
Project cash flow				
Total revenue	US$M	1,472	11,160	13,426
Total operating costs (including royalty)	US$M	677	5,024	6,120
Total EBITDA	US$M	795	6,136	7,306

24.1.7 Conclusions

Accelerating HPMSM production improves the K.Hill Project economics, as illustrated by the increase in post-tax NPV (8%) from the base case value of US$984M to US$1,528M for the upside case (Table 24.5). A larger operation is more cost-effective and offers Giyani the opportunity to secure a larger portion of the non-Chinese HPMSM supply market.

The capital cost premium of a twin-stream processing plant, as compared to a single-stream double capacity processing plant, is the offset of the benefit of choice – essentially an option cost: the option to select the timing of the second processing plant construction (if at all), the option to include process improvements from the Phase 1 operation, and the option to run a single stream if the other processing plant is unavailable. An additional benefit is the ability to use cashflow from Phase 1 operations to fund Phase 2 capital costs.

24.2 Base case project execution plan

The Project Execution Plan (PEP) sets out the method in which the scope presented in this Technical Report will be designed, procured, built, and commissioned. The PEP includes a description of activities to be undertaken leading up to project execution, execution scope, strategy, schedule, project governance, and the approach to be taken to comply with health, safety, and environment (HSE) guidelines and best practices.

24.2.1 Post-Preliminary Economic Assessment study activities

Giyani will undertake the following post-PEA activities to continue to optimise aspects of the K.Hill Project, develop opportunities, and mitigate residual risk. Key work streams will include the following:

- Demonstration plant: A large-scale demonstration plant that emulates the continuous process of the proposed full-scale K.Hill commercial plant is under construction in South Africa. The demonstration plant will validate the process flowsheet, mitigate commercial processing plant risk, and facilitate off-take qualification.
- Prefeasibility phase: The Project scheme described in the PEA will be developed in a prefeasibility phase that will include laboratory and field work, trade-off studies, etc., as recommended in Section 26.
- Feasibility phase: The outcome of the prefeasibility phase will inform an optimised K.Hill Feasibility Study. The optimised Feasibility Study will include results from the operation of the demonstration plant.

24.2.2 Project scope

The scope of the K.Hill Project execution, after receipt of all necessary permits and approvals from authorities, will include the following areas:

- Mine: All vegetation will be cleared and any topsoil will be removed from the open pit area, WRD, and stockpile sites. Production with an owner-operated fleet can then start immediately.
- Processing plant: All vegetation will be cleared and the pads prepared for processing plant construction. The processing plant will be constructed as described in Section 18, including all foundations, concrete, steel, equipment, electrical, and instrumentation and access.
- TMF: Site clearing and grubbing and removal of 0.5 m of topsoil will take place. Foundation will be prepared, perimeter drains constructed, and the geomembrane installed. The pond will be constructed and the return water pump and return pipeline will be installed along the tailings haul road.
- On-site infrastructure: All vegetation will be cleared and pads will be prepared for building erection. Components of infrastructure will be constructed, as described in Section 18, including all foundations, concrete, steel, equipment, and power supply. Key components include:
    - perimeter fence, gatehouse, and weighbridge
    - administration, warehouse, and workshop buildings
    - WTP
    - site roads, parking, and laydown areas

- ✦ main substation and power distribution to switch rooms
- ✦ explosive storage facility
- ✦ fuel farm

- Off-site Infrastructure:
  - ✦ main access road from the B202 highway to the main gate
  - ✦ water boreholes and connecting pipeline to feed into the WUC water network and water pipeline to extract water via the WUC pump station to a termination point at the gate
  - ✦ solar plant
  - ✦ high-voltage power supply to be provided by BPC

- Existing infrastructure relocation: The K.Hill Project is located on a brownfield site that is disturbed by old workings, a communications tower (BOCRA), WUC water reservoirs, concrete strip roads, lots of tracks, and evidence of waste dumping. Giyani will work with the owners and local authorities to relocate the following existing infrastructure from the site:
  - ✦ Two water reservoirs that are part of WUC's water supply system to Kanye will be removed. A site has been identified for construction of the new reservoirs. The reservoirs will be built and commissioned before the existing reservoirs are demolished.
  - ✦ A BOCRA spectrum monitoring tower on top of the hill will be dismantled and relocated to a site north of Kanye.
  - ✦ Two active low-voltage three-phase power lines cross the Project site and will have to be rerouted.

24.2.3 Project execution strategy

Project objectives

The K.Hill Project will be executed to achieve the following objectives:

- compliance with the management and monitoring plans set out in the EIA, conditions that the Botswana DEA will apply to the environmental authorisation, and other conditions stipulated in the Mining Licence
- compliance with HSE management plans, monitoring plans, policies, and procedures
- on-time and on-budget Project execution

Project delivery model

The K.Hill Project will be delivered using an EPCM project delivery model, whereby the Project scope will be divided into a number of work packages. The engineering consultant firm that is selected as the EPCM contractor will be responsible for providing all aspects of EPCM for certain work packages, but some work packages may also be self-performed and managed by the owner's project team.

Stakeholders

Stakeholder engagement will be managed through a Stakeholder Engagement Plan (SEP) that applies to the K.Hill Project LOM (i.e., from construction to rehabilitation and closure). The SEP will incorporate and build on the consultation associated with the EIA. Stakeholder engagement will comply with legislative and regulatory requirements and conform with international standards.

Engineering

The engineering design will meet technical, regulatory, and functional requirements. After completing the optimised Feasibility Study and undertaking value engineering, the design for the K.Hill Project will move into basic and then detailed engineering.

Long-lead items will be specified for procurement during basic engineering, and general equipment and instrumentation specifications will be specified for procurement at the start of detailed engineering. Readily available off-the-shelf equipment will be selected, where possible, taking care to consider maintenance and spare requirements throughout the LOM.

Even though Giyani will seek to use local fabricators, it is anticipated that most of the equipment and bulk materials will be sourced from Southern African vendors and fabricators or imported via South Africa. Processing plant and infrastructure components will be preassembled in road transportable units, as much as possible, to minimise erection time on site.

Various engineering firms will be employed to design each of the major speciality areas, namely mine, processing plant, and general infrastructure; TMF; solar plant; and external infrastructure.

Procurement

Procurement will be managed on behalf of Giyani by each of the engineering firms. The following general principles will be applied:

- Pre-qualification: Engineers, consultants, suppliers, and contractors will be pre-qualified following HSE, legal, and financial due diligence. Shortlisted vendors will have a demonstrated track record and the capacity to deliver and service Southern Africa.
- Complete life-cycle procurement: Equipment and spares will be sourced from vendors that can continue to provide parts and services throughout the LOM. Local component distribution warehouses will alleviate the site inventory and associated working capital. Rationalisation of the vendor list to encourage commonality in spares and more effective supply chains will be supported by a standardised criteria in the engineering phase.
- Tender process: A request to tender will be sent to a minimum of three shortlisted companies. The tender procedure will be fair and transparent with every step documented and auditable.
- Specialist vendor packages: The crystallisation and effluent treatment plants are examples of bespoke packages designed by specialist vendors. To secure their services and finalise the process design during basic engineering, these procurement packages will be advanced at the outset.

Contracting strategy

Giyani will distribute the works over several key contracts selected following a formal tender process. Contractors will be identified and shortlisted as per the pre-qualification process; contracting with a Botswana legal entity will be considered an advantage.

Commissioning

The owner's EPCM team will have a dedicated team to facilitate commissioning and handover to operations. Commissioning will be undertaken by a specialist consultant on hydro-metallurgical systems, who will be integrated into the owner's EPCM team and brought on board during the pre-commissioning execution phase to prepare and finalise the commissioning plan. This team will be responsible for managing the equipment vendor representatives.

24.2.4 Project execution schedule

Sequence

The Project will undergo the design and construction of major components, for example the processing plant and TMF. Construction in each of these areas will go through typical phases of engineering, procurement, delivery, fabrication, and construction. These phases will be sequential but can partly overlap to compress the schedule. The start of every phase will be represented by a milestone in the Project execution schedule, which will serve as a decision gate to ensure the risk assessment for the Project is updated and that conditions are met to commence with the next phase.

A basic sequence of major activities underpins the Project execution schedule. These activities include, in approximate chronological order:

- finalising the Project Description and basic engineering scope of work following the completion of value engineering
- completing a tender enquiry and subsequent appointment of the key technology partners for the design and supply of specialist vendor packages
- completing a tender enquiry and appointing the EPCM contractor
- completing a tender enquiry and appointing the contractors to relocate the existing site infrastructure, including the water reservoirs and the high-voltage power supply
- specifying and completing the final material characterisation test work
- completing the processing plant basic engineering phase to produce the following typical deliverables:
  - process design package, including process flow diagrams, mass balance, process design criteria, and control philosophy
  - Project specifications, including general, mechanical, structural, civil and earthworks, electrical, control and instrumentation, and site conditions
  - discipline design criteria
  - mechanical equipment list
  - long-lead item specifications and associated procurement packages
  - general arrangements
  - AACE® International Class 2 capital cost estimate and subsequent control budget

- finalising construction contracts
- commencing with the specification and procurement of equipment
- commencing detailed engineering
- mobilising contractors to site and commencing site preparation
- commencing construction
- commissioning

Schedule

Key milestone dates for the K.Hill Project are shown in Table 24.6.

Table 24.6 K.Hill Project key milestone dates

Milestone	Date
Start basic engineering	Q3 2024
Start construction	Q2 2025
Start mine pre-stripping	Q4 2026
Start processing plant commissioning	Q4 2026
First commercial production	Q1 2027
End of ramp up	Q2 2029

It should be noted that Environmental and Social Management Plans (ESMPs) and Giyani's policies and procedures will be applied throughout the LOM. Specific ESMPs will be developed before construction begins, in readiness for implementation from the outset of construction (e.g., Emergency Preparedness and Response Plan; Water Management Plan; Alien and Invader Species Management Plan).

24.2.5 Project governance

Steering committee

The K.Hill Project execution will be overseen by a steering committee that will consist of senor members of Giyani's management team and designated consultants. A combination of internal and external technical specialists will provide ad hoc support. The steering committee will have regular meetings and adjust their meeting frequency to suit.

Owner's project team

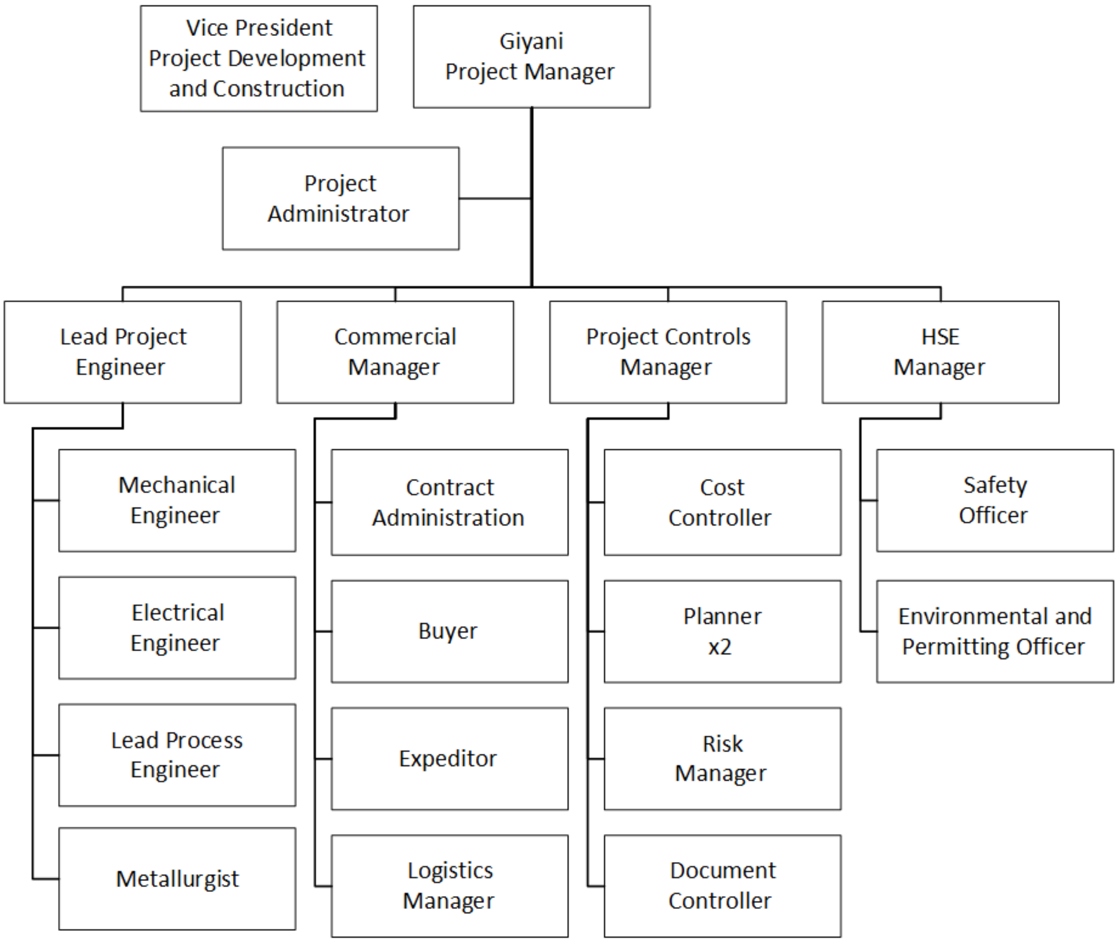
A dedicated owner's project team, comprising engineering and construction professionals, will be established under the leadership of an owner's project manager. Once the Project is handed over, the project team will be demobilised.

The owner's project team will be integrated with several functions of the operations team, such as tax, finance, legal, information technology, environmental, health and safety, sales, and production. Project participants of these functions report to department heads inside the operations team. Some members of the Giyani operations team may be seconded to the project team.

The project team will prepare a PEP and submit it to the steering committee for approval. In general, the project team operates autonomously within the agreed plan. All major decisions required to deliver the Project, as well as deviations from the approved plan, will be presented to the steering committee for review and approval. The status of the Project will be communicated formally in a monthly report and on an ad hoc basis for major upset conditions.

The proposed owner's project team is shown in Figure 24.4 and comprises Giyani project staff supplemented with external consultants.

Figure 24.4 Owner's project team



24.2.6 Health, safety, and environment

HSE management plans, monitoring plans, policies, and procedures are fundamental for delivering the Project. Management and monitoring plans will be included in the EIA, and implementation by Giyani will be a condition of environmental authorisation, as will the right for the authorities to undertake site visits and audits. Undertaking audits (internal, third party, and those undertaken by the authorities) will be important.

A safety induction will be mandatory for anybody who visits or works on the construction site. The construction site will comprise three zones, and the level of induction and required personal protective equipment will be geared to each of them:

- green: the visitor area with meeting rooms, which is freely accessible
- orange: the access-controlled area with construction camp, laydown areas
- red: the construction site itself

Health and safety

Contract conditions imposed by Giyani will require that all contractors be required to provide ESMPs, method statements, and standard operating procedures for all works for which they are responsible; these plans will cover the period from mobilisation through to demobilisation. The documents will describe the scope of work, mitigation and management, number of workers and skills, safety measures, equipment, and risk assessment. All documents will need to be approved by the steering committee prior to mobilisation.

Environment and community

It is a standard condition of environmental authorisation that the ESMPs presented in the EIA are implemented for each phase of the Project (construction, operations, rehabilitation and closure, and post-closure monitoring). Giyani will develop other ESMPs and procedures before construction begins to manage all aspects of the overall environment, including, but not limited to:

- surface and groundwater monitoring
- air and noise monitoring
- waste (non-mining and mining waste)
- materials management procedure, including fuels and hazardous materials (chemical register, transport, handling, usage, storage, disposal, spillage)
- topsoil management (stripping, maintenance of quality, contaminated soil)
- emergency preparedness and response plan
- SEP
- grievance procedure
- community security, health and safety
- cultural heritage and restricted areas
- legal obligations (legal registers, auditing)

All ESMPs and procedures will be reviewed annually, at a minimum, when there are changes to the Project or in response to non-conformances. As the Project transitions from one phase to another, significant changes will occur and will warrant document revisions and updates.

25 INTERPRETATIONS AND CONCLUSIONS

25.1 Geology and Mineral Resource estimates

In February 2022, Giyani released an MRE for the K.Hill Project based on data from 115 RC and DD holes totalling 4,793 m of drilling. The updated MRE contained in the Technical Report was prepared using data from 187 RC and DD holes from all drilling campaigns conducted over the Project since 2018, totalling 10,710 m. This additional drilling included 40 step-out holes along strike into a previously untested, but mineralized, section to the south.

This collated and complete set of drilling data, together with a new density determination programme and updated optimization parameters for the constrained pit shell, resulted in the conversion of Inferred to Indicated Mineral Resources resulting in an overall increase in Indicted Mineral Resources of more than 300% and an increase of more than 100% in Inferred Mineral Resources.

The MRE reported has been restricted to all classified material located within an optimized pit shell.

Potential risks include:

- Average densities for each domain in the block model were taken from a small database of densities calculated from wax-coated core samples. Although of higher quality, the quantity of data present a risk to the K.Hill Project. The densities may not be entirely representative of the mineralised units. It is recommended that the wax-coated core densities continue during the next phase of exploration and are re-evaluated during the next MRE update.
- Channel structures interpreted in the base of the overlying CDB unit that erodes the upper Mn-shale horizon have been observed. These structures have been modelled where they are intersected, and any similar structures that may exist are expected to be smaller than the approximately 50 m drill spacing at the K.Hill Project. These structures may influence the interpretation and estimation in this area.
- Inferred Mineral Resources account for approximately 40% of the tonnage above the reporting cut-off. Inferred Mineral Resources have a lower level of confidence than that applied to Indicated Mineral Resources and are based on limited information and sampling.

Potential opportunities include:

- Mineralisation is open along strike and down dip in some areas. These areas represent an opportunity to increase the size of the Mineral Resource with step-out drilling.

25.2 Mineral processing

The mineralogical investigations indicate that most of the manganese is associated with the minerals cryptomelane and bixbyite. When subjected to a reducing leach using sulphur dioxide in sulphate media, manganese extraction is greater than 90%.

The metallurgical investigations also confirm the findings of the mineralogical studies, where the optimal grind size is estimated to be a P_{80} of 150 µm, yielding a manganese extraction of 97%. The manganese losses during the purification process were found to be 0.5% during the precipitation stage and 4.0% in the fluoride polishing stage; together with a 97% recovery by precipitation from the crystalliser bleed recycle stream yielded an overall manganese recovery of 88.5%.

Potential risks include the following:

- Metallurgical recovery from the processing plant has been estimated based on the laboratory test work results, with appropriate allowances for likely processing plant operating conditions. These recoveries should be verified by further laboratory and optimisation studies.
- The mine schedule shows that the mineralised material characteristics vary across the orebody, and may result in variations in grinding energy requirements over the LOM.
- The variability leach test work has shown differences in the manganese extraction between different lithologies; therefore, the overall processing plant recoveries can also be expected to vary during the LOM.
- A prevalence of higher valency manganese in the processing plant feed will require increased reduction requirements and lead to higher sulphur dioxide consumptions.
- Highly variable potassium or iron levels in the mineralized material, or unexpectedly poor iron leaching, will cause increased impurities downstream, requiring an increase in reagents for impurity removal.
- Redissolution of precipitated base metal sulphides owing to changes in pH, residence time, or both will impair product purity requiring additional crystalliser recycle and an increased crystalliser bleed stream, which in turn will increase reagent costs and decrease overall manganese recovery.
- Insufficient calcium and magnesium removal in the fluoride polishing step is due to an insufficient addition and poor control of hydrofluoric acid addition.
- The solid/liquid separation duties are much lower than those indicated in the test work, leading to undersized equipment duties and limited processing plant throughput.
- The crystalliser requires more operational intervention than expected, leading to more frequent stoppages than expected. In turn, the buffer capacity may be inadequate and full processing plant stoppages may be necessary.
- The WTP recovery is lower than anticipated, causing an increase in the processing plant water requirement and additional disposal losses.
- Overall manganese recovery is lower due to recycled manganese hydroxide not fully dissolving into solution.

Potential opportunities include the following:

- Improving the iron extraction to reduce the amount of ferric sulphate required for jarosite precipitation.
- The use of the recycled manganese hydroxide as a neutralising agent will decrease the barium hydroxide requirement and decrease operating costs.
- The fluoride polishing kinetics can be improved by adding hydrofluoric acid over multiple stages. This will lower the calcium and magnesium in solution, decreasing the bleed stream size which will lead to higher overall manganese recovery and a decrease in operating costs.

- A selection of a fluoride adsorption media with a higher loading capacity will decrease the size of the activated alumina columns, which could lead to a lower water requirement. This will also decrease the size of the WTP and the amount of water discharged for dust suppression.
- Salting out of manganese sulphate as an alternative to the fluoride polishing step will obviate the need for barium hydroxide and hydrofluoric acid and eliminate the activated alumina columns from the purification scheme. This can potentially lead to a significant decrease in both capital and operating costs by ensuring the feed to the crystalliser is fully saturated with manganese sulphate.
- Alternative purification and precipitation steps involving alternative manganese salts and compounds may hold promise for even further cost reductions.
- The mill media cost and other processing plant consumable costs can be negotiated with suppliers based on long-term purchase arrangements, leading to potential savings in processing plant operating costs.

25.3 Mining methods

Based on the work undertaken to date, the following key conclusions are made regarding mining at the K.Hill Project:

- The pit optimisation indicates that the US$3,800/t HPMSM pit shells (RF 0.6 to 0.7) are considered optimal and have been selected to maximise mineralised inventory, while maintaining a sufficient profit margin to ensure profitability should prices reduce or costs increase.
- The final pit design contains a mineral inventory, including Inferred material, of 11.1 Mt at a diluted grade of 15.2% manganese oxide, above a marginal cut-off of 7.4% manganese oxide. The pit design inventory includes a total contained metal quantity of 1,691 kt manganese oxide.
- The assumed pit design parameters are supported by the geotechnical assessment implementing a 41° overall slope, including a 10 m bench height, 70° batter angle, and 7.6 m catch berm, which are planned to be mined out in 2.5 m or 5 m flitches.
- With a planned processing plant throughput rate of 200 kt/a, the LOM plan has 49 years of mining with another 8 years of stockpile rehandling of low-grade material, which results in a total LOP of 57 years.
- The planned mill feed rate of 200 kt/a will be achieved after an initial ramp-up period of 2 years. During the first two quarters or Year 1, mainly lower-grade material (<10% manganese oxide) will be fed to the processing plant during the commissioning period. Thereafter, the high grading policy takes effect, and lower-grade material is stockpiled.
- The WRDs will be constructed east of the open pit, with a total capacity of 124 Mt. Where possible, the weathered material will be encapsulated by harder waste material obtained deeper in the mine.
- The overall slopes of the WRDs have been designed at 28°, which is appropriate for WRD rehabilitation as part of progressive mine closure. In-pit dumping has not been included in the mine plan, but this may be an option later in the LOM once the northern area of the open pit has been mined out.

- A low-grade stockpile will be built north of the processing plant, allowing for easy rehandling to the ROM pad, with a peak quantity of approximately 2 Mt. Lower-grade material (<9.0% manganese oxide) will be reclaimed toward the end of the LOM.
- The mining method is based on the use of backhoe excavators with a 3.8 m^3 bucket capacity, which are appropriately sized for the scale and selectivity requirements for this operation. Haul trucks with 30 t capacity will be matched to the planned loader bucket size.
- Assuming the Project will use a contract mining operation, the average mining costs have been estimated at US$2.5/t mined for waste and the US$3.0/t for ROM mineralized material. These costs have been benchmarked against previous studies for Giyani and contract quotes for other mines in the region.
- The load-and-haul fleet requirements at peak production (3.5 Mt/a of rock movement) are estimated at 4 excavators and 14 trucks, assuming a single 12-hour shift operating 7 days per week. This is based on overall utilisation factors of 75% and 65%, respectively, and that planned maintenance can be undertaken by the mining contractor on the off shift, as required.
- The mining contractor will be responsible for all activities in the open pit, including drill and blast, load and haul, road maintenance, equipment maintenance, and WRD management. The owner will be responsible for overall management and technical support.

25.4 Geotechnical considerations

Available data has been reviewed and used to support geotechnical assessment suitable for PEA level of analysis.

Stability analyses on representative sections of the provided pit design indicate that the sections meet DAC. As such, the pit design is considered appropriate for PEA.

25.5 Hydrological considerations

25.5.1 Mine inflows and dewatering

The previously completed Technical Report (SRK and Tetra Tech 2022) undertaken on the original MRE found that the final elevation of the pit floor would be above the water table; therefore, there would be no groundwater inflows into the open pit. However, the updated mining sequence, based on the updated MRE (Section 14) will extend significantly deeper than the open pit proposed for the original MRE, and it is anticipated that groundwater will be intersected during the LOM. Not only will there be a component of groundwater inflow into the open pit there will also be a larger volume of surface water runoff flowing into the new proposed open pit (due to the larger surface area of the updated MRE). Using the layout of the open pit at the end of LOM (year 50), it is calculated that surface runoff into the open pit area is expected to be in the order of 85,850 m^3/day in the event of a 1:100-year return event storm (equivalent to 3,577 m^3/hour). These calculations do not take into consideration possible backfilling of the open pit.

Groundwater inflow volumes into the excavated Northern pit at the end of the LOM are expected to be in the range of 430 m^3/day to 690 m^3/day. Groundwater inflows into the Southern satellite pit are expected to be in the order of 40 m^3/day to 50 m^3/day.

25.5.2 Water supply

Water supply for mining operations is to be sourced from a combination of the existing WUC network and supplemented by groundwater extracted by Giyani via a new proposed wellfield. Water supply options for the Project are well documented and understood and are considered to be at an appropriate level of confidence for a PEA study.

25.5.3 Water Infrastructure

A number of new pieces of infrastructure will be required for the K.Hill Project. Water-related infrastructure requirements, including potable water storage tanks, a new WTP, and the associated systems required to treat and distribute processing plant water, fire water, and potable water are well documented and are considered to be at an appropriate level of confidence for a PEA study.

With the updated open pit expected to breach the groundwater level, and mine dewatering being required, water infrastructure to manage the open pit dewatering water will be required. This will include the following:

- Pit dewatering: This can be done in-pit via a sump, or by using out-of-pit dewatering boreholes. Based on the expected relatively low aquifer transmissivity limiting the zone of influence of out-of-pit dewatering wells, and the relatively low volume of groundwater inflows into the pit being expected, it is recommended that dewatering takes place via an in-pit sump.
- Pollution control dam/return water dam: it is expected that the water pumped from the open pit will be routed to the process plant for use. Therefore, it is expected that the dewatered water will be pumped into a pollution control dam/return water dam, from where it will be absorbed into the water management system and pumped to the processing plant.

25.5.4 Water management plan

Based on a review of the available information it is evident that preliminary plans are in place to control surface water at the site, which is accompanied by appropriate water management designs. Clean and dirty water systems will be separated. Runoff and seepage rates from the WRD were calculated as part of this study (Section 18.9.3). The available information and studies undertaken to date in terms of water management is considered to be at an appropriate level of confidence for a PEA study.

25.6 Project Infrastructure

The site infrastructure layout was driven largely by the site topography, the open pit extents and blast radius, TMF, public roads, utilities corridors, and prevailing wind direction. All permanent infrastructure has been positioned outside the 250 m blast radius of all open pits, and the administration and maintenance areas are outside the 500 m blast radius of all pits.

Additional major infrastructure such as the explosives storage has been positioned in accordance with standard African mining practice. The fuel farm has been located on the corner of the ROM pad, so heavy and light vehicles both have access.

Site utilities, which include water, power, and communications, will run along side the public road from Kanye and will enter the site close to the processing plant. The water supply will be piped to the water treatment area, power will be distributed via two, 10 kV ring mains and communication masts will be strategically positioned around the site.

25.7 Tailings management facility

The TMF will be located southeast of the processing plant. Tailings material will leave the processing plant in the form of highly thickened tailings at an estimated 60% to 70% solids ratio. The filtered tailings will be trucked in tankers to the lined TMF.

To target zero discharge of process-affected fluids, the TMF will consist of a fully lined facility and a settlement pond in which surface runoff and seepage water will be collected.

The TMF embankment will be constructed in five phases using the downstream construction method. Embankment Phases 1 to 4 will consist of 10 m lifts and the final Phase 5 will consist of a 5 m lift. The embankment will be constructed of borrowed waste rock. An inter-cell bund will be constructed for decanting and tailings deposition.

An assessment of embankment slope stability was conducted as part of the PEA design to demonstrate that the TMF external slopes meet the accepted criteria set out in the CDA guidelines.

The TMF will be closed by installing a suitable cover that will cover the surface with vegetation to minimise water ingress and erosion of faces, while decreasing the potential for dust generation once the final raise has been constructed.

25.8 Environmental studies, permitting, and social or community impact

Conclusions drawn from the current EIA indicate that the K.Hill Project will cause environmental impacts, both negative (e.g., dust raised in working areas) and positive (e.g., employment opportunities and increased economic activity). Mitigation measures will vary with each phase of the Project, with limited additional impacts during operations.

Regular monitoring by Giyani will be required to make sure that the Project environmental standards are achieved (e.g., water, dust).

25.9 Capital and operating costs

The total estimated initial capital cost for the K.Hill Project is US$282.6M, including the contingency of US$62.5M. The estimate is inclusive of direct costs required to execute the Project and indirect costs associated with the design, installation, and commissioning of all Project facilities. Closure costs of US$8.4M have been estimated.

The total operating costs are estimated based on the mining rate, process design work, and the consumption of reagents. The total unit operating cost is estimated at US$579/t of mineralised material processed, which is made up of mining, processing, TMF, and G&A costs. Processing accounts for 89.4% of the total operating cost per tonne of mineralised material processed.

25.10 Economic analysis

The Project economics were analysed on a 100% equity basis, which demonstrates that the K.Hill Project provides a robust post-tax NPV at an 8% discount rate of US$984M and a post-tax IRR of 29.4% in real terms under the considered economic assumptions.

Based on the work carried out for this PEA, CSA Global has made the following conclusions:

- The K.Hill Project reflects a positive NPV of US$984M at an 8% discount rate with the assumptions made for input parameters.
- The K.Hill Project is highly sensitive to the HPMSM price but has lower sensitivities to capital and operating costs.

26 RECOMMENDATIONS

26.1 Geology and Mineral Resource estimates

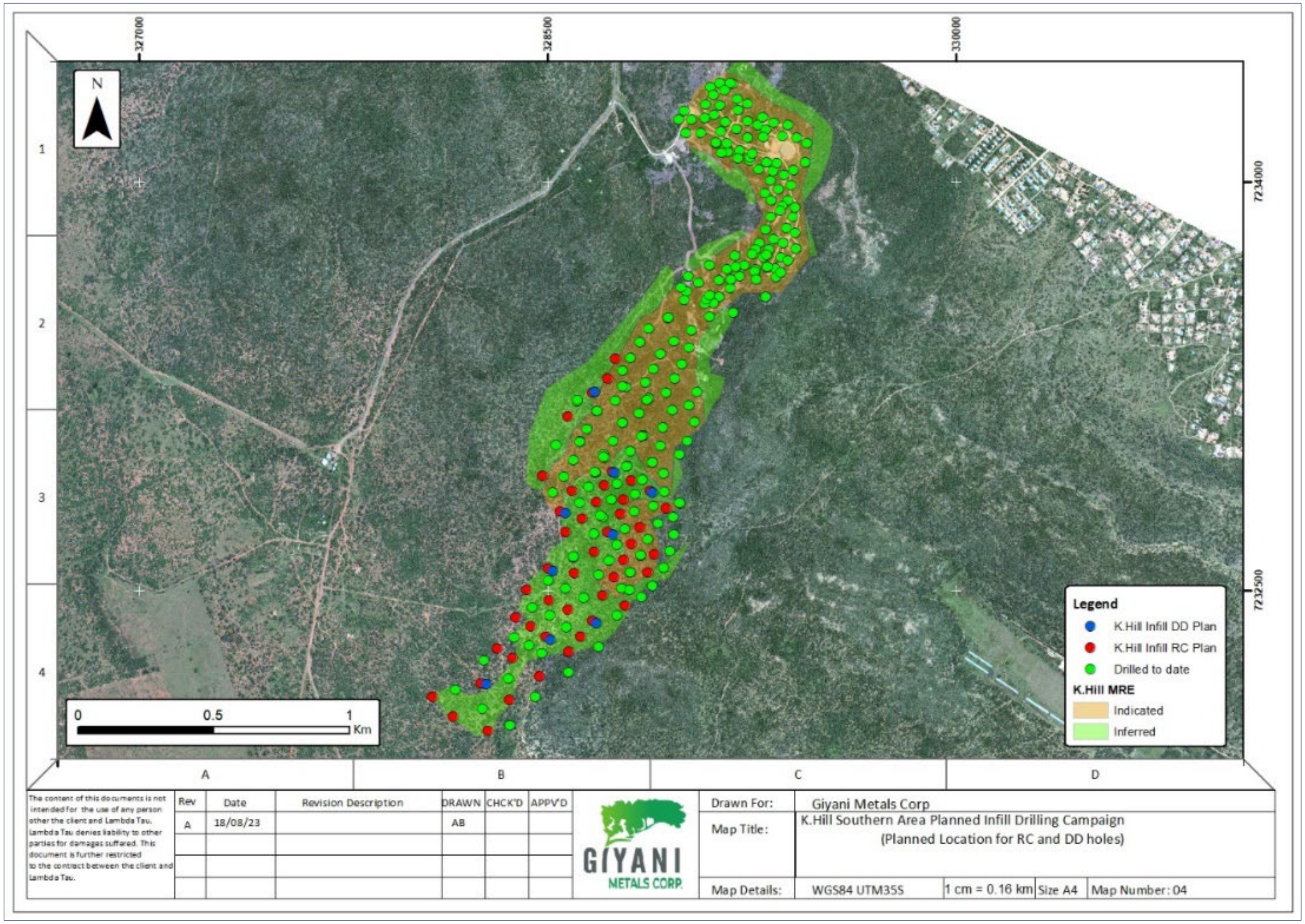
Areas of improvement were identified during the QP's site visit and subsequent Mineral Resource update:

- Average densities for each domain in the block model were derived from few data determined from wax-coated core samples. Although of higher quality, the quantity of data should be increased. The densities may not be representative of the mineralised units. Therefore, it is recommended that the wax-coated core density determinations continue during the next phase of exploration. These will be re-evaluated during the next MRE update.
- A detailed structural investigation should be conducted prior to further Mineral Resource updates to improve the Mineral Resource confidence. Angled drilling can be used to confirm fault interpretations, their position, and the impact, if any, on the mineralisation.
- It is reasonable to expect that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued infill drilling. Infill drilling is recommended to improve the Mineral Resource classification.
- The drilling program recommended aims at improving the geological confidence of the K.Hill deposit and allow for improving the portion of the current Mineral Resource classified as Inferred to Indicated. This will require an infill drilling program, which will decrease the drillhole spacing in the areas where the classification should be upgraded. A total of 55 holes are planned, 46 as RC and 9 as DD (Figure 26.1).
- A total of 4,454 drilling meters is planned (Table 26.1). Geochemical assays and density determinations using the Archimedes principle will add to the existing geological database and allow for a larger, more robust set of data to inform the geological model and future revisions of the MRE.

Table 26.1 Drilling plan

Drilling	Number of holes	Total meters
RC	46	3,725
DD	9	729
Total	**55**	**4,454**

Figure 26.1 Location of planned drillholes to be completed in the next phase of drilling



Notes:
Red dotes indicated planned drillholes. Blue dots indicate where DD holes will be twinned. Green dots indicate the collars for all existing drillholes at the K.Hill Project, RC and DD.
Source: Giyani

- The total expected cost for this program is US$737,651. Table 26.2 shows a summary of the expected costs.

Table 26.2 Total expected costs for the next phase of drilling

Item	Description	Unit	Quantity	Rate (US$)	Total (US$)
RC Drilling	All-in per meter cost for drilling, casing water, chemicals etc.	per meter	3,725	92	342,700
DD Drilling	All-in per meter cost for drilling, casing water, chemicals etc.	per meter	745	184	137,080
Analysis	XRF analysis at accredited laboratory	per sample	1,620	29	46,980
Geology	All-in cost for project management and geological services	per month	3	36,880	110,640
MRE update	All-in cost for geological modelling, MRE, and reporting	per unit	1	33,192	33,192
				Contingency (10%)	67,059
				Total	**737,651**

26.2 Mineral processing

A number of areas have been identified for further investigation and evaluation that are expected to confer both performance and economic benefits on the K.Hill Project, including:

- completing additional metallurgical tests to optimise and evaluate alternative methods to fluoride polishing for solution purification to reduce capital and operating costs; in particular, alternative methodologies for the rejection of both calcium and magnesium
- investigating alternative sources and methods for the supply of ferric iron to the process, which is required for potassium removal by jarosite precipitation
- reviewing different solid/liquid separation methods used in the base metal precipitation area targeting reduced solids concentration and filtration residence time after precipitation to reduce precipitate re-dissolution
- reviewing the crushing circuit equipment selection, with consideration given to two-stage crushing as opposed to three-stage crushing

It is recommended that Giyani undertake more extensive metallurgical variability sampling in line with the mine plan and test work completed on the samples to confirm:

- material extraction, solution purification requirements, and reagent consumption data
- further detailed engineering studies in relation to mill design, bulk materials handling requirements, slurry rheology, dissolution, and precipitation performance

Samples and sampling requirements for metallurgical test work will need to be identified and incorporated into future drilling and exploration programmes. A comprehensive metallurgical testing scheme designed to evaluate grinding requirements, alternative precipitation chemistries, solid/liquid separation requirements, and variability testing can be expected to cost between US$250,000 and US$300,000.

In addition, it is recommended that Giyani investigate QA/QC requirements for the supply of consumables and reagents to allow the characterisation of these materials; this includes, but is not limited to, grinding media, lime, and sulphur. All process inputs are potential sources of impurity that can adversely affect production of battery-grade HPMSM.

Further, it is recommended that Giyani evaluate production of agricultural-grade manganese sulphate as a potential outlet for off-specification material set aside for repurification or as a potential source of high-grade manganese in the event of major process interruptions.

26.3 Mining methods

Key mining-related recommendations include the following:

- The Mineral Resource model should be updated in the next stage of study with further in-fill drilling to improve the confidence in the block model estimates and exclude Inferred material from the evaluation of the Mineral Reserve. The proportion of the Measured to Indicated category of material mined in the early years should be maximised to reduce Project risk. This cost is included in Table 26.2.
- The potential for reducing waste stripping requirements by steepening the open pit slopes should be evaluated through a comprehensive geotechnical analysis of both the North and South open pit areas. To date the geotechnical analysis has only been carried out on the North pit and these recommendations have been applied to the Southern Extension pit. A dedicated programme of geotechnical drilling needs to be carried out to confirm the slope parameters for all slopes included in the open pit design.
- Using the information gained from the proposed geotechnical drilling programme to define the strength properties, the geotechnical parameters for the WRD and the low-grade stockpile also need to be confirmed.
- The estimation of the modifying factors for material loss and dilution through mining should be explicitly modelled with techniques such as skin analysis of block regularisation. This study will help to establish the optimal bench/flitch configuration, in conjunction with the equipment selection, to maximise the Project value.
- The open pit optimisations should be rerun when additional operating cost and/or geological information is available to establish the optimal pit limit to be used for pit design, ensuring it is appropriate and aligned with the project objectives and risk strategy. The recovery and process cost relationships used in the optimisation should be informed by a geo-metallurgical model.
- The mine plan should consider the option to ramp-up production. Alternative optimisation techniques, such as direct block scheduling, provide an opportunity to optimise the pit limit and mining sequence in a single pass. This overcomes many of the limitations implicit in the standard Lerch-Grossman methodology and can add significant value when there are complex objective functions and/or constraints that vary with time.

- The ramp width should be reviewed prior to the detailed engineering stage of the Project, and while considering the final decision on mining fleet selection. The open pit slope deign angles should be reviewed based on the improved estimates of geotechnical strength parameters for the surrounding rock.
- Detailed pit stage designs should be created to identify the haul routes for material and waste so that a haulage simulation can be included in the optimisation and the LOM schedule. This haul simulation needs to be integrated with the WRD planning process.
- The variable cut-off grade strategy, with respect to the objectives of maximising Project value, while managing blending or other constraints, should be further assessed and optimised.
- Trade-off studies should be conducted to optimise the equipment selection. The usage of biodiesel to minimize the Project's carbon footprint should also be taken into consideration. Additionally, electric powered equipment should be considered where practical.
- The geotechnical studies should aim to provide information with respect to digging conditions for the various rock types (weathered, transition, and fresh) so that a trade-off study can be completed to establish the need for blasting versus free dig and/or ripping.
- Technical personnel from the equipment suppliers should be involved with further equipment size assessments as well as preliminary discussions with potential mining contractors (and other service providers) should be used to confirm the planning assumptions.
- A trade-off study should be completed to compare owner mining versus contract mining.

26.4 Geotechnical considerations

A number of limitations of the mining geotechnical assessment have been highlighted. To address some of these limitations and allow assessment at a preliminary feasibility study level, the following work is recommended:

- geotechnical investigation of units in the vicinity of the walls of the proposed open pit; this investigation should include:
  - ✦ inclined and oriented drilling to allow structural orientation measurements (US$225,000)
    - televiewer survey may allow a reduction in DD requirements if drillholes remain open to allow downhole survey
  - ✦ detailed geotechnical rock mass and structural logging (US$20,000).
  - ✦ sample selection to allow laboratory test work, including consideration of consolidated undrained triaxial testing of weak rock units (US$15,000)
- geotechnical analysis incorporating (US$50,000):
  - ✦ results of the investigation and laboratory test work
  - ✦ updated geological and structural models, which extend into the areas of the open pit walls
  - ✦ hydrogeological data relevant to the open pit areas
  - ✦ derivation of representative rock mass and defect plane shear strengths
  - ✦ slope stability analysis of rock mass, structural, and composite failure mechanisms
  - ✦ recommendations on slope angles by open pit area

- an assessment of the excavatability of the main rock mass units.
- consideration for the WRD and long-term stability for closure

26.5 Hydrological considerations

Given the changes associated with the updated MRE and proposed mining extent, the following water-related aspects are recommended in order to take the Project to a feasibility level of confidence.

- It is recommended that Giyani develop a numerical groundwater model in which the groundwater inflows into the open pit, in yearly increments, are simulated. The numerical model can also be used to calculate the zone of influence of the groundwater level drawdown and the pore pressure in the open pit walls for input into the open pit design. The volume of surface water runoff into the open pit should also be recalculated based on the new mining sequence so that appropriate dewatering strategies can be put in place accompanied by a dewatering management plan (US$20,000 to US$30,000).
- It is recommended that Giyani update the existing mine water balance to ensure that enough water storage capacity is designed and installed at the processing plant site to effectively manage the additional volume of storm water due to the pit extension and to account for the groundwater inflows into the open pit (US$5,000 to US$10,000).
- It is recommended that the updated water balance re-evaluate the processing plant water demand, based on the updated MRE, to ensure that the identified water supply options will be sufficient to meet the mining and processing plant water demands (budget included in water balance update).
- Taking into consideration the updated MRE and larger mining footprint, it is recommended that the runoff volumes for each of the open pit, WRD, and TMF areas be recalculated and that, if necessary, the conceptual channel designs be updated in terms of channel locations and sizes (US$15,000 to US$25,000)

26.6 Project infrastructure

Tetra Tech makes the following recommendations for additional investigations at the next stage of the study:

- mine geotechnical assessment
- infrastructure geotechnical assessment for the site infrastructure, buildings, TMF, and WRD areas
- mineralised material and waste geochemistry testing and acid rock drainage test work
- a value-engineering study, evaluating such things as:
    - tailings disposal philosophy, whether it would be more economical to slurrify the tailings and pump to the TMF or as currently proposed, dry the tailings and truck
    - designing a dump pocket and ROM bin as opposed to the current proposal of an in-ground ROM feeder
- confirm a policy with regard to employee housing and transport requirements

26.7 Tailings management facility

The TMF presented in this study has been designed based on information known at the time of the PEA. Knight Piésold recommends a number of actions to optimise the design prior to Project execution and during detailed engineering.

- Ground investigation and materials testing:
  - The design presented in the PEA includes a basal lining system to prevent seepage of contact water from the tailings to the surrounding environment and groundwater. Geochemical testing of tailings should be undertaken to determine potential contaminants and evaluate other possible solutions to prevent seepage to the environment.
  - Further materials test work should be completed on representative tailings samples generated from the operation of Giyani's demonstration plant. This should be used to inform the in-situ density for updated capacity analysis and strength parameters for updated stability analysis.
  - Direct shear testing of the geomembrane with sand bedding material on one side and tailings on the other should be performed prior to detailed design.
  - Undertake ground investigations with piezometer installations, sampling and monitoring at revised TMF locations to determine geotechnical and groundwater conditions. In-situ and laboratory testing should be conducted to provide reliable material parameters for encountered strata. Stability analyses to design the TMF to meet CDA/GISTM guidance should be conducted.
  - Ground investigations, sampling, and laboratory testing at proposed fill source locations should be conducted to determine material parameters.

A cost of approximately US$250,000 should be allowed for the above work to be completed in the next stage of the Project.

- Detailed engineering:
  - Carry out a site-specific seismic assessment.
  - Carry out a tailings dam breach assessment.
  - Conduct sequencing analysis to optimise waste rock placement and backfilling.
  - Develop the tailings transport and deposition strategy.
  - Develop design for the tailings facility with external loading design criteria consistent with the GISTM consequence of failure classification selected based on current conditions.
  - Undertake hydrogeological assessment to consider the need and performance requirements of a basal lining system.
  - Knight Piésold recommends that piezometers are specified in the detailed design phase so that water-pore pressure distribution within the tailings and embankment can be understood during the operational phase.
  - A weather station should be installed to collect data including evaporation, rainfall, temperature, and humidity prior to construction to provide some site-specific data.
  - Review climate change assessment and establish if a new assessment is required.

A cost of approximately US$230,000 should be allowed for detailed engineering on the TMF.

26.8 Environmental studies, permitting, and social or community impact

The current scope of the EIA is based on an 11-year LOM. It is recommended that Giyani obtain approvals for the extension of the LOM.

26.9 Geochemical assessment

A geochemical assessment of the material that will be mined, processed, and stored must be conducted in order to determine the potential for producing acid rock drainage, as well as the long-term quality of leachate emanating from the different stockpiles and facilities. The material assessed should include the topsoil, waste rock, ROM stockpile, and tailings material. Sediment samples from nearby streams can also be collected to assist in the impact assessment on the streams, if applicable.

- initial round of analysis (acid-base accounting, mineralogical and static leach testing) for all samples: US$650/sample, 12 total samples (4 × topsoil, 4 × waste rock, 2 × ROM, 2 × TMF), for a total budget of US$7,800
- kinetic leach testing (24 weeks for 12 samples): US$15,000
- geochemical modelling to simulate long-term evolvement of leach qualities: US$10,000

26.9.1 Sampling procedure

Topsoil stockpiles

Composite samples of the topsoil will be collected from the study area as part of the soil investigation. Soil samples should be collected from an undisturbed untouched soil surface at a desired depth. At the desired sampling depth, use the digging device to remove enough volume of soil to fill the required sample container(s), or if the soil density allows, push a stainless steel or brass sleeve directly into the soil and remove. Samples are then collected in the required containers outlined by the laboratory.

Waste rock and run-of-mine stockpiles

Waste rock (overburden) and mineralized material must be sampled from exploration drill logs. It is proposed that composite samples be collected from different exploration boreholes spread over the project area. Specific sampling positions (borehole and specific depth) should be selected based on the available information (exploration drilling geological logs and ore body intersections). Similar samples, in term of geology, state of weathering etc., are grouped together to form composite samples which are representative of the geology in the study area. Boreholes are selected based on a number of parameters, including:

- Position in relation to the overall mine layout: focus is placed on exploration boreholes that intersected the waste rock (overburden) as well as the orebody. Boreholes are located within the proposed mining areas.
- Lithologies intercepted by the boreholes: different lithologies can have different geochemical signatures and leach qualities. Boreholes are selected based on the lithologies intercepted to ensure that all the different lithologies will be mined, processed, and stored on site will be represented in the samples.

Tailings material

Tailings material will be obtained from metallurgical bench or pilot testing. Tailings material is typically relatively homogenous; therefore, it is proposed that only two samples be provided from the pilot testing.

The water fraction of the tailings material will also be sampled and analysed to determine the expected quality of the water that will be deposited on the TMF together with the tailings material.

River sediments

River sediments can be collected from the nearby streams to assist in the stream impact assessment. Samples will be collected from strategically important areas, such as where poor quality groundwater is expected, that could change the pH and metals concentrations could enter the streams.

26.9.2 Analyses

Geochemical analyses that are required include acid-base accounting testing, mineralogical assessment, and static and (potentially) kinetic leach testing.

Acid-base accounting testing

The acid-base accounting testing that should be performed is summarised in Table 26.3.

Table 26.3 Acid-base accounting testing

Test	Unit
Paste pH	-
Sulphur speciation (total, sulphide, and sulphate)	wt%
NAG pH; NAG pH4.5; NAG pH pH7	kg H_2SO_4/t
Acid potential: total sulphur and sulphide sulphur	kg $CaCO_3$/t
Neutralisation potential: total sulphur and sulphide sulphur	kg $CaCO_3$/t
Net neutralisation potential: sulphide sulphur	kg $CaCO_3$/t
Neutralising potential ratio: sulphide sulphur	-

Static and kinetic leach testing

Distilled leach testing (1:4 solid to fluid ratio) should be performed on each of the samples. Testing should start with a once-off static leach test. During later phases of the Project, where long-term evolvement of the chemical characteristics of leachate emanating from any stockpiles or the TMF during the operational and post-closure phases of the operations is needed to a greater level of certainty, kinetic testing can be done. It is currently expected that the kinetic leach testing will span around 24 weeks before completion. The results should be compared to the IFC effluent discharge standards. Elements to be analysed are summarised in Table 26.4.

Table 26.4 Static and kinetic leach testing elements to be analysed

Analyte	Unit	Analyte	Unit	Analyte	Unit
pH		NO_3	mg/L	Mn	mg/L
Na	mg/L	Al	mg/L	Mo	mg/L
K	mg/L	As	mg/L	Ni	mg/L
Ca	mg/L	Cd	mg/L	Pb	mg/L
Mg	mg/L	Co	mg/L	Sb	mg/L

Analyte	Unit	Analyte	Unit	Analyte	Unit
NH_3	mg/L	Cu	mg/L	Se	mg/L
Cl	mg/L	F	mg/L	V	mg/L
SO_4	mg/L	Fe	mg/L	Zn	mg/L
T-Alkalinity	mg/L	Hg	mg/L		

Mineralogical assessment

Mineralogical assessment of the samples by x-ray diffraction must include the minerals shown in Table 26.5.

Table 26.5 Mineral assessment by x-ray diffraction

Mineral	Unit	Mineral	Unit
Quarts	wt%	Hematite	wt%
Plagioclase	wt%	Gypsum	wt%
Actinolite	wt%	Kaolinite	wt%
Epidote	wt%	Muscovite	wt%
Pyrite	wt%	Chlorite	wt%
Pyrrhotite	wt%	Smectite	wt%
Barite	wt%		wt%

Particle size distribution

Particle size distribution testing must be performed on the tailings material only.

26.10 Capital and operating costs

CSA Global recommends that Giyani investigate the true cost of mining contracting especially around mobilization and demobilization over the life of the project. CSA Global expects that there will be multiple mobilizations and demobilizations given the length of mining contracts in the industry which are generally around 5 years, and the expected life of the Project.

26.11 Economic analysis

Based on CSA Global's observation of the work and estimates of costs carried out to date, CSA Global recommends the following:

- Further detailed technical and economic analyses is warranted to increase the degree of accuracy in the estimates and the economics of the Project.
- Further work on capital and operating cost estimation will be required for the next level of study.
- Investigate the implementation of the upside case.

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Qualified Person Certificate - EUR ING Andrew Carter, BSc, CEng, MIMMM QMR, MSAIM, SME

I, EUR ING Andrew Carter, BSc, CEng, MIMMM QMR, MSAIM, SME do herby certify:

- I am head of Mining Services for Tetra Tech Europe, with a business address at 3 Sovereign Square, Sovereign Street, Leeds LS1 4ER, United Kingdom.
- This certificate applies to the technical report entitled *K.Hill Battery-Grade Manganese Project Preliminary Economic Assessment, National Instrument 43-101 Technical Report* with an effective date of 31st July 2023 (the Technical Report).
- I am a graduate of the University of Leeds (BSc Mineral Processing, 1980). I am a member in good standing of the Institution of Mining, Metallurgy and Materials (#46421) Qualified for Minerals Reporting, the Society for Mining, Metallurgy and Exploration (#4112502) and a Member of the South African Institute of Mining and Metallurgy (#0464339). I am registered as a Chartered Engineer with the Engineering Council UK (#378467) and as a European Engineer with the European Federation of National Engineering Associations (#c2960GB). I have practiced my profession continuously since 1980. My relevant experience comprises over 40 years in operations, engineering, and consulting practice in relation to the extractive metallurgy of gold, precious, base, and ferrous metal ores. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the Instrument).
- I have not completed a personal inspection of the K.Hill Project in Botswana.
- I am responsible for Sections 1.8, 1.11, 1.12, 1.18.2, 1.18.4, 12.3, 12.7, 13, 17, 18.1 to 18.7, 21.2.3, 25.2, 25.6, 26.2, 26.6, and 27 of the Technical Report.
- I am independent of Giyani Metals Corp., as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of this Technical Report.
- I have read the Instrument, and the sections of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.
- As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 25th day of August 2023.

"signed and dated"

EUR ING Andrew Carter, BSc, CEng, MIMMM QMR, MSAIM, SME
Head of Mining UK and Ireland
Tetra Tech Europe

Qualified Person Certificate - Richard Elmer, BSc, MSc, CEng, MIMMM, MCSM

I, Richard Elmer, BSc, MSc, CEng, MIMMM, MCSM do herby certify:

- I am a Principal Geotechnical Engineer and Director with Knight Piésold Limited, with an office address at St Magnus House, 3 Lower Thames Street, London, EC3R 6HD, United Kingdom.
- This certificate applies to the technical report entitled *K.Hill Battery-Grade Manganese Project Preliminary Economic Assessment, National Instrument 43-101 Technical Report* with an effective date of 31st July 2023 (the Technical Report).
- I am a graduate of the University of Southampton (BSc Geology) and the Camborne School of Mines (MSc Mining Geology). I am a Member of the Institute of Materials, Minerals & Mining (membership no. 0049205). I have practiced my profession for 34 years. I am a geotechnical engineer that has been directly involved in feasibility studies and detailed designs and construction supervision for mine waste facilities (tailings facilities and waste rock facilities) around the world, for multiple commodities, as well as completing audits on operational facilities.
- I am a "Qualified Person" for the purposes of National Instrument 43-101 (the Instrument).
- I have not completed a personal inspection of the K.Hill Project in Botswana.
- I am responsible for Sections 1.13, 1.18.5, 12.4, 12.7, 18.8, 21.2.4, 25.7, 26.7, and 27 of the Technical Report.
- I am independent of Giyani Metals Corp., as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of this Technical Report.
- I have read the Instrument, and the sections of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.
- As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 25th day of August 2023.

"signed and dated"

Richard Elmer, BSc, MSc, CEng, MIMMM, MCSM
Principal Geotechnical Engineer/Director
Knight Piésold Limited

Qualified Person Certificate - Anton Geldenhuys, MEng, FGSSA, PrSciNat

I, Anton Geldenhuys, MEng, FGSSA, PrSciNat do herby certify:

- I am a Principal Consultant with CSA Global South Africa (Pty) Ltd., at Woodlands Office Park, Woodlands, Sandton, Gauteng, 2148, South Africa.
- This certificate applies to the technical report entitled *K.Hill Battery-Grade Manganese Project Preliminary Economic Assessment, National Instrument 43-101 Technical Report* with an effective date of 31st July 2023 (the Technical Report).
- I am a graduate of Rand Afrikaans University (South Africa) (BSc [Hons]) and the University of Witwatersrand (South Africa) (MEng). I am a Fellow in good standing of the Geological Society of South Africa and a registered Professional Natural Scientist (PrSciNat) with the South African Council for Natural Scientific Professions (membership number 400313/04). My experience includes 22 continuous years in the exploration and mining industry. I am familiar with National Instrument 43-101 and, by reason of education, experience in exploration, mineral resource development and the evaluation of mining projects, and professional registration.
- I am a "Qualified Person" for the purposes of National Instrument 43-101 (the Instrument).
- My most recent personal inspection of the K.Hill Project in Botswana was during the period from 20th to 21st April 2023.
- I am responsible for Sections 1.2 to 1.7, 1.9, 1.18.1, 2, 3.1, 4 to 11, 12.1, 12.7, 14, 23, 25.1, 26.1, and 27 of the Technical Report.
- I am independent of Giyani Metals Corp., as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of this Technical Report.
- I have read the Instrument, and the sections of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.
- As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 25th day of August 2023.

"signed and dated"

Anton Geldenhuys, MEng, FGSSA, PrSciNat
Principal Consultant
CSA Global South Africa (Pty) Ltd.

CSA Global
Mining Industry Consultants
an ERM Group company

Qualified Person Certificate - Martiens Prinsloo, MSc, MBA, PrSciNat

I, Martiens Prinsloo, MSc, MBA, PrSciNat do herby certify:

- I am a Technical Director with ERM with a business address at Building 27, The Woodlands Office Park, Woodlands Drive, Woodmead, Gauteng, 2148, South Africa. I am subcontracted through CSA Global South Africa (Pty) Ltd. for the hydrological work on this Technical Report.
- This certificate applies to the technical report entitled *K.Hill Battery-Grade Manganese Project Preliminary Economic Assessment, National Instrument 43-101 Technical Report* with an effective date of 31st July 2023 (the Technical Report).
- I am a graduate of The University of Pretoria (B.Sc. Earth Sciences); The University of the Free State (M.Sc. Geohydrology) and the University of Cape Town (MBA). I am a registered Natural Scientific Professional with the South African Council for Natural Scientific Professions (SACNASP) (membership number400248/04. My experience includes 25 years' worth of consulting to the mining industry. I have successfully delivered more than 200 individual projects across Africa, parts of Europe, South America and Australia. I have experience in all stage of a project, from PEA through feasibility, operational and post-closure. My main fields of focus include hydrogeology, geochemistry and hydrology.
- I am a "Qualified Person" for the purposes of National Instrument 43-101 (the Instrument).
- I have not completed a personal inspection of the K.Hill Project in Botswana.
- I am responsible for Sections 12.6, 12.7, 16.3, 18.9, 25.5, 26.5, 26.9, and 27 of the Technical Report.
- I am independent of Giyani Metals Corp., as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of this Technical Report.
- I have read the Instrument, and the sections of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.
- As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 25th day of August 2023.

"signed and dated"

Martiens Prinsloo, MSc, MBA, PrSciNat
Technical Director
ERM

Qualified Person Certificate - Matthew Randall, BSc (Hons), PhD, MIMMM, CEng

I, Matthew Randall, BSc (Hons), PhD, MIMMM, CEng do herby certify:

- I am a Certified Mining Engineer with Axe Valley Mining Consultants Ltd., with a business address of 138 High Street, Swanage, Dorset BH19 2PA, United Kingdom.
- This certificate applies to the technical report entitled *K.Hill Battery-Grade Manganese Project Preliminary Economic Assessment, National Instrument 43-101 Technical Report* with an effective date of 31st July 2023 (the Technical Report).
- I am a graduate the Camborne School of Mines, Cornwall, UK (BSc [Hons] Mining). I am a member in good standing of Institute of Materials, Metallurgy and Mining (member no. 458442). I have practiced my profession for more than 35 years.
- I am a "Qualified Person" for the purposes of National Instrument 43-101 (the Instrument).
- I have not completed a personal inspection of the K.Hill Project in Botswana.
- I am responsible for Sections 1.10, 1.18.3, 12.2, 12.7, 15, 16.1, 16.2, 16.5 to 16.10, 24.1, 21.2.2, 25.3, 26.3, and 27 of the Technical Report.
- I am independent of Giyani Metals Corp., as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of this Technical Report.
- I have read the Instrument, and the sections of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.
- As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 25th day of August 2023.

"signed and dated"

Matthew Randall, BSc (Hons), PhD, MIMMM, CEng
Certified Mining Engineer
Axe Valley Mining Consultants Ltd.

Qualified Person Certificate - Howard Simpson, BSc Eng (Hons), BCom, FAusIMM (CP Mining), RPEQ

I, Howard Simpson, BSc Eng (Hons), BCom, FAusIMM (CP Mining), RPEQ do hereby certify:

- I am a Mining Engineer with CSA Global South Africa (Pty) Ltd., at 260 Queens Street, Brisbane, 4000, Queensland, Australia.
- This certificate applies to the technical report entitled *K.Hill Battery-Grade Manganese Project Preliminary Economic Assessment, National Instrument 43-101 Technical Report*, effective 31st July 2023 (the Technical Report).
- I am a graduate of the University of Cardiff (BSc (Hons) Mining Engineering) the University of South Africa (BCom Accounting and Quantitative Management). I am registered fellow with the Australasian Institute of Mining and Metallurgy (FAusIMM; number 326398). I am an experienced mining professional with over 30 years of experience and has delivered mining engineering, mine planning and economic evaluation for projects, technical studies, and operations. He has delivered projects and studies across multiple geographies and commodities, with responsibilities for design, planning, scheduling of mine operations, and economic evaluation.
- I am a "Qualified Person" for the purposes of National Instrument 43-101 (the Instrument).
- I have not completed a personal inspection of the K.Hill Project in Botswana.
- I am responsible for Sections 1.16, 1.17, 1.18.7, 1.18.8, 2, 21.1, 21.2.1, 21.2.5, 22, 24.2, 25.9, 25.10, 26.10, 26.11, and 27 of the Technical Report.
- I am independent of Giyani Metals Corp., as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of this Technical Report.
- I have read the Instrument, and the sections of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.
- As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 25th day of August 2023.

"signed and dated"

Howard Simpson, BSc Eng (Hons), BCom, FAusIMM (CP Mining), RPEQ
Mining Engineer
CSA Global South Africa (Pty) Ltd.

Qualified Person Certificate - Sifiso Siwela, PrSciNat, FGSSA, MSAIMM, MSEG

I, Sifiso Siwela, PrSciNat, FGSSA, MSAIMM do herby certify:

- I am a Principal Consultant and Manager - Africa with CSA Global South Africa (Pty) Ltd., at Building 27, Woodlands Office Park, Woodlands, Sandton, Gauteng, 2148, South Africa.
- This certificate applies to the technical report entitled *K.Hill Battery-Grade Manganese Project Preliminary Economic Assessment, National Instrument 43-101 Technical Report* with an effective date of 31st July 2023 (the Technical Report).
- I am a graduate of BSc Geology (Hons) from university of Johannesburg in 2013 and a Graduate Diploma in Engineering (GDE) in Geostatistics/Mineral Resource Evaluation from University of Witwatersrand in 2013. I am a registered Professional Natural Scientist (PrSciNat) with the South African Council for Natural Scientific Professions (membership number 400124/10). My experience includes 18 continuous years in the exploration and mineral resource development and the evaluation of mining projects.
- I am a "Qualified Person" for the purposes of National Instrument 43-101 (the Instrument).
- I have not completed a personal inspection of the K.Hill Project in Botswana.
- I am responsible for Sections 1.1, 1.14, 1.15, 1.18.6, 1.19, 3.2, 3.3, 19, 20, 25.8, 26.8, and 27 of the Technical Report.
- I am independent of Giyani Metals Corp., as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of this Technical Report.
- I have read the Instrument, and the sections of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.
- As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 25th day of August 2023.

"signed and dated"

Sifiso Siwela, PrSciNat, FGSSA, MSAIMM, MSEG
Principal Consultant and Manager - Africa
CSA Global South Africa (Pty) Ltd.

Qualified Person Certificate - Rob Thomas, MSci ARSM, FAusIMM(CP), CEng, CSci, REnvP, MIMMM

I, Rob Thomas, MSci ARSM, FAusIMM(CP), CEng, CSci, REnvP, MIMMM do herby certify:

- I am a Principal Consultant associate of CSA Global South Africa (Pty) Ltd., at Woodlands Office Park, Woodlands, Sandton, Gauteng, 2148, South Africa.
- This certificate applies to the technical report entitled *K.Hill Battery-Grade Manganese Project Preliminary Economic Assessment, National Instrument 43-101 Technical Report* with an effective date of 31st July 2023 (the Technical Report).
- I am a graduate of Imperial College London (MSci) and an Associate of the Royal School of Mines (ARSM). I am a Fellow and Chartered Professional of the Australasian Institute of Mining and Metallurgy (membership number 309377) and a Professional Member, Chartered Engineer and Chartered Scientist of the Institute of Materials, Minerals and Mining (membership number 479469). My experience includes over 15 years undertaking, managing and delivering geotechnical assessments within the resource sector.
- I am a “Qualified Person” for the purposes of National Instrument 43-101 (the Instrument).
- I have not completed a personal inspection of the K.Hill Project in Botswana.
- I am responsible for Sections 12.5, 12.7, 16.4, 25.4, 26.4, and 27 of the Technical Report.
- I am independent of Giyani Metals Corp., as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of this Technical Report.
- I have read the Instrument, and the sections of the Technical Report for which I am responsible have been prepared in compliance with the Instrument.
- As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 25th day of August 2023.

“signed and dated”

Rob Thomas, MSci ARSM, FAusIMM(CP), CEng, CSci, REnvP, MIMMM
Principal Consultant (Associate)
CSA Global South Africa (Pty) Ltd.